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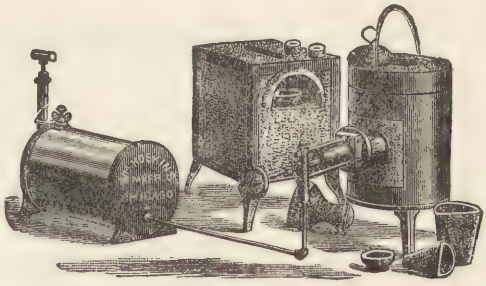
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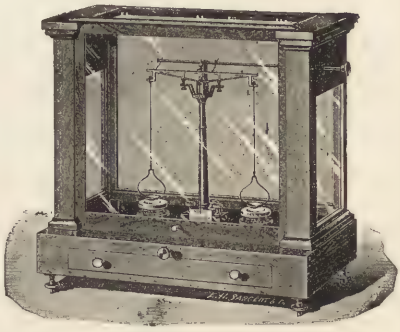
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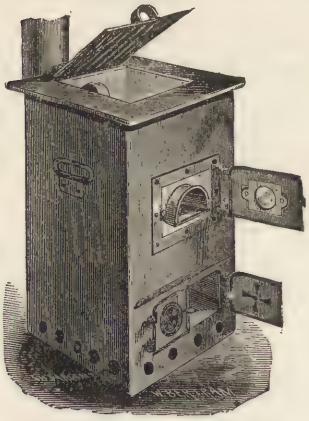


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ITS

STATISTICS, TECHNOLOGY AND TRADE,

IN THE

UNITED STATES AND OTHER COUNTRIES

TO THE END OF

1896.

VOL. V.

EDITED BY

RICHARD P. ROTHWELL,

Editor of the Engineering and Mining Journal; Ex-President American Institute of Mining Engineers; Member American Society of Civil Engineers; Fellow Royal Statistical Society, London, Etc., Etc., Etc.

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PREFACE.

In presenting the fifth volume of this work we gratefully acknowledge the enthusiastic reception which the previous volumes have met with and the many testimonials of warm approval and commendation which have been accorded to them. The following extracts from the preface to Volume I. outline the objects in view when this stupendous undertaking was inaugurated, and the pages of the volumes themselves testify how far these aims have already been attained:

"This volume is a result of the development of the annual statistical numbers of the *Engineering and Mining Journal*, and owes its existence to the appreciation with which these statistics have been received by business men, by experts, and by others interested in the mineral industry throughout the world.

"The modern newspaper has made promptness in furnishing information not only familiar, but indispensable to the man of affairs, and accurate and timely statistics have now become absolutely necessary for the intelligent direction of industry, trade, and legislation. The collection of such statistics in an industry which extends over the face of the entire globe is, however, a work so vast and difficult that it has hitherto been considered impossible except through the unlimited resources of governments; and as the machinery of government is not adapted to the rapid attainment of results, the statistics of the mineral industry have been so tardily collected and published in all countries that their value has been greatly impaired. . . . For many years the *Engineering and Mining Journal*, as the leading representative of this great industry, has accumulated vast stores of statistical information relating to it and has greatly improved the machinery for the collection of these statistics. . . .

"The universal appreciation of the work done by the *Engineering and Mining Journal* called for its extension, and consequently in this initial volume there are given, for the first time, the statistics of substantially all the minerals and metals produced in the United States and in many other countries for the full year 1892, and often from the earliest times. This series of annual volumes it is intended shall, in due time, cover the entire mineral industry of the world, giving its statistics, its technology, and its trade, each succeeding volume not repeating

the data given in previous issues, but supplementing them, and carrying forward the current history of the industry almost to the day of publication. Unaided by any governmental powers to enforce the making of returns, we have relied for success solely upon personal courtesy and confidence and upon the intelligent appreciation of the value of the work to the industry at large, and this great volume is the monument we have erected to the courtesy of those whose prompt and willing co-operation has alone rendered its success possible. Long experience in this kind of work has fully demonstrated the fact that men are in general more willing to give important and correct information to the private individual who can be held responsible for its proper use than to the more or less impersonal 'government.' It is indeed extremely rare that any producer neglects or refuses to give full, truthful, and satisfactory replies to our requests for information. . . .

"Accuracy should always be the first care of the statistician, but it is scarcely less important to the business man that the information should be promptly furnished. Belated statistics are ancient history, of little practical value in the active affairs of an industry, or as a guide for legislation affecting it. . . . Subjects which are uppermost in the business world naturally demand the greatest attention in this industrial work. . . .

"Much attention has been devoted to the subject of cost of production. The itemization of cost is the first essential step in securing economy in producing any article, and the history of every country and of every industry has shown that prosperity, whether national, industrial, or individual, is, in a general way, inversely proportional to the cost of supplying the rest of the world with what one produces. The great economies which command the markets for products are due, not to reductions in wages, but to increased knowledge and intelligence, and are accompanied by higher remuneration and a betterment of the condition of those engaged in the industry. . . .

"It is the object of the *Engineering and Mining Journal* to give in detail, and of this supplemental volume of the *Journal* to summarize, the facts which show how such results are accomplished; to photograph, as it were, from time to time, the condition of the several departments of the mineral industry in various parts of the world, placing within the reach of all the information that intelligence can apply to the reduction of cost in producing and marketing the useful minerals and metals and in promoting the welfare of those engaged in this industry. In every country this information will enable those who legislate for and those who administer this industry to do so with an intelligent appreciation of the conditions affecting it in its every department, and, widely disseminated, will promote the national prosperity.

"It is with the very greatest regret that we have been obliged in this work to use other than the metric system of weights and measures, which are now legalized in nearly every civilized country and should be universally adopted. The necessity of conforming to custom and popular prejudice in a work so expensive as this explains the use here of that nightmare of weights and measures which, as a relic of barbarism, survives and is used in all English-speaking countries as 'the English system.' We have, however, where possible, reduced the number of varieties of measures as used in the publications of the United States Govern-

ment. All foreign statistics are given in this work in metric weights, and the United States products are given in the metric system as well as in the customary weights.

"The advertising pages of this book will well repay the careful perusal and study of every reader who wishes to be well informed upon the present condition of the mineral industry. They give an admirable and practical insight into the present state of the mining and metallurgical arts, for in them nearly every manufacturer or dealer of note in this country advertises the machines, appliances, and processes which are now in vogue, or which it is sought to introduce, while the names and qualifications of the most eminent members of the engineering professions indicate the importance and directions of modern mining and metallurgy. These advertising pages are no less important to those who desire a clear knowledge of the means by which this country has come to be far the most important producer of minerals and metals, than to him who wishes to know where to get that full and reliable information concerning the values of properties, machinery, processes, and products which should precede the investment of capital."

Had any doubt existed as to the utility and even absolute need of such a work as this, it would have been quickly dispelled by the prompt and enthusiastic recognition of its value and the unstinted and unanimous words of praise which welcomed in every part of the world the first volume of THE MINERAL INDUSTRY, ITS STATISTICS, TECHNOLOGY AND TRADE. It is not too much to say that probably no other technical book was ever so universally praised, as is shown in the brief extracts given on another page from some of the thousands of letters received, and doubtless never before was a technical book sold in so large numbers within so short a time. These results are extremely gratifying not alone from a business standpoint, but chiefly because they constitute a flattering evidence of the value of the work as a whole, and of the ability of the many specialists who contributed to its pages.

This success has increased the obligation we are under to the great captains of the industry throughout the world who have, with uniform courtesy and unlimited confidence, furnished the data which alone have permitted the compilation and prompt publication of these valuable statistical volumes.

From the very first the amount of useful material grew so rapidly with the plans for the work that it quickly became evident that the book must become larger and more costly than anticipated. The original object has, however, been kept in view of placing it within the reach of all.

The importance of collecting in this volume, as far as possible, the existing statistics of production, imports, exports, and consumption of the various minerals and metals in every country, and the large space devoted to this and to the description of the best existing practice in the chief departments of metallurgy and chemical technology, have crowded over into the current issues of the *Engineering and Mining Journal* and into succeeding volumes of this book much valuable information, and retarded the issue of this book beyond the date at which it is expected future volumes will appear.

The data collected in the five volumes present already a good foundation on which to commence an intelligent study of the industry and of the conditions

which affect it, and we earnestly invite the co-operation in this work of all who possess further facts which may be of use in it.

This introduction may be appropriately ended with the following closing paragraph of the preface to the first volume:

"No one can appreciate more fully than the statistician himself the limits and shortcomings which are inseparable from all statistical work of this character. As further facts come to light and a higher degree of accuracy rewards our continuous efforts to render these volumes trustworthy, corrections will be made in the statistics should errors of importance be found. He is but a dishonest statistician who retains known important erroneous statements in order that the public, in its ignorance, may believe, from the absence of corrections, that his figures are accurate. Readers of this book are therefore earnestly requested to notify its editor of any errors or omissions which may be found in it, in order that corrections may be made in subsequent volumes, and suggestions which may render future volumes more valuable will be gratefully received."

RICHARD P. ROTHWELL.

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Linear.					Capacity.							
Inches to Millimeters.	Feet to Meters.	Yards to Meters.	Miles to Kilometers.		Drams to Cubic Centimeters.	Ounces to Milliliters.	Quarts to Liters.	Gallons to Liters.	Cubic Inches to Cubic Centimeters.	Cubic Feet to Cubic Meters.	Cubic Yards to Cubic Meters.	Bushels to Hectoliters.
25.4000	0.304801	0.914402	1.60935	= 1 =	3.70	29.57	0.94636	3.78544	16.387	0.02832	0.765	0.35242
50.8001	0.609601	1.828804	3.21869	= 2 =	7.39	59.15	1.89272	7.57088	32.774	0.05663	1.529	0.70485
76.2001	0.914402	2.743205	4.82804	= 3 =	11.09	88.72	2.83908	11.35632	49.161	0.08495	2.294	1.05727
101.6002	1.219202	3.657607	6.43739	= 4 =	14.79	118.30	3.78544	15.14176	65.549	0.11327	3.058	1.40969
127.0002	1.524003	4.572009	8.04674	= 5 =	18.48	147.87	4.73180	18.92720	81.936	0.14158	3.823	1.76211
152.4003	1.828804	5.486411	9.65608	= 6 =	22.18	177.44	5.67816	22.71264	98.323	0.16990	4.587	2.11454
177.8003	2.133604	6.400813	11.26543	= 7 =	25.88	207.02	6.62452	26.49808	114.710	0.19822	5.352	2.46696
203. .004	2.438405	7.315215	12.87478	= 8 =	29.57	236.59	7.57088	30.28352	131.097	0.22654	6.116	2.81938
228.6004	2.743205	8.229616	14.48412	= 9 =	33.28	266.16	8.51724	34.06896	147.484	0.25485	6.881	3.17181

Square.					Weight.				
Square Inches to Square Centimeters.	Square Feet to Square Decimeters.	Square Yards to Square Meters.	Acres to Hectaers.		Grains to Milligrams.	Avoirdupois Ounces to Grams.	Avoirdupois Pounds to Kilograms.	Troy Ounces to Grams.
6.452	9.290	0.836	0.4047	= 1 =	64.7989	28.3495	0.45359	31.10348
12.903	18.581	1.672	0.8094	= 2 =	129.5978	56.6991	0.90719	62.20696
19.355	27.871	2.508	1.2141	= 3 =	194.3968	85.0486	1.36078	93.3.044
25.807	37.161	3.344	1.6187	= 4 =	259.1957	113.3981	1.81437	124.41392
32.258	46.452	4.181	2.0234	= 5 =	323.9946	141.7476	2.26796	155.51740
38.710	55.742	5.017	2.4281	= 6 =	388.7935	170.0972	2.72156	186.62089
45.161	65.032	5.853	2.8328	= 7 =	453.5924	198.4467	3.17515	217.72437
51.613	74.323	6.689	3.2375	= 8 =	518.3914	226.7962	3.62874	248.82785
58.065	83.613	7.525	3.6422	= 9 =	583.1903	255.1457	4.08233	279.93133

1 chain	=	20.1169 meters.
1 square mile	=	259 hectaers.
1 fathom	=	1.829 meters.
1 nautical mile	=	1853.27 meters.
1 foot=0.304801 meter,		9.4840158 log.
1 avoir. pound	=	453.5924277 gram.
15432.35639 grains	=	1 kilogram.

TABLES FOR CONVERTING METRIC TO UNITED STATES WEIGHTS AND MEASURES.

Linear.					Capacity.							
Meters to Inches.	Meters to Feet.	Meters to Yards.	Kilometers to Miles.		Milliliters or Cubic Centiliters to Fluid Drams.	Centiliters to Fluid Ounces.	Liters to Quarts.	Dekaliters to Gallons.	Hektoliters to Bushels.	Cubic Centimeters to Cubic Inches.	Cubic Meters to Cubic Feet.	Cubic Meters to Cubic Yards.
39.3700	3.28083	1.093611	0.62137	= 1 =	0.27	0.338	1.0567	2.6117	2.8375	0.0610	35.314	1.308
78.7400	6.56167	2.187222	1.24274	= 2 =	0.54	0.676	2.1134	5.2834	5.6750	0.1220	70.629	2.616
118.1100	9.84250	3.280833	1.86411	= 3 =	0.81	1.014	3.1700	7.9251	8.5125	0.1831	105.943	3.924
157.4800	13.12333	4.374444	2.48548	= 4 =	1.08	1.352	4.2267	10.5668	11.3500	0.2441	141.258	5.232
196.8500	16.40417	5.468056	3.10685	= 5 =	1.35	1.691	5.2834	13.2085	14.1875	0.3051	176.572	6.540
236.2200	19.68500	6.561667	3.72822	= 6 =	1.62	2.029	6.3401	15.8502	17.0250	0.3661	211.887	7.848
275.5900	22.96583	7.655278	4.34959	= 7 =	1.89	2.368	7.3968	18.4919	19.8625	0.4272	247.201	9.156
314.9600	26.24667	8.748889	4.97096	= 8 =	2.16	2.706	8.4534	21.1336	22.7000	0.4882	282.516	10.464
354.3300	29.52750	9.842500	5.59233	= 9 =	2.43	3.043	9.5101	23.7753	25.5375	0.5492	317.830	11.771

Square.					Weight.			
Square Centimeters to Square Inches.	Square Meters to Square Feet.	Square Meters to Square Yards.	Hectares to Acres.		Kilograms to Grains.	Hectograms to Ounces Avoirdupois.	Kilograms to Pounds Avoirdupois.	Grams to Ounces Troy.
0.1550	10.764	1.196	2.471	= 1 =	15432.36	3.5274	2.20462	0.03215
0.3100	21.528	2.392	4.942	= 2 =	30864.71	7.0548	4.40924	0.06430
0.4650	32.292	3.588	7.413	= 3 =	46297.07	10.5822	6.61386	0.09645
0.6200	43.055	4.784	9.884	= 4 =	61729.43	14.1096	8.81849	0.12860
0.7750	53.819	5.980	12.355	= 5 =	77161.78	17.6370	11.02311	0.16075
0.9350	64.583	7.176	14.826	= 6 =	92594.14	21.1644	13.22773	0.19290
1.0850	75.347	8.372	17.297	= 7 =	108026.49	24.6918	15.43235	0.22505
1.2400	86.111	9.568	19.768	= 8 =	123458.85	28.2192	17.63697	0.25721
1.3950	96.874	10.764	22.2	= 9 =	138891.21	31.7466	19.84159	0.28936

The only material standard of customary length authorized by the U. S. Government is the Troughton scale, whose length at 59°.62 Fahr. conforms to the British standard. The yard in use in the United States is therefore equal to the British yard.

The only authorized material standard of customary weight is the Troy pound of the Mint. It is of brass

of unknown density, and therefore not suitable for a standard of mass. It was derived from the British standard Troy pound of 1758 by direct comparison. The British avoirdupois pound was also derived from the latter, and contains 7000 grains Troy.

The grain Troy is therefore the same as the grain avoirdupois, and the pound avoirdupois in use in the United States is equal to the British pound avoirdupois.

The British gallon = 4.54346 liters.

The British bushel = 36.3477 liters.

By the concurrent action of the principal Governments of the world an International Bureau of Weights and Measures has been established near Paris. Under the direction of the International Committee, two ingots were cast of pure platinum-iridium in the proportion of 9 parts of the former to 1 of the latter metal. From one of these a certain number of kilograms were prepared, from the other a definite number of meter bars. These standards of weight and length were intercompared, without preference, and certain ones were selected as International prototype standards. The others were distributed by lot to the different Governments and are called National prototype standards.

The metric system was legalized in the United States in 1866.

The International Standard Meter is derived from the Mètre des Archives, and its length is defined by the distance between two lines at 0° Centigrade, on a platinum-iridium bar deposited at the International Bureau of Weights and Measures.

The International Standard Kilogram is a mass of platinum-iridium deposited at the same place, and its weight *in vacuo* is the same as that of the Kilogramme des Archives.

The liter is equal to a cubic decimeter of water, and it is measured by the quantity of distilled water which, at its maximum density, will counterpoise the standard kilogram in a vacuum, the volume of such a quantity of water being, as nearly as has been ascertained, equal to a cubic decimeter.

Long ton: 2240 lbs. avoirdupois	= 1016 kilo.	Barrel of petroleum	= 42 gal.	= 1.59	hec.
Short ton: 2000 " "	= 907.2 "	" " salt	= 280 lbs.	= 127	kilo.
Pound avoirdupois	= 453.6 grams.	" " lime	= 200 "	= 90.720	"
Flask of mercury = 76½ lbs. avoir.	= 34.700 kilo.	" " natural cement	= 300	= 136.080	"
Troy ounce	= 31.104 grams.	" " Portland cement	= 400 "	= 181.440	"
Gallon	= 3.785 litres.	Gold coining value per oz. c. $20.6718	= $0.6646	per gram.	
		Silver " " " " c. $1.2929	= $0.04157	" "	

OFFICIAL UNITED STATES VALUES OF FOREIGN COINS, APRIL 1, 1894.

Country.	Standard.	Unit.	Value in U.S. Gold.	Coins.
			cts.	
Argentina	Both	Peso	96.5	Gold: argentine ($4.824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	20.3	Gold: former system—4 florins ($1.929), 8 florins ($3.858), ducat ($2.287), and 4 ducats ($9.158). Silver: 1 and 2 florins. Present system—Gold: 20 crowns ($4.052) and 10 crowns ($2.026).
Belgium	Both	Franc	19.3	Gold: 10 and 20 francs. Silver: 5 francs.
Bolivia	Silver	Boliviano	46.3	Silver: boliviano and divisions.
Brazil	Gold	Milreis	54.6	Gold: 5, 10, and 20 milreis. Silver: ½, 1, and 2 milreis.
Canada	Gold	Dollar	100	
Central America: Costa Rica, Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	46.5	Silver: peso and divisions.
Chile	Both	Peso	91.2	Gold: escudo ($1.824), doubloon ($4.561), and condor ($9.123). Silver: peso and divisions.
China	Silver	Tael ‡	68.6 / 76.5	
Colombia	Silver	Peso	46.5	Gold: condor ($9 647) and double-condor. Silver: peso.
Cuba	Both	Peso	92.6	Gold: doubloon ($5.017). Silver: peso.
Denmark	Gold	Crown	26.8	Gold: 10 and 20 crowns.
Ecuador	Silver	Sucre	46.5	Gold: condor ($9.647) and double-condor. Silver: sucre & divisions.
Egypt	Gold	Pound	494.3	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters. Silver: 1, 2, 5, 10, and 20 piasters.
Finland	Gold	Mark	19.3	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Both	Franc	19.3	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	23.8	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound	486.65	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Both	Drachma	19.3	Gold: 5, 10, 20, 50, and 100 drachmas. Silver: 5 drachmas.
Haiti	Both	Gourde	96.5	Silver: gourde.
India	Silver	Rupee	22.1	Gold: mohur ($7.105). Silver: rupee and divisions.
Italy	Both	Lira	19.3	Gold: 5, 10, 20, 50, and 100 lire. Silver: 5 lire.
Japan	Both*	Yen	99.7 / 50.1	Gold: 1, 2, 5, 10, and 20 yen. Silver: yen.
Liberia	Gold	Dollar	100	
Mexico	Silver	Dollar	50.5	Gold: dollar ($0.983), 2½, 5, 10, and 20 dollars. Silver: dollar (or peso) and divisions.
Netherlands	Both	Florin	40.2	Gold: 10 florins. Silver: ½, 1, and 2½ florins.
Newfoundland	Gold	Dollar	101.4	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	26.8	Gold: 10 and 20 crowns.
Peru	Silver	Sol	46.5	Silver: sol and divisions.
Portugal	Gold	Milreis	108	Gold: 1, 2. 5, and 10 milreis.
Russia	Silver	Ruble ‖	77.2 / 37.2	Gold: imperial ($7.718) and ½ imperial† ($3.86). Silver: ¼, ½, and 1 ruble.
Spain	Both	Peseta	19.3	Gold: 25 pesetas. Silver: 5 pesetas.
Sweden	Gold	Crown	26.8	Gold: 10 and 20 crowns.
Switzerland	Both	Franc	19.3	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
Tripoli	Silver	Mahbub §	41.9	
Turkey	Gold	Piaster	04.4	Gold: 25, 50, 100, 250, and 500 piasters.
Venezuela	Both	Bolivar	19.3	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

*Gold the nominal standard. Silver practically the standard. †Coined since January 1, 1886. Old half-imperial = $3.986. ‡Shanghai and Haikwan (Customs). §Of 20 piasters. ‖Silver the nominal standard. Paper the actual currency, the depreciation of which is measured by the gold standard.



RICHARD P. ROTHWELL.

CONTRIBUTORS.

It would be impossible to name here all who have aided us in the collection of statistics and other information for the present volume, but we give in the following pages brief biographies of those who have contributed special articles on technical and other topics, in order that readers may appreciate the high professional standing of those who have assisted in the work. In a number of cases these sketches are accompanied by portraits.

While space prevents us from further extending the list, we desire to express our grateful appreciation of the ready assistance and coöperation we have received from thousands of miners and other producers, and of State and other officials, in the collection of statistics of mineral production and other information for the present volume. It is this general appreciation of the value of the work, this hearty coöperation, that has enabled us to make the statistics complete and to bring the figures up to the latest dates. Special acknowledgment is due to Mr. Worthington C. Ford, chief of the Bureau of Statistics of the Treasury Department, who has with unflagging interest aided in the compilation of the statistics of imports and exports of the United States; to William B. Cogswell, Esq., for statistics relating to the Solvay process; to W. R. Shields, Esq., for statistics of bromine; to F. M. Smith, president of the Pacific Coast Borax Company, for statistics of borax; to Mr. J. Langeloth, president of the American Metal Company, Mr. John Stanton, of the Copper Producers' Committee, and Mr. Lucius A. Cole, president of the National Lead Company, for valuable aid in compiling the statistics of metal production; to Mr. A. Hanauer, Jr., of Salt Lake City, H. M. Curry, general manager of the Carnegie Steel Company, A. Hecksher, Esq., of the New Jersey Zinc Company, Marshall Morris, president of the Breckenridge Asphalt Company, of Louisville, Ky., and to the California State Mining Bureau, which furnished us with its statistics of mineral production in California in advance of its own publication, and to the inspectors of coal mines in the several States and Territories.

We wish once more, not only for ourselves, but on behalf of all who are interested in the great industry which this volume represents, to thank those who have aided us in making it a faithful history of a year's progress of the industry.

The sketches of the special contributors in the following list are given in alphabetical order.

Arnold, J. O., of Irish birth, was educated at King Edward's Grammar School, Birmingham. For two years he was a cadet on H. M. S. Conway, and for two years a midshipman in the Royal Naval Reserve. In 1878 he entered the engineering department of the Sheffield Steel and Iron Works. Subsequently he served in the laboratory and testing department, ultimately becoming chief chemist and test master. He was also for two years consulting chemist to the Farnley Iron Company, Leeds; two years chemist and manager at the Spanish Steel Works, Sheffield; and two years in practice in Sheffield as consulting chemist and metallurgist. In 1889 he was appointed professor of metallurgy at the Sheffield Technical School, then the technical department of Firth College, and recently incorporated by royal charter as the Technological Department of Sheffield University College. In 1887 he was awarded by the Institute of Civil Engineers a Telford premium for a paper on "The Influence of Chemical Composition on the Strength of Bessemer Steel Tyres," and in 1895 a Telford medal and premium for researches on the influence of carbon on iron.

BARRINGTON, REGINALD W., was born at Feltham, England, in 1868, but was taken to Cartagena, Spain, at an early age, and did not return to England until after graduating from the Murcia "Instituto." At the age of 18 he entered business with his father, who had long been connected with the iron and manganese industry of Spain, but in 1890 started on his own account as mine owner and exporter of iron and manganese ores. In 1891 the firm of Barrington & Holt was established; it is at present operating upward of fifteen iron and manganese mines in the Cartagena and Portman district. The management of these mines is entrusted to Mr. Barrington. Mr. Barrington has taken great interest in the development of the mineral resources of Spain, and has made visits to most of the important iron districts of the south and southwest of the Peninsula, several of which he has described in papers for the technical press.

BARTLETT, F. L., was born at Hanover, Me., in 1852. At the age of 18 he entered the University of Michigan, where he took special courses in chemistry and mineralogy. At 20 he was appointed State Assayer at Portland, Me., which post he held for 16 years. At the same time he filled the position of professor of natural science at Westbrook College, near Portland. During the mining excitement in Eastern Maine Dr. Bartlett visited England, Wales, Germany, and France, under commission from the Governor of Maine, to study the European methods of ore reduction, and on his return built and successfully operated the Portland Smelting Works. From 1878 to 1890 Dr. Bartlett conducted extensive experiments in the treatment of zinky ores at the Portland works and afterwards he removed to Colorado and built the American Zinc-Lead Works, which have been in operation since 1891. Dr. Bartlett contributes to this volume a monograph on the "Treatment of Zinc-Lead Sulphide Ores."

BLANDY, JOHN F., was born in Newark, Del., in 1833, and studied at Delaware College. At 18 years of age he was attached to the engineer corps of the Philadelphia & Reading Railroad, but after a year's service went to Germany and studied there three years. From 1855 to 1863 he was engaged in the Lake Superior copper region. In 1863 he became connected with the Little Schuylkill Railroad and Coal Company, and managed that company's affairs until he went to Arizona in 1880, where he was Territorial Geologist in 1889 and 1890. Mr. Blandy has contributed to the *Engineering and Mining Journal* and to the various volumes of THE MINERAL INDUSTRY many notes in relation to the mines and minerals of Arizona.

BLUE, ARCHIBALD, was born in 1840 in Oxford, Ontario, Canada. After being engaged as a teacher and as a journalist, in 1881 he was appointed to organize and take charge of the Bureau of Industrial Statistics under the Department of Agriculture; in 1884 he was made deputy head of the department. During 1889–90 he served as secretary of the Royal Commission upon the Mineral Resources of Ontario, and in 1891 became director of the Bureau of Mines of Ontario. Mr. Blue furnishes statistics relating to Ontario for this volume.

BORCHERS, WILHELM, Ph.D., was born in 1856, in Goslar, Hanover, and studied at the universities of Greifswald (1875–76), Erlangen (1876–78), and Munich (1878–79). In 1879 he became connected with the chemical works of E. de Haën in Hanover. In 1882 he went to the United States, and built the works of the Colonial Chemical Company in West Medford, Mass., which he managed until 1887, when he returned to the works of De Haën in Hanover. In 1891 he left this position to make some special studies in the Bergakademie at Clausthal and to devote himself to electro-metallurgy, but in the following year he was made instructor in metallurgy and technical chemistry in the Royal Engineering and Metallurgical School at Duisburg. Dr. Borchers has written and published *Elektrometallurgie* (1891), *Anorganische Chemie* (1893), and in connection with Dr. W. Nernst, *Jahrbuch der Elektrochemie* (1895 and 1896). He is now one of the editors of *Zeitschrift für Elektrochemie*. He contributes to Vol. V. the article on the "Progress of Electro-chemistry and Electro-metallurgy."

BREWER, WILLIAM M., was born at Kensington, England, in 1851, was educated at the Philological School, London, and came to the United States in 1870, proceeding directly to the West, where he remained, chiefly in the Black Hills of South Dakota, until 1891. In that year he removed to the Southern States, where he has since then been occupied as consulting engineer, and in 1893 and 1894 as Assistant State Geologist of Alabama.

BROMLY, A. H., was born in Bristol, England. In 1883 he was articled to Messrs. John

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.



J. O. ARNOLD.



F. L. BARTLETT.



R. W. BARRINGTON.



JOHN F. BLANDY.



W. BORCHERS.



WILLIAM M. BREWER.



LUCIUS P. BROWN.

Warner & Sons, of London, engineers. In 1888 and 1889 he was in South Africa as assistant engineer to Professor Louis, who was then general manager of a Transvaal mine. From 1892 to 1894 he was manager of the Carndochan mine, North Wales. In 1894 he went to Burma, where he has since been engaged in gold mining, having erected at Choukpazat the first stamp mill to be operated in that country. Mr. Bromly has contrib ted the article on Burma in this volume.

BROWN, LUCIUS P., was born in Tennessee, and studied at the University of Virginia. He was then engaged as chemist with the State Experiment Station of Tennessee for nearly a year, and later opened an analytical laboratory at Ætna, Tenn., becoming associated subsequently with Mr. C. G. Memminger. Mr. Brown has given much study to the subject of agricultural chemistry and fertilizers, and has been connected with the phosphate industry in Tennessee from the first opening of the deposits.

COGSWELL, WILLIAM B., was born in Oswego, N. Y., in 1836, and after several years of experience in civil engineering on the Oswego & Syracuse Railroad, entered the Rensselaer Polytechnic Institute, from which he graduated in 1852. After serving in the Lawrence Machine Shop, at Lawrence, Mass., he was made master mechanic of the Marietta & Cincinnati Railroad, and was subsequently in charge of the Broadway Foundry in St. Louis. During the civil war he served as engineer in the navy, having charge of repair shops at Port Royal and elsewhere on the Atlantic coast. After spending some time in bridge building and other work, he was appointed manager of the Mine la Motte in Missouri, which position he held for several years. In 1881 he took charge of the construction of the Solvay Works at Syracuse, N. Y., and since that time has been connected with the company as treasurer and general manager. Mr. Cogswell furnishes to this, as to previous volumes, valuable information concerning the operations of his company.

CREMER, RICHARD, born at Tecklenburg, Westphalia, early in life engaged in coal-mining in Westphalia; later he studied at the mining academies at Clausthal and Berlin. After completing his academical studies he was employed at the Gold und Silber-Scheide-Anstalt at Magdeburg, and subsequently at the Ilseder Hütte in Hanover, and at the Shamrock Colliery in Westphalia. He then received his present appointment as mining engineer of the Vereins fur die bergbaulichen Interessen im Oberbergamtsbezirk Dortmund in Essen, and at the same time became technical editor on the staff of *Glückauf*. Mr. Cremer has contributed the article on "Coal Washing in England and Germany" to the present volume.

DENNIS, L. M., studied at the University of Michigan 1881–86, receiving the degree of Ph.B. in 1885 and B.S. in chemistry in 1886. After teaching for two years in La Porte, Ind., he was for two years instructor in chemistry at Cornell University. During 1889–91 he studied in Germany under Fresenius in Wiesbaden, Classen in Aachen, Hempel in Dresden, and Krüss in Munich. In 1891 he returned to Cornell University as assistant professor and was made associate professor of analytical chemistry in 1893. He wrote the article on "The Rare Elements" for Vol. IV., and on the same subject for the present volume.

DOUGLAS, JAMES, a Canadian by birth, came to the United States in 1895 to take charge of copper works at Phœnixville, Pa., and since that time has resided in this country. He is best known in connection with the copper industry of Arizona, with which he has been identified almost since its beginning. He is president of the Copper Queen Consolidated Mining Company and also of the Arizona & Southeastern Railroad Company. He was associated with Dr. T. Sterry Hunt in the invention of the well-known Hunt & Douglas process, and is also the inventor of an improved roasting furnace and other metallurgical appliances. In 1895 Mr. Douglas was elected Cantor lecturer at the University of Oxford. He has given much valuable assistance in the preparation of this volume.

FORD, WORTHINGTON C., is chief of the Bureau of Statistics in the Treasury Department of the United States, in the work of which he has made many noteworthy improvements. Mr. Ford has had long experience in statistical work, having for several years filled the office of Chief of Bureau of Statistics in the State Department, where his work is worthy of the highest commendation. We are indebted to Mr. Ford in the preparation of this, as in previous volumes, for valuable assistance in the compilation of our statistics.

FOSTER, CLEMENT LE NEVE, was born in 1841 at Camberwell, England. After four years at school in France he entered the Royal School of Mines in 1857 and acquired the associateship in two years, going then to Freiberg to complete his studies. In 1860 he received an appointment to the Geological Survey of the United Kingdom under Murchison, and held it until 1865. In 1865 he obtained the degree of Doctor of Science from London University. Resigning from the Geological Survey Dr. Foster became lecturer to the Miners' Association of Cornwall and Devon, succeeding Richard Pearce. In 1867 he went to Egypt to make geological and mining explorations in the peninsula of Sinai, under Mr. H. Bauerman, and in 1868 went to Venezuela to report on the Cataral gold fields. He was then for three years engaged by the Pestarena Gold Mining Company in the Alps. In 1873 he received the appointment as Inspector of Mines for Cornwall, and in 1880 was transferred to the district of North Wales, where he has been stationed ever since, being now second in order of seniority among H. M. Inspectors of Mines. Dr. Foster is a Chevalier of the Legion of Honor, and since 1889 has been a fellow of the Royal Society. He is a member of many other scientific societies, from which he has received high honors. He translated from the Dutch the treatise on the tin deposits of Banca by Van Diest, and in conjunction with Galloway translated Callon's *Lectures on Mining*. He is the author of the monograph on "Mining" in the *Encyclopedia Britannica*, and the text-book *Ore and Stone Mining*, published in 1894, and is now the editor of *The Mineral Statistics*, and *Annual General Reports on the Mineral Industry of the United Kingdom*.

GILPIN, EDWIN, JR., born in 1850 at Halifax, N. S., graduated from King's College, Windsor, N. S., in 1871, and then served for a time at the Albion mines, Pictou County. Later he was at the collieries of Sir George Elliott and others in England. Upon his return to Nova Scotia he engaged in professional work, and succeeded Sir William Dawson in the mapping of the iron ores of Pictou County. In 1879 Mr. Gilpin was appointed Inspector of Crown Mines in Nova Scotia, and in 1886 was made Deputy Commissioner of Public Works and Mines. Mr. Gilpin rendered valuable assistance to THE MINERAL INDUSTRY by supplying the statistics relating to Nova Scotia.

GRAVES, H. G., of London, was educated at the Royal School of Mines, of which he became an associate in mining and metallurgy in 1884 and 1885, also taking the De La Beche medal in the latter year. After a short time with the Rio Tinto Company he was appointed an examiner in the Patent Office. He is a member of the Federated Institution of Mining Engineers, an abstractor for the *Journal of the Iron and Steel Institute*, and a contributor to various technical publications.

GROHMANN, E., who contributes the article on the mineral industry of Greece to the present volume, is a resident of the island of Seriphos, where he is engaged in mining.

HELMHACKER, R., born in 1840 at Prague, having completed his education at the polytechnic schools and universities at Prague and Vienna, became attached to the mining and metallurgical school at Pribram, Bohemia, as a crown stipendiary. In 1863 he was appointed mining engineer and surveyor in the coal mines at Rovitz, Moravia, and was afterward engaged as superintendent of the collieries and iron mines at Kladno, Bohemia; also at the gold and antimony mines and smelting works of Bohemia. He was then appointed professor of mineralogy, mining geology, and paleontology at the mining school at Leoben, Styria. Recently he was engaged as managing mining engineer and geologist on the Altai Mountain gold placers belonging to the Russian crown and at the imperial gold and platinum mines in the Oural Mountains. He is now consulting engineer to a Franco-Russian company which is developing coal and iron mines and erecting smelting works in Siberia. Mr. Helmhacker has contributed to the present volume the article on "Salt in Austria." It is interesting to note that Professor Helmhacker, who knows five languages, writes his contributions to THE MINERAL INDUSTRY in English.

HOBART, FREDERICK, A.M. of the College of the City of New York, has been connected with the Jersey City Locomotive Works; Bullock Machine Company, Jersey City; Wrigley Machine Works, Newark, N. J.; Grant Locomotive Works, Paterson, N. J.; has been assistant editor of the *Railroad Gazette*, associate editor *Railroad and Engineering Journal*, assistant editor *Engineering and Mining Journal*, a contributor to various technical periodicals, and is the

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.



W. B. COGSWELL.



RICHARD CREMER.



L. M. DENNIS.



JAMES DOUGLAS.



C. LE NEVE FOSTER.



EDWIN GILPIN, JR.



H. G. GRAVES.

translator of *Notes on Steam Hammers* and *Economies in the Combustion of Fuel.* Mr. Hobart has contributed to this volume the reviews on "Iron and Steel."

HOFMAN, H. O., was born in 1852 at Heidelberg, Germany. He studied at the Bergakademie at Clausthal, where he graduated in 1877 in mining engineering and metallurgy. He was then appointed chemist and assistant at the government smelting and refining works at Lautenthal in the Harz. In 1881 he came to the United States and was employed successively at Mine La Motte in Missouri, at the Argentine Smelting and Refining Works, and as metallurgist of the Delaware Lead Company in Philadelphia. When the last-named works were closed he went to Colorado, and after running the Rico smeltery for a short time went to Park City, Utah, to study the amalgamation and lixiviation of silver ores at the Ontario Mill, where Russell was then making his experiments. After a short time spent in charge of a smeltery in Mexico he was appointed assistant to Professor Richards at the Massachusetts Institute of Technology in Boston; from there he went to the School of Mines of South Dakota as professor of metallurgy and assaying, where he remained until called back to the Massachusetts Institute of Technology to the professorship of metallurgy, which he now holds. Dr. Hofman has made numerous contributions to technical literature, his most important work being the the admirable treatise on *The Metallurgy of Lead.* For his paper on the "Dry Assay of Tin Ores" the degree of Ph.D. was conferred on him by the University of Ohio. For the present volume of THE MINERAL INDUSTRY he has reviewed the progress in metallurgy of lead, continuing the valuable series of articles from his pen which was begun in the first volume.

HOFMANN, OTTOKAR, was born in 1843 at Ruskberg in Austria-Hungary. He was educated at the Realschule and Polytechnic Institute in Vienna, and the Bergakademie at Freiberg, graduating from the last in 1866. In 1867 he came to the United States, first entering the assay office of Guido Küstel in San Francisco as a partner. In 1868 Mr. Hofmann erected lixiviation works at La Dura, Sonora, Mexico, which was the first introduction of the silver leaching process in North America. The success at La Dura attracting much attention, Mr. Hofmann received numerous engagements for the construction of leaching works, building among others those at La Trinidad and San Marcial, Sonora, and Triunfo, Lower California. While at La Dura he substituted calcium polysulphide for sodium polysulphide as a precipitant. Returning to the United States Mr. Hofmann served as metallurgist in various amalgamating and chlorination mills in Nevada and California. In 1878 he introduced a successful leaching process of his own device for gold-silver ore at Monitor, Cal., and in 1880 erected the works of the Silver King Company at Pinal, Ariz. Since that time Mr. Hofmann has been occupied almost exclusively in this branch of metallurgy, of which he is the leading exponent, He has been engaged successively at St. Louis, Mo. (1882–83), Cusihuiriachic, Chihuahua (1885), Yedras, Sinaloa (1886–87), Cusihuiriachic (1887), Parral, Chihuahua (1888). In 1890 he was appointed managing director of the North Mexican Mining and Milling Company, operating at Cusihuiriachic, which position he held until the spring of 1894. While at "Cusi" he substituted trough-lixiviation for tank-lixiviation. In 1894 he was engaged professionally at Sombrerete and at Parral. In 1895 Mr. Hofmann went to Argentine, Kan., to take charge of the leaching department, in which the Hunt & Douglass process is employed, and still occupies that position. Mr. Hofmann has made many valuable contributions to metallurgical literature, chiefly on his own specialty, through the *Engineering and Mining Journal* and the *Transactions of the American Institute of Mining Engineers.*

INGALL, ELFRIC DREW, was born in 1858 at Greenhithe, Kent, England. He studied at the Royal School of Mines, and afterward in Cornwall and Wales. In 1880 he went to Canada to examine mining properties on Lake Superior, and later became manager of a company there. He subsequently spent some time in Wyoming examining mines, and in 1884 joined the Canadian Geological Survey, with which he has since then remained as mining engineer in charge of the Division of Mineral Statistics. We are indebted to Mr. Ingall for statistics relating to Canada furnished to this volume.

INGALLS, WALTER RENTON, a well-known mining engineer and metallurgist, is a graduate of the Massachusetts Institute of Technology. In 1886–90 he was engaged in mining at Leadville, and elsewhere in Colorado. In 1890–92 he was assistant editor of the *Engineering and*

Mining Journal, resigning that position to go to Mexico to open tin mines in the State of Durango. In 1893 and 1894 he visited professionally various mining districts in the United States, Canada, Belgium, Germany, and Poland, devoting himself especially to the metallurgy of zinc, and was assistant editor of THE MINERAL INDUSTRY, Vols. I. and II. In 1895 he was manager of cyanide reduction works at Cripple Creek, Colo., and in 1896 of copper-matte smelting works in Durango, Mexico. He is the author of the articles on mining and metallurgical machinery in *Appleton's Cyclopedia of Mechanics* (1892), and contributed to the present volume the articles on "Borax and Boracic Acid," "Zinc and Cadmium," and "Lead."

JENKS, CHARLES N., was born in 1855 at North Brookfield, Mass. He became engaged in corundum mining in 1871 and followed it more or less until 1889, when he was appointed superintendent of The Western North Carolina Mining and Improvement Company, and was employed in locating and prospecting for corundum deposits in North Carolina and Georgia. In 1891 he became also superintendent of The Western North Carolina Corundum Company, and in 1892 the manager of The Sapphire Valley Company. Mr. Jenks contributes to this volume the articles on "Sapphire" and "Corundum."

KERL, BRUNO, was born in 1824 at St. Andreasberg in the Upper Harz. He studied first at the Bergschule at Clausthal, and then at the university at Göttingen. Having completed his studies he received an appointment at the Okerhütte in the Lower Harz, but in 1846 left it to become instructor in chemistry, assaying, and metallurgy at Clausthal, where he remained until 1867, having been in the meanwhile promoted to a professorship. During this time he also filled several subsidiary positions, such as mint-warden, mines-assayer, and assessor with the Mines and Forests Office. In 1867 he was called to the chair of metallurgy, assaying and chemical technology at the Bergakademie at Berlin, which he has held to the present time. For many years he was also a member of the Imperial Patent Office, and of the Royal Commission for Technical Industry under the Prussian Ministry of Commerce, but with increasing years was obliged to give up these duties. He received the honorary title of Geheimer Bergrath, and since 1891 has been an honorary member of the American Institute of Mining Engineers. Professor Kerl has been a voluminious contributor to technical literature. Among his more important works have been *Oberharzischen und Rammelsberger Hüttenprozesse*, *Handbuch der metallurgischen Hüttenkunde* (4 vols.), *Grundriss der Eisenhuttenkunde*, *Metallurgische Probirkunst*, *Probirbuch*, *Leitfaden bei Lothrohrenuntersuchungen*, *Grundriss der Salinenkunde*, *Handbuch der Thonwaarenindustrie*, and *Grundriss der Metallhüttenkunde*. Several of these have been translated into English. In conjunction with Professor Stohmann, Professor Kerl has recently rendered into German the fourth edition of Muspratt's *Technical Chemistry*. In 1859 Professor Kerl became one of the editors of the *Berg- und Huttenmännischen Zeitung*, his associate being Friedrich Wimmer. On April 1 of the current year his advanced age and poor health compelled him to give up this editorship, which he had held continuously for 38 years. Under his guidance this journal has been one of the most complete compendiums of progress in mining and metallurgy, and its bound volumes hold an honored place in every technical library. The amount of work that Professor Kerl himself contributed to it has been immense, and he laid down his pen with honor and with the regret of the whole profession, many of the members of which studied under him at Clausthal and Berlin, while many more know him through his valuable series of text-books, which are classics.

KEYES, DR. CHARLES ROLLIN, was born in Des Moines, Iowa, in 1864. He studied in Des Moines and at the Iowa State Academy at Iowa City, graduating in 1887. He took a course at the Johns Hopkins University in Baltimore and undertook certain original investigations which resulted in the preparation of several valuable papers. When the Iowa State Geological Survey was instituted Dr. Keyes was appointed Assistant State Geologist. In 1894 he took charge of the Missouri Bureau of Geology and Mines. Dr. Keyes has published many monographs and articles, the most important of which are the three volumes of the Iowa Geological Survey upon *The Geological Formations of the State*, *The Coal Deposits*, and *The Gypsum Deposits*, and the *Report on the Paleontology of Missouri*. Dr. Keyes has contributed to the present volume the article on "Distribution and Character of Missouri Clays."

KNIGHT, WILBUR C., was born in Illinois in 1858 and moved to the Western frontier when a

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.



R. HELMHACKER.



FREDERICK HOBART.



H. O. HOFMAN.



OTTOKAR HOFMANN.



WALTER RENTON INGALLS.



CHARLES N. JENKS.



CHARLES R. KEYES.

boy. He was educated at the University of Nebraska, taking special work in geology and chemistry. After graduation he was the first Assistant Territorial Geologist of Wyoming for two years; and for six years was superintendent and general manager of mining corporations in Wyoming and Colorado. In 1893 he was called to the chair of mining in the School of Mines, University of Wyoming, and in 1894 was elected to the chair of geology and mining in the same school. Professor Knight furnishes the paper on "The Petroleum Fields of Wyoming" in the present volume.

LANGELOTH, J., president of the American Metal Company and vice-president of the Balbach Smelting and Refining Company, has furnished very valuable information concerning the trade department of the mineral industry with which his company is so prominent a factor, and has with unfailing courtesy and good will aided in many ways our efforts to secure full and reliable information.

LE BARON, J. F., was born in Boston in 1847, graduated at Lawrence Academy and studied engineering in the office of J. H. Shedd, Boston. He entered active practice in 1868, in city work and railroad work in Massachusetts, subsequently becoming chief engineer of the Fitchburg Railroad. In 1877 he went to Florida to make scientific explorations for Harvard, was then appointed chief engineer and superintendent of the St. John's & Indian River Railway, and later became a United States assistant engineer in South Florida. He then became engineer of the Florida Ship Canal and Transit Company. In 1881 he discovered large phosphate deposits in Florida. In 1887 he became connected with the Nicaragua Canal, becoming finally general superintendent and acting chief engineer, resigning this position in 1890. In the same year he was appointed United States engineer with the corps engaged on river improvements in Tennessee and Alabama.

LOUIS, HENRY, was born in London in 1855. He first attended school in Bavaria, then in London, and in 1873 entered the Royal School of Mines, the associateship of which he obtained with high honors in 1876, receiving the De la Beche medal. He was then engaged for some time in Dr. Percy's private laboratory, making some of the researches for Dr. Percy's treatise on *Silver*. Thence he was sent by Sir William Siemens to Nova Scotia, for the Steel Company of Canada. Later he went to South America, where he was employed in gold mining in Colombia and Ecuador; from there to the Gold Coast, and finally to South Africa, where he spent over three years in gold and diamond mining. He then spent three years managing and inspecting mines and exploring in the far East, including the Malay Peninsula (gold and tin), Siam, Tonkin, and Bengal. Upon his return to Europe he devoted himself to consulting work and the management of an iron mine in Spain. In 1895 he was appointed professor of mining in the Durham College of Science, Newcastle-upon-Tyne, and in 1896 received from the University of Durham the honorary degree of M.A. Mr. Louis is the author of a *Handbook on Gold Milling* (1894), and a revised and enlarged edition of *Phillips' Treatise on Ore Deposits* (1896). To this volume of THE MINERAL INDUSTRY he contributes the elaborate monograph on the "Metallurgy of Tin."

LUNGE, GEORGE, was born in 1839 at Breslau, Silesia, Prussia. He studied at the University of Breslau, where he received the degrees of M.A. and Ph.D., and then at Heidelberg under Bunsen and Kirchhoff. In 1864 he went to Great Britain, where he became chemist and later superintendent of the coal-tar and ammonia works at Wolverton. In 1865 Dr. Lunge took a position as chemist at the sulphuric acid and alkali works at South Shields, of which he became later superintendent and finally part owner, his connection with them lasting until 1876, when he left England to accept the post made vacant by the death of Emil Kopp at the Swiss Federal Polytechnic School at Zurich, where he is now professor of technological chemistry and dean of the faculty of chemistry. Dr. Lunge has contributed to technical literature some of its most important works. His *Coal Tar and Ammonia*, *The Manufacture of Sulphuric Acid and Alkali* and *The Alkali-Maker's Hand-Book* are standard works on these subjects. To the present volume Dr. Lunge has contributed "Progress and Profits in the German Chemical Industry," and "Modern Coke Ovens and their By-products." It is interesting to note that all of Dr. Lunge's treatises have been written by himself in both English and German, while the French translations have been revised by him.

PENHALE, MATTHEW, was born in Cornwall in 1839, and while still a boy went to work in the tin dressing works and later in the mines. He came to America in 1866, and was first employed at the Bruce mines, and then successively at various mines in Lake Superior, and in the New Hamburg tunnel of the Hudson River Railway. In 1871 he became connected with the Musconetnong Iron Company of New Jersey. In 1882 he became captain of the Stirling mines. In 1887 he went to Quebec as superintendent for the Scottish Canadian Asbestos Company, and is now manager of the same mines for the Glasgow & Montreal Asbestos Company, Limited. Mr. Penhale has made a careful study of asbestos mining, and few men are better acquainted with the Canadian mines and methods.

PHILLIPS, WILLIAM BATTLE, was born in 1857 at Chapel Hill, N. C. He was educated at the Bingham School, at Davidson College, and at the University of North Carolina, graduating from the last in 1877. Immediately thereafter he entered the State Agricultural Experiment Station as chief chemist under Dr. A. R. Ledoux. In 1883 he accepted the position of chemist to the Navassa Guano Company, Wilmington, N. C., resigning this place in 1885 to become professor of agriculture, chemistry and mining in the University of North Carolina in 1885. In 1885–86 he spent a year at the Bergakademie at Freiberg. In 1890 Dr. Phillips accepted the professorship of chemistry and metallurgy in the University of Alabama. In January, 1892, he resigned this position to become chemist and assistant superintendent to the Grand Rivers Company, Grand Rivers, Ky. Recently he has been connected with the Tennessee Coal, Iron and Railway Company. Dr. Phillips has been a frequent contributor to the *Engineering and Mining Journal* and *Transactions of the American Institute of Mining Engineers*. He was assistant editor of the *Engineering and Mining Journal* for a short time in 1892–93, and in conjunction with E. Prochaska is the translator of Wedding's *Basic Bessemer Process*.

RICHARDS, ROBERT HALLOWELL, professor of mining and metallurgy at the Massachusetts Institute of Technology, was born in 1844 at Gardiner, Me. He graduated in 1868 from the Massachusetts Institute, being a member of its first class, and became assistant in chemistry in the corps of instruction, passing successively to the posts of instructor, assistant professor of chemistry, professor of mineralogy and assaying, professor of mining engineering, and in 1884 to his present professorship of mining and metallurgy. Under his administration the mining and metallurgical laboratory, which was the first of its kind in an educational institution, has been developed to a high degree of excellence, and has been a model for similar laboratories in other colleges. In addition to his professorial duties Professor Richards has been actively engaged as a consulting engineer in mining and metallurgical work, and has been the inventor of several ingenious devices, which have found extended use in practice. He has contributed many valuable papers to the technical press and to the transactions of various scientific societies. His most recent work on the principles of ore-dressing is the admiration of the entire body of engineers engaged in that branch of metallurgy. To the several volumes of THE MINERAL INDUSTRY he has contributed the comprehensive reviews of "Progress in Ore Dressing.

ROBERTS-AUSTEN, W. CHANDLER, F.R.S., was born in 1843. The ancestry of his father (George Roberts) was Welsh; his mother belonged to the old Kentish family of Chandler. In 1885, at the request of his uncle, the late Major Austen, J.P., of Haffenden and Camborne, in Kent, Mr. Roberts obtained royal license to add the name of Austen to his own. Roberts-Austen entered the Royal School of Mines in 1861, with a view of becoming a mining engineer, but on obtaining the associateship of the school, the late Professor Graham, then Master of the Mint, secured his services. With Graham he conducted a remarkable series of researches, and on Graham's death, in 1869, he succeeded to Graham's appointment as Assayer to the Mint, being subsequently in 1882 entrusted with the duties of the Queen's Assay Master. In 1880, on the retirement of Dr. Percy, Roberts-Austen was appointed to the chair of metallurgy in the Royal School of Mines, a post which he still holds in addition to his office at the Mint. He was elected a Fellow of the Royal Society in 1875, and was one of the founders of the Physical Society of London, of which he was for some time secretary, and afterward a vice-president. He was a member of the executive council of the Inventions Exhibition in London, 1885, and served on the British Executive Council of the Paris Exhibition in 1889. He was chosen vice

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.



BRUNO KERL.



WILBUR C. KNIGHT.



HENRY LOUIS.



GEORGE LUNGE.



WILLIAM B. PHILLIPS.



ROBERT H. RICHARDS.



W. C. ROBERTS-AUSTEN.

president of the International Mining and Metallurgical Congress in Paris. Professor Roberts-Austen has made many valuable researches in metallurgy, chiefly on the subject of alloys. He contributes to the present volume of THE MINERAL INDUSTRY a valuable article on the "Diffusion of Metals." Besides his scattered contributions to metallurgical literature, Professor Roberts-Austin is the author of the treatise *An Introduction to the Study of Metallurgy.*

ROTHWELL, JOHN E., was born at Kingston, Ontario. He acquired his knowledge of mining and milling by practical experience, beginning in the machine shops attached to a mine and mill in Canada and subsequently working his way up gradually to be superintendent of the plant while still only a youth. He was afterward engaged under the late John Heard, at the works of the Brunswick Antimony Company, near Boston, Mass. He hen went to Deadwood, S. Dak., where he remodeled and afterward managed with great success the mill of the Golden Reward Company, using barrel chlorination. Subsequently he built several mills in Nevada and Colorado. Although experienced both in mining work and as a mechanical engineer, Mr. Rothwell has made a specialty of the roasting and treatment of refractory gold ores, making an excellent record for successful and economical work. He has made many improvements in chlorination plants and processes and has contributed valuable papers on the subject to technical periodicals. He has furnished to the present volume "The Present Development of the Barrel Chlorination Process."

SHIELDS, W. R., of Ohio, furnished THE MINERAL INDUSTRY with valuable statistics concerning the bromine industry, with which he is closely connected.

SPRAGUE, TIMOTHY WILSON, is a graduate of the Massachusetts Institute of Technology, class of 1887, mining department. After graduation he remained as assistant to Prof. Robert H. Richards one year, then for a year was on the staff of the *Electrical Review.* He then became connected with the Thomson-Houston Electric Company in its motor and mining department, and remained with it and its successor, the General Electric Company, five years. He was one year with the engineering firm of Westinghouse, Church, Kerr & Co., and is now consulting engineer in New York and Boston, his specialties being electrical mining installations and power transmission. Mr. Sprague has contributed to this volume the article on "Electricity in Mining Work."

STANTON, JOHN, is the secretary and treasurer of the Atlantic, Allouez, Central and Wolverine copper-mining companies of Michigan, whose affairs he has conducted with great ability and marked success. Mr. Stanton has for several years acted as statistician for the great copper producers of the United States, who exchange statistics monthly with the European producers, and has aided in the preparation of this volume by much valuable statistical information relating to copper.

ULKE, TITUS, was born in 1866, at Washington, D. C. In 1889 he graduated from the Royal School of Mines at Freiberg, Saxony, as metallurgical engineer. After spending some time in visiting the various mines and metallurgical works of Europe, Mr. Ulke returned to this country and was engaged as chemist to the Harney Peak Tin Company in South Dakota. In 1891 he became assayer for the United Smelting Company, and afterward was engaged by the Anaconda Mining Company as chemist at its electrolytic copper refining works. In 1893 Mr. Ulke acted as metallurgist to the Mines and Mining Department of the Chicago Columbian Fair, and was later employed at the Guggenheim works at Perth Amboy, N. J. Mr. Ulke has contributed articles on "The Electrolytic Determination of Copper and Nickel," "Progress in Electrolytic Copper Refining," and "Parting and Refining Gold and Silver," to the present volume.

WILLOUGHBY, WILLIAM FRANKLIN, was born in 1867 at Alexandria, Va. Graduating from Johns Hopkins in 1888 he first studied law, but in 1890 entered the service of the United States Department of Labor as expert, with which he has since been connected. He has been sent to Europe on three occasions on official work for this department, the last time to represent the government at the International Congress in Relation to Accidents to Labor and Workingmen's Insurance, which met at Milan, and to make a special investigation of industrial communities. The report of this inquiry was published recently in a series of articles in the *Bulletin* of the Department. In 1894 Mr. Willoughby was appointed the American corre-

spondent of the *Musée Social* of Paris, an institution endowed by the Comte de Chambrun with over a quarter of a million of dollars, and having for its purpose the scientific investigation of labor problems and conditions throughout the world. Mr. Willoughby is a member of the American Economic and Statistical Associations and a frequent contributor to economic reviews on economic and statistical subjects.

WURTZ, HENRY, was born at Easton, Pa., 1828, and graduated from Princeton, 1848, where he enjoyed the tutelage of the illustrious Joseph Henry. Afterwards he studied at the Lawrence Scientific School, Cambridge, where he graduated in 1850. He was then appointed instructor at the Yale Scientific School, and while there made the first determinations of alkalies in silicates by fusing with earthy chlorides. In 1854 he became State Chemist of New Jersey, and was attached to the geological survey of that State; in 1857 professor of chemistry at Queen's College, Kingston, Canada; in 1858 professor at the National Medical College, Washington. From 1851 to 1861 inclusive he was chemical examiner in the Patent Office; from 1870 to 1875, inclusive, editor of the *American Gas Light Journal;* in 1876 a judge at the Centennial Exposition, and received an honorary medal for chemical investigation of Japanese porcelain-minerals, and identification thereof of petrosilex. In 1877 he received for his chemical discoveries the honorary degree of Ph.D. from the Stevens Institute of Technology, this being the first degree conferred by that institution. In 1858 he discovered the relations of sodium to gold, and invented sodium-amalgamation. In 1866 he first set forth the dynamic origin of heat of rock-metamorphism, and predicted the finding of gold in the ocean six years before Sonstadt's discovery. He predicted publicly fifteen years before natural gas utilization all that was afterward realized. He predicted, and largely contributed to the introduction of cheap water-gas. In 1876 he first introduced geometric principles into chemistry, and since then has made leading discoveries which are soon to be published.

THAT most important department of this work, its business and financial administration, has been, as in previous years, under the control of MRS. SOPHIA BRAEUNLICH, who is treasurer and business manager of the Scientific Publishing Company and of the *Engineering and Mining Journal*, and who has been for many years on the staff of that paper. To her remarkable business ability is due in no small degree the success which has crowned this undertaking.

SOME OF THE CONTRIBUTORS TO THE MINERAL INDUSTRY.



JOHN E. ROTHWELL.



TIMOTHY W. SPRAGUE.



JOHN STANTON.



HENRY WURTZ.



W. F. WILLOUGHBY.



SOPHIA BRAEUNLICH.



TITUS ULKE.

INTRODUCTION.

THE total value of the mineral production of the United States in 1896 was $746,768,071, against $732,941,518 in 1895. Of these amounts $251,731,614 represented the value of the metals produced in 1896, against $244,290,167 in 1895, the remainder in each year representing the value of non-metallic substances. These figures are intended only to give an approximate idea of the extent of the mineral industry in the United States. They are not exact, because it is impossible to compile them in every case in precisely the same manner. The plan followed generally is to take the value of each substance at the mine, quarry, or works; but in the case of the metals this is not always possible, and consequently these are calculated at their average quotation in the principal markets, New York prices being used in most instances; pig iron, however, is calculated at the average price in Pittsburg, which is a more important market for this article than New York. It is obvious, therefore, that in these valuations a more or less proportion of money paid for transportation is included. A further source of error lies in the fact that the metals themselves are of varying quality, and do not bring the same prices; for instance, the standard copper is the metal produced from Lake Superior mineral; other kinds of copper sell for lower prices, as much as one-half cent per pound less. There are similar differences with respect to lead, zinc, and antimony. Our statistical compilations have not yet gone far enough to show precisely how much of each quality of these metals is made. We have therefore calculated all of the lead at the average price of ordinary soft lead; all of the zinc at the average price of Western spelter, though there are kinds which bring more; and all of the copper at one-fourth cent per pound less than the average price of Lake copper.

Attention also must be called to the fact that many of the crude substances reported in the table are used in the production of manufactured substances which are also reported, wherefore the same thing is counted twice; thus, all the bauxite produced in the United States is used in the manufacture of alum and aluminum. The iron ore and most of the manganese ore are used in making pig iron. Nearly all of the white lead manufactured for pigment is made from pig lead. The zinc white made in the United States, however, is derived directly from ores, and consequently does not duplicate any part of the spelter reported. A large amount of the salt production is consumed in the manufacture of soda ash, and a large

amount of limestone is burned for lime. The production of coke is of course derived from coal. All of the copper sulphate is made from metallic copper, and all of the antimony ore reported is smelted to antimony metal. Making the deductions to eliminate these duplications the net value of the mineral and metallurgical production in 1896 was $690,512,682 against $672,676,901 in 1895.

We have included iron ore in our tables, though we do not report lead, copper, zinc, silver, or gold ore separately. This admitted inconsistency is in part due to the fact that the same ores yield gold, silver, and copper, or gold, silver and lead, and in some cases also lead and zinc are obtained from the same ores, so that an exact distribution of quantities and values would be impossible and indeed of little practical value.

The production of so-called "mineral waters" is omitted, as heretofore, from these statistics. All waters contain more or less mineral matter, and, indeed, absolutely pure water is properly a mineral substance. If the production of "mineral waters" is to include all the water sold, then the supplies of our town and city water works should be counted, and they amount to over 1,000,000,000,000 gals., or say 42,100,000,000 short tons a year, sold at an aggregate value of about $200,000,000; while our consumption of ice, natural and artificial, should not be omitted. Moreover, the vast amount of water sold for irrigating, for power, or other purposes, and not included in the above, should not be neglected, while a comparatively infinitesimal amount, largely artificial mixtures, varying from carbon dioxide water to "ginger ale," is cited by some authorities as the production of "mineral waters."

The totals given in the table are, of course, of values only, since it would be useless to present a total of quantities made up of such various products. In studying the table, however, it must be remembered that to make the comparison a fair one the quantities of the various articles produced should be considered.

The statistics given, like all those in THE MINERAL INDUSTRY, have been obtained, as a rule, directly from the producers themselves, and the greatest care has been taken in their collection and arrangement. It is true that no statistics of this kind, representing so many and varied products and in a country as large as the United States, can be absolutely correct, but we believe that these figures are the fullest and the most nearly accurate that have ever been obtained. In making the comparisions the figures for 1895, as published in Vol. IV. of THE MINERAL INDUSTRY, have been revised wherever necessary in accordance with our practice, which is to present the latest and best information. Those using these statistics should therefore commence with the latest volume, where the latest and most accurate figures are recorded.

We have given the production of the different articles in metric tons (or kilograms in the case of precious metals), as well as the customary measures, for the reason that the metric measures are those recognized and used by almost all of the civilized world, and are rapidly gaining recognition in the few remaining countries which have not yet fully adopted them. In our own country they are already legalized, with the prospect of their compulsory and exclusive use at an early date. How desirable such a change will be can be best appreciated by those who have had occasion to collect or use statistics of this kind. Since the appearance of Vol. IV. the metric system has been adopted by our neighbor, Mexico.

UNITED STATES MINERAL AND METAL PRODUCTION.

Number.	Products.	Customary Measures.	1895. Quantity. Customary Measures.	1895. Quantity. Metric Tons.	1895. Value at Place of Production. Totals.	1895. Value at Place of Production. Per M. Ton.	1896. Quantity. Customary Measures.	1896. Quantity. Metric Tons.	1896. Value at Place of Production. Totals.	1896. Value at Place of Production. Per M. Ton.
	NON-METALLIC.									
	Abrasives:									
1	Carborundum	Sh. T.	113	102	$67,800	*j*$0.66	595	539	$365,612	*j*$0.68
2	Corundum	Sh. T.	385	349	53,900	154.44	250	227	35,000	154.18
3	Emery	Sh. T.	1,700	1,542	119,000	77.17	1,550	1,406	108,500	77.17
4	Garnet	Sh. T.	877	795	39,465	49.65	2,842	2,578	96,628	37.48
5	Grindstones	Sh. T.	36,389	33,012	290,378	8.79	31,301	28,395	294,338	10.36
6	Millstones	.....	.....	.....	15,925	.....	.....	.....	14,852	.....
7	Tripoli & inf. earth	Sh. T.	1,783	1,617	26,049	16.11	2,425	2,199	31,028	14.11
8	Whetstones	.....	.....	.....	93,303	.....	.....	.....	105,201	.....
9	Alum	Sh. T.	117,900	106,959	3,537,000	33.07	77,700	70,489	2,331,000	33.07
10	Antimony ore	Sh. T.	1,083	982	37,905	38.60	150	136	4,750	34.92
	Asbestos and talc:									
11	Asbestos	Sh. T.	664	602	11,837	19.66	716	650	12,670	19.49
12	Soapstone	Sh. T.	22,133	20,079	342,298	17.04	24,319	22,062	265,806	12.05
13	Talc, fibrous	Sh. T.	40,000	36,288	320,000	8.82	45,000	40,824	315,000	7.71
14	Asphalt	Sh. T.	25,525	23,156	170,500	7.36	20,500	18,597	66,900	3.59
15	Asphaltic limestone	Sh. T.	5,550	5,035	16,650	3.31	3,119	2,829	8,714	3.08
16	Bitumin's sandstone	Sh. T.	42,374	38,441	139,945	3.64	53,119	48,189	138,714	2.88
17	Barytes	Sh. T.	20,255	18,375	99,020	5.39	21,900	19,867	87,600	4.41
18	Bauxite	L. T.	18,800	19,100	56,400	2.95	17,096	17,369	68,384	3.93
19	Borax	Lbs.	13,506,356	6,126	742,850	*j*.12	15,258,014	6,921	762,900	*j*.11
20	Bromine	Lbs.	394,854	179	102,662	*j*.57	550,285	249	143,074	*j*.57
21	Calcium carbide	.....	.....	.....	.....	.....	.....	3,000	180,000	60.00
22	Cement, nat. hydraul.	Bbls[a]	7,694,053	1,046,999	4,597,285	4.39	7,454,611	1,014,417	4,353,377	4.29
23	Cement, Portland	Bbls[b]	749,059	135,904	1,430,089	10.52	1,032,654	187,363	1,710,151	9.12
24	Clay, refractory	Sh. T.	3,750,000	3,401,977	4,500,000	1.32	4,000,000	3,628,775	4,800,000	1.32
25	Clay, china	Sh. T.	30,910	28,041	258,431	9.22	29,418	26,683	217,551	8.15
26	Clay, common	Sh. T.	.....	.....	*i*60,100,000	.....	.....	.....	*i*65,000,000	.....
27	Coal, anthracite	Sh. T.	51,897,297	47,080,919	80,250,652	1.70	48,855,563	44,321,476	88,105,837	1.98
28	Coal, bituminous (*h*)	Sh. T.	137,086,346	124,363,917	123,547,936	1.00	137,331,048	124,585,909	114,930,647	.92
29	Coal, cannel	Sh. T.	69,747	63,274	191,804	3.03	54,660	49,587	146,488	2.95
30	Coke	Sh. T.	12,493,464	11,333,996	19,018,276	1.68	10,359,584	9,398,153	17,271,871	1.84
31	Cobalt oxide	Lbs.	6,400	*j*2,903	8,640	*j*2.98	12,825	*j*5,817	16,672	*j*2.87
32	Copperas	Sh. T.	14,118	12,807	69,846	5.45	11,170	10,133	52,662	5.19
33	Copper sulphate	Lbs.	45,000,000	20,412	1,750,000	85.73	48,732,840	22,105	1,949,313	88.18
34	Chrome ore	L. T.	1,553	1,578	16,795	10.64	702	713	7,775	10.90
35	Feldspar	L. T.	22,195	22,550	104,082	4.61	22,199	22,554	112,829	5.00
36	Fluorspar	Sh. T.	4,000	3,628	24,000	6.61	4,000	3,628	24,000	6.61
37	Gilsonite (*k*)	Sh. T.	1,150	1,043	34,500	33.08	1,700	1,542	54,500	35.34
38	Graphite, crystalline	Lbs.	377,450	*j*171,210	17,286	*j*.10	405,006	*j*183,709	18,225	*j*.10
39	Graphite, amorphous	Sh. T.	840	762	4,700	6.17	574	520	3,850	7.40
40	Gypsum	Sh. T.	298,572	270,863	974,219	3.59	231,649	210,150	739,799	3.52
41	Iron ore	L. T.	15,988,000	16,243,808	27,979,000	1.72	14,650,000	14,884,400	28,567,500	1.92
42	Lime	Bbls[c]	*i*60,000,000	5,443,164	30,000,000	5.05	*i*60,000,000	5,443,164	30,000,000	5.05
43	Magnesite	Sh. T.	2,200	1,995	14,700	7.37	2,067	1,875	13,435	7.16
44	Manganese ore	L. T.	170,509	173,237	328,107	1.89	162,526	165,126	339,083	2.05
45	Mica, ground	Lbs.	740,000	335	31,956	95.40	877,000	398	37,711	94.75
46	Mica, sheet	Lbs.	6,200	*j*2,812	6,400	*j*2.27	8,000	*j*3,629	8,405	*j*2.31
47	Mineral wool	Sh. T.	6,989	6,340	74,962	11.92	5,953	5,400	61,714	11.43
48	Monazite	Lbs.	1,900,000	862	114,000	132.25	17,500	8	875	109.37
49	Natural gas	.....	.....	.....	12,000,000	.....	.....	.....	10,000,000	.....
50	Paints, mineral	Sh. T.	47,084	42,714	1,086,767	25.45	54,153	49,127	973,268	19.81
51	Paints, vermilion	Sh. T.	118	107	118,190	1104.58	96	87	94,677	1088.25
52	Paints, white lead	Sh. T.	92,939	84,314	8,768,656	104.00	95,068	86,246	7,802,267	90.46
53	Paints, zinc oxide	Sh. T.	22,690	20,584	1,588,300	77.16	16,799	15,240	1,259,925	82.67
54	Petroleum (crude)	Bbls[d]	52,961,943	7,215,896	47,624,913	6.60	61,396,394	8,364,631	56,963,137	6.81
55	Phosphate rock	L. T.	1,098,017	1,115,585	3,296,757	2.96	863,754	877,574	2,591,262	2.95
56	Marls	L. T.	217,700	221,183	587,790	2.66	155,000	157,480	418,500	2.65
57	Precious stones	.....	.....	.....	250,000	.....	.....	.....	200,000	.....
58	Pyrites	L. T.	107,731	109,455	342,587	3.13	117,782	119,666	297,853	2.49
59	Salt, evaporated	Bbls[e]	12,091,265	1,535,591	5,643,383	3.61	11,156,270	1,416,846	4,828,179	3.41
60	Salt, rock	Bbls[e]	2,184,589	277,443	828,954	2.98	2,198,303	279,184	711,919	2.55
61	Silica, sand & quartz	L. T.	523,640	532,018	553,128	1.04	720,399	731,925	1,076,038	1.47
62	Slate, roofing	Sq'es.	752,606	.....	2,494,213	*n*3.31	699,100	.....	2,260,862	*n*3.24
63	Slate, manufactures	.....	.....	.....	392,877	.....	.....	.....	467,575	.....
64	Soda, natural	Sh. T.	1,900	1,724	47,500	27.55	3,000	2,722	65,000	23.88
65	Soda, manufactured	M. T.	.....	167,000	3,841,000	23.00	.....	158,975	3,656,425	23.00
66	Stone, limestone (flux)	L. T.	4,723,154	4,798,724	1,889,262	.39	4,311,563	4,380,548	1,724,625	.39
67	Stone, marble	.....	.....	.....	2,888,114	.....	.....	.....	2,600,429	.....
68	Stone, onyx	Cu. ft	1,200	.....	12,000	*p*10.00	3,000	.....	24,000	*p*8.00
69	Stone, oölitic	Cu. ft	4,790,422	.....	1,005,192	*p*.21	4,040,999	.....	755,891	*p*.18
70	Other building stones	.....	.....	.....	26,269,943	.....	.....	.....	27,219,484	.....
71	Sulphur	L. T.	1,650	1,676	41,250	24.60	2,800	2,845	64,200	22.57
72	Est. prod. unspecified	.....	.....	.....	5,000,000	.....	.....	.....	5,000,000	.....
	Total non-metals	.....	.....	.....	488,651,351	.....	.....	.....	495,036,457	.....

UNITED STATES MINERAL AND METAL PRODUCTION—*Continued.*

Number.	Products.	Customary Measures.	1895. Quantity. Customary Measures.	1895. Quantity. Metric Tons.	1895. Value at Place of Production. Totals.	1895. Value at Place of Production. Per M. Ton.	1896. Quantity. Customary Measures.	1896. Quantity. Metric Tons.	1896. Value at Place of Production. Totals.	1896. Value at Place of Production. Per M. Ton.
	METALS.									
73	Aluminum	Lbs.	900,000	*j*408,237	495,000	*j*1.21	1,300,000	*j*589,676	520,000	*j*.85
74	Antimony	Sh. T.	466	422	70,332	166.66	613	556	85,700	154.13
75	Copper (*m*)	Lbs.	386,453,850	175,294	40,616,300	231.13	467,822,973	212,203	49,729,582	234.35
76	Gold	Ozs. *f*	2,265,612	*j*70,467	46,830,200	*j*664.56	2,558,433	*j*79,576	52,886,209	*j*664.60
77	Iron, pig	L. T.	9,446,308	9,597,449	108,632,542	11.32	8,623,127	8,761,097	91,577,610	10.45
78	Lead, value at N. Y.	Sh. T.	156,854	142,297	10,132,768	71.20	174,792	158,570	10,381,843	65.47
79	Platinum	Ozs. *f*	150	*j*.4.66	2,250	482.83	200	*j*.6.21	2,800	450.89
80	Quicksilver	Flks*g*	33,978	1,179	1,313,589	1114.15	33,180	1,151	1,227,660	1066.60
81	Silver, comm'l value	Ozs. *f*	46,331,235	*j*1,441,051	30,254,296	*j*20.99	58,488,810	*j*1,819,208	39,245,991	*j*21.57
82	Zinc	Sh. T.	81,858	74,261	5,942,890	80.03	77,637	70,432	6,074,219	86.24
	Total metals	.....	.....	.....	244,290,167	.....	.....	.....	251,731,614	.....
	Grand totals	.....	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) Barrels of 300 lbs.; (*b*) 400 lbs.; (*c*) 200 lbs.; (*d*) 42 gals.; (*e*) 280 lbs. (*f*) Troy ounces. (*g*) Flasks of 76½ lbs. (*h*) Bituminous coal includes brown coal and lignite. The anthracite production is the total for Pennsylvania, Arkansas, and Colorado. (*i*) Estimated. (*j*) Kilograms or per kilogram. (*k*) Including bitumen from Texas. (*m*) The value of the copper production is calculated at 0.25c. per lb. less than the average price of Lake copper at New York. (*n*) Value per square. (*p*) Value per cubic foot.

Abbreviations: Sh. T., short tons (2000 lbs.); L. T., long tons (2240 lbs.); M. T., metric tons (2204.6 lbs.); Sq'es, squares (100 sq. ft., lapped and laid).

Monthly Production of Metals.—As it is a matter of much interest and importance to many producers to be able to follow the fluctuations of production and prices as closely as possible, we have given in the following table the monthly production and average prices of the leading metals. In this table the prices given of silver, copper, lead, and spelter are based on the New York quotations, while those of pig iron are the average of the selling prices in Pittsburg, the leading primary iron market.

MONTHLY PRODUCTION FROM DOMESTIC AND FOREIGN ORES AND BULLION, AND PRICES OF THE CHIEF METALS IN THE UNITED STATES.

1895.

Metals.	January.	February.	March.	April.	May.	June.	July.
Gold, ozs	169,669	161,631	186,249	186,164	206,999	227,239	223,754
Silver, ozs	6,244,825	6,378,921	6,658,842	5,944,042	6,603,208	6,311,144	5,632,613
Price per oz	59.69c.	59.90c.	61.98c.	66.61c.	66.75c.	66.64c.	66.75c.
Copper, long tons (*a*)	11,800	12,735	14,248	13,669	14,995	12,856	13,749
Price per lb	10.00c.	10.00c.	9.75c.	9.75c.	10.25c.	10.63c.	11.25c.
Lead, short tons	18,333	17,186	14,251	17,856	18,290	18,983	19,402
Price per lb	3.10c.	3.12c.	3.12c.	3.08c.	3.16c.	3.25c.	3.25c.
Spelter, short tons (*a*)	7,006	6,355	7,491	7,081	6,508	6,353	6,344
Price per lb	3.28c.	3.20c.	3.23c.	3.30c.	3.50c.	3.65c.	3.75c.
Pig iron, long tons (*a*)	762,474	641,550	688,170	666,315	664,600	664,449	730,156
Price, ton, gray forge	$9.15	$9.15	$9.08	$9.32	$9.70	$10.70	$11.40
Price, ton, Bes'mer pig	9.96	10.12	10.22	10.70	11.30	12.60	14.13

Metals.	August.	September.	October.	November.	December.	Totals. Cust. Meas.	Totals. Metric Tons
Gold, ozs	236,594	221,618	228,264	218,056	216,701	2,482,937	*b*77,229
Silver, ozs	6,532,103	6,159,011	6,715,977	6,584,380	6,672,005	76,437,071	*b*2,377,439
Price per oz	66.61c.	66.90c.	67.64c.	67.40c.	66.47c.	65.28c.	.....
Copper, long tons	15,535	15,496	16,270	16,485	14,696	172,534	175,294
Price per lb	12.00c.	12.25c.	12.00c.	11.00c.	10.50c.	10.76c.	.....
Lead, short tons	19,983	17,048	21,622	23,500	21,145	227,599	206,476
Price per lb	3.50c.	3.35c.	3.33c.	3.25c.	3.22c.	3.23c.	.....
Spelter, short tons	6,827	6,812	7,310	6,948	6,823	81,858	74,261
Price per lb	4.15c.	4.30c.	4.10c.	3.55c.	3.49c.	3.63c.	.....
Pig iron, long tons	780,855	899,400	940,630	979,933	1,027,776	9,446,308	9,597,449
Price, ton, gray forge	$12.10	$13.48	$13.20	$12.70	$11.35	$10.65	.....
Price, ton, Bes'mer pig	14.95	17.60	16.25	14.60	12.35	13.03	.....

1896.

Metals.	January.	February.	March.	April.	May.	June.	July.
Gold, ozs	203,565	205,933	223,566	188,758	241,672	277,302	279,448
Silver, ozs	6,898,669	7,022,142	8,151,717	6,937,003	7,754,878	8,187,487	8,079,512
Price per oz	67.13c.	67.67c.	68.40c.	67.92c.	67.88c.	68.69c.	68.75c.
Copper, long tons	17,503	18,947	19,353	18,553	18,164	17,456	18,026
Price per lb	9.87c.	10.64c.	11.03c.	10.98c.	11.15c.	11.67c.	11.40c.
Lead, short tons	19,893	20,321	22,612	19,158	21,846	23,174	22,375
Price per lb	3.08c.	3.19c.	3.14c.	3.07c.	3.03c.	3.03c.	2.96c.
Spelter, short tons	6,680	6,689	6,525	6,355	6,391	6,371	6,360
Price per lb	3.75c.	4.03c.	4.20c.	4.09c.	3.98c.	4.10c.	3.97c.
Pig iron, long tons	945,900	800,022	882,181	835,116	821,923	691,094	751,593
Price, ton, gray forge	$12.25	$11.50	$11.25	$11.50	$11.15	$11.00	$10.75
Price, ton, Bes'mer pig	13.35	13.50	12.75	13.85	13.40	13.35	12.65

Metals.	August.	September.	October.	November.	December.	Totals.	
						Cust. Meas.	Metric Tons
Gold, ozs	256,997	256,173	282,123	260,614	291,686	2,967,737	*b*92,307
Silver, ozs	8,004,668	7,984,329	8,084,306	7,607,186	6,910,442	91,622,339	*b*2,849,777
Price per oz	67.34c.	65.68c.	65.05c.	64.98c.	65.24c.	67.10c.	.....
Copper, long tons	18,231	18,408	19,589	17,796	19,057	221,083	224,620
Price per lb	10.98c.	10.66c.	10.66c.	11.23c.	11.28c.	10.88c.	.....
Lead, short tons	21,700	22,206	22,362	18,616	20,588	254,851	231,201
Price per lb	2.73c.	2.77c.	2.80c.	2.96c.	3.04c.	2.98	.....
Spelter, short tons	6,043	6,034	7,042	6,588	6,559	77,637	70,432
Price per lb	3.76c.	3.60c.	3.72c.	3.99c.	4.14c.	3.94	.....
Pig iron, long tons	645,200	602,508	466,200	509,000	672,390	8,623,127	8,761,097
Price, ton, gray forge	$10.00	$10.16	$10.25	$10.80	$10.60	10.93	.....
Price, ton, Bes'mer pig	11.75	12.00	12.25	12.75	12.20	12.65	.....

(*a*) Domestic production only. (*b*) Kilograms.

Production of Metals from Foreign Ores.—In our large table the figures are limited strictly to our domestic production. In the cases of gold, silver, copper, and lead the representation is an altogether inadequate one, for large quantities of those metals are obtained from foreign ores and base bullion smelted or refined here. The particulars are given fully in the articles on the different metals, and we add here a supplementary table, showing the total amount of this production in 1895 and 1896. This is necessary in order to show fully and fairly the work done by our metallurgical establishments.

METALS PRODUCED FROM FOREIGN ORES AND BULLION IN 1895 AND 1896, AND INCLUDED IN THE PRECEDING TABLE.

Metals.	1895.				1896.		
	Customary Measure.	Quantities.		Values.	Quantities.		Values.
		Customary Measure.	Metric Tons or Kgms.		Customary Measure.	Metric Tons or Kgms.	
Copper	Pounds	.....	.....	.....	27,603,520	Tons 12,323	$3,003,273
Gold	Troy ozs	217,234	Kgms 6,761	4,490,227	409,315	Kgms 12,731	8,461,023
Lead	Short tons	70,745	Tons 64,179	4,570,127	80,159	Tons 72,266	4,769,780
Nickel	Pounds	3,880,000	Tons 1,760	970,000	3,697,039	Tons 1,677	1,920,022
Silver	Troy ozs	30,105,836	Kgms. 936,388	19,659,111	33,133,529	Kgms 1,030,569	22,232,598
Total values	.....	.....	.....	29,689,465	.....	.....	39,686,696

These metals were obtained chiefly from material received from Mexico and British North America. In the case of our northern neighbor the smelting ores from the mines are usually sent to our smelters at Tacoma, Puget Sound, San Francisco, and elsewhere. From Mexico we receive chiefly base bullion, from which a very considerable part of our lead supply is obtained, as noted elsewhere.

The nickel is all from ores or matte received from the Sudbury mines in Ontario. We have not included above the iron smelted from foreign ores, which is small in actual amount, and insignificant in comparison with the total output. The lead industry is the trade most affected by the use of foreign material, the quantity of copper being small in comparison with the total output. In the case of nickel scarcely any ores of that metal are now produced in the United States.

METALS.

Aluminum.—The production increased from 900,000 lbs. ($495,000) in 1895 to 1,300,000 lbs. ($520,000) in 1896. The Pittsburg Reduction Company is still the only producer, though the litigation which closed the competitive works is not yet finally settled. During the year the Pittsburg Reduction Company put in operation its new works at Niagara Falls, N. Y.

Antimony.—The domestic production of antimony ore declined from 1083 short tons ($37,905) to 150 short tons ($4750), all of which came from California. American antimony ore is mostly subject to so high freight charges that it cannot be mined profitably at present prices. The production of antimony metal was 466 short tons ($70,332) in 1895 and 613 short tons ($85,790) in 1896. A small amount of this was smelted in California, but the more part was produced in New York from imported ore.

Copper.—The domestic production increased from 386,453,850 lbs. in 1895 to 467,822,973 lbs. in 1896, Arizona, Michigan, and Montana each showing a large gain. The domestic consumption fell off from 270,392,841 lbs. to 216,483,123 lbs., but the exports increased from 121,703,978 lbs. to 281,905,217 lbs., which maintained the market in good condition. The average price of the standard Lake copper was 10.88c. per lb. against 10.76c. in the previous year.

Gold.—The production in 1896 was 2,558,422 ozs. ($52,886,209) against 2,265,612 ozs. ($46,830,200) in 1895. This great increase was due to greater activity in most of the Rocky Mountain districts, nearly every State showing gains, which were especially large in Alaska, Arizona, California, and Colorado. The most productive single district (if the Mother Lode of California, which extends through a long succession of districts, be excepted) was Cripple Creek, where the mines yielded $9,099,300. Next in rank is the Black Hills, to which the total production of South Dakota, namely, $4,919,800, may be credited. Some important new mines were discovered in Arizona. The indications are for a further increase in gold production in 1897.

Iron.—The production of pig iron decreased from 9,446,308 long tons in 1895 to 8,623,127 long tons in 1896. The average price of iron was also less than in the previous year. A noteworthy feature of the trade, however, was the exportation of pig iron from Birmingham, Ala., to Europe, which, together with the increased exports of American steel, caused some consternation in the foreign markets. The production of steel rails in the United States fell off in somewhat the same proportion as pig iron.

Lead.—The production of pig lead increased from 156,854 short tons to 174,792, notwithstanding the lower average price and the strike in Leadville, which cut off the production of those mines for nearly six months. The increase is attributable chiefly to the Cœur d'Alêne, Idaho, and to Southeastern Missouri,

where the mines are capable of producing large quantities of metal at a very low cost.

Platinum.—The production of this metal amounted to 200 ozs. in 1896, practically all of it being obtained in refining bullion at the Philadelphia and San Francisco mints, especially the latter.

Quicksilver.—There was no material change in this industry, California remaining the only producer. The output was 33,180 flasks, against 33,978 in 1895. Quotations for the metal were lower in 1896 than in the previous year.

Silver.—The production increased from 46,331,235 ozs. in 1895 to 56,222,322 ozs. in 1896. This great increase was foreshadowed by the heavy increases in the outputs of lead, copper, and gold, since the more part of the American silver is obtained in association with those metals. The Anaconda copper mine of Montana is the largest producer in this country, its outturn exceeding 5,000,000 ozs. per annum, while the aggregate yield of the silver-copper mines of Butte was 10,614,003 ozs., or nearly 20% of the total silver outturn of the United States.

Zinc.—The production of zinc declined from 81,858 short tons in 1895 to 77,637 short tons in 1896, notwithstanding the average price was higher in the latter year. The restriction in output was largely due, however, to the policy of the company, organized early in the year, which absorbed a large number of smaller concerns. The year was also noteworthy for the large exports of zinc to Europe and the beginning of an important export trade in zinc ore.

ORES AND NON-METALLIC SUBSTANCES.

Abrasives.—Under this caption are included carborundum, corundum, emery, garnet, grindstones, millstones, tripoli and infusorial earth, and whetstones. The production of carborundum increased from 113 short tons ($67,800) in 1895 to 595 short tons ($365,612) in 1896. The company producing this substance now has works at Niagara Falls, where facilities are much better than they were in the old works at Pittsburg, Pa. The production of corundum declined from 385 short tons ($53,900) to 250 short tons ($35,000); that of emery from 1700 short tons ($119,000) to 1550 short tons ($108,500). Neither of these industries seems to be especially prosperous in the United States. The output of garnet, however, increased from 877 short tons ($39,465) to 2842 short tons ($96,628), although the average value of this substance was a good deal less in 1896 than in the previous year. The output of grindstones decreased from 36,289 short tons ($290,378) to 31,301 short tons ($294,338); that of millstones declined from $15,925 to $14,852, while whetstones increased from $93,303 to $105,201. The output of tripoli and infusorial earth increased from 1783 short tons ($26,049) to 2425 short tons ($31,028). Not all of this amount of tripoli, however, is used as an abrasive, a good deal being employed as a non-conductor for protecting steam-pipes, etc., as a filtering medium, and for various other minor purposes.

Alum.—The production decreased from 117,900 short tons ($3,537,000) to 77,700 short tons ($2,331,000). These figures represent the production of crystallized alum and sulphate of aluminum calculated in terms of crystallized alum. In reality the production of sulphate of aluminum is now far more important in the United States than that of alum proper, the two substances being made

approximately in the proportion of 3 : 1. Since sulphate of aluminum contains more alumina than alum, the actual tonnage produced is less than the figures quoted.

Asbestos and Talc.—The production of asbestos increased from 664 short tons ($11,837) to 716 short tons ($12,670). Nearly all of the output in 1896 came from the Sall Mountain mine in Georgia, but a little was mined in Washington. The Georgia mineral is true asbestos, differing mineralogically from the Canadian, which is classed as chrysotile. It is generally short-fibered and consequently of comparatively low value. The production of soapstone or massive talc increased from 22,133 short tons ($342,298) to 24,319 short tons ($265,806). The output of this substance comes chiefly from Pennsylvania, North Carolina, and Georgia. The production of fibrous talc increased from 40,000 short tons ($320,000) to 45,000 short tons ($315,000). This is entirely the production of the mines in St. Lawrence County, N. Y., and with the exception of a few tons is wholly consumed in paper-making.

Asphaltum.—The production decreased from 25,525 short tons ($170,500) to 20,500 short tons ($66,900). The falling off in the value per ton is especially noticeable. These figures represent only the production of California, the production of gilsonite in Utah and bitumen in Texas being reported separately. The asphaltum industry in California promises to assume greater importance, since there is prospect that some of its product will be exported in 1897.

Asphaltic Limestone.—The production decreased from 5550 short tons ($16,650) to 3119 short tons ($8714). This substance was produced in the two States of Utah and Texas.

Bituminous Sandstone.—The production increased from 42,374 short tons ($139,945) to 53,119 short tons ($138,714). This rock is mined in California, Kentucky, and Utah.

Barytes.—The production increased from 20,255 short tons ($99,020) to 21,900 short tons ($87,600). The barytes grinding industry was much depressed in 1896, and several small mills were obliged to close down.

Bauxite.—The production decreased from 18,800 long tons ($56,400) to 17,096 long tons ($68,384). The falling off was due especially to the comparatively small manufacture of alum, and would have been still more had it not been for the increased demand from the aluminum manufacturers. The high quality of American bauxite is now generally recognized. It has already practically entirely displaced foreign bauxite in the domestic market, and last year preparations were made for beginning an export trade in the mineral.

Borax.—The production increased from 13,506,356 lbs. ($742,850) to 15,258,014 lbs. ($762,900). A part of this output was converted into boracic acid in the United States, which is a new industry for this country. A very important event of the year was the consolidation of the Pacific Coast Borax Company with the Redwood Chemical Works of England. This combination now practically controls the borax trade of the world.

Bromine.—The production increased from 394,854 lbs. ($102,662) to 550,285 lbs. ($143,074). The market was quiet, the production of this substance being still controlled by a combination of the producers.

Calcium Carbide.—The manufacture of this substance was begun in the United

States in 1886, and has already become an important industry. During the year there were four works in operation: one at Sault Ste. Marie, Mich., one at Appleton, Wis., one at Niagara Falls, N. Y., and one at Lockport, N. Y. These works were unable to supply the demand for the carbide, and each of them has been enlarged since January 1, 1897, or is to be enlarged during the current year, while a new works of much greater capacity than any of them is being erected in West Virginia. The production of the four works in operation in 1896 was 3000 metric tons, valued at $180,000. The value of calcium carbide at the works ranged from $55 to $65 per short ton; small lots, *i.e.*, less than one ton, put up in packages ready for shipment sold at the rate of $75 per ton. A large amount of calcium carbide was exported to Europe, where the demand was far greater than the European works could supply. The only works in actual operation in Great Britain at present are those of the Acetylene Company at the Falls of Foyers, Scotland, in conjunction with the British Aluminum Company. The Willson system is used at these works. A Liverpool syndicate has taken up the Pietet process and proposes to erect works. The Patin process is being successfully applied in Paris and is about to be introduced in Belgium. In Switzerland carbide is being made at the great Rheinfelden works.

Cement, Natural Hydraulic.—The production decreased from 7,694,053 bbls. ($4,597,285) to 7,454,611 bbls. ($4,353,377). This industry was very much depressed; prices were low and few producers are said to have made any money. The outlook for the future is also gloomy on account of the headway that is being made by artificial, or Portland, cement.

Cement, Portland.—The production increased from 749,059 bbls. ($1,430,089) to 1,032,654 bbls. ($1,710,151). Prices were a little lower than in the previous year, but the industry was prosperous nevertheless. Several new works were erected and improvements were made in the equipment of others. American Portland cement is displacing the foreign, of which the imports into this country are decreasing, and is also displacing the natural hydraulic. The outlook is for a further great increase in production in 1897.

Clay, Refractory.—The production increased from 3,750,000 short tons ($4,500,000) to 4,000,000 short tons ($4,800,000). The consumption of refractory clay is chiefly in the manufacture of firebrick, and as mortar for laying the same, but a good deal is used in making zinc retorts, glass pots, melting crucibles, etc. Included in this category also are the ball clays used by potters.

Clay, China.—The production decreased from 30,910 short tons ($258,431) to 29,418 short tons ($217,551). This product is obtained by washing crude kaolin. It is mostly used by the East Liverpool, Ohio, and Trenton, N. J., potters, but a little is consumed in Colorado as a filler in paper manufacture.

Clay, Common.—The value of the production is estimated to have increased from $60,100,000 to $65,000,000. On account of the manner in which this industry is conducted it is very difficult to arrive at the exact statistics of the production, which is probably underestimated.

Coal, Anthracite.—The production decreased from 51,897,297 short tons ($80,250,652) to 48,855,563 short tons ($88,105,837). This was almost entirely the production of Pennsylvania.

Coal, Bituminous.—The production increased from 137,086,346 short tons

($123,547,936) to 137,331,048 short tons ($114,930,647). These figures include the production of brown coal and lignite. Soft coal sold in many parts of the United States at lower prices in 1896 than had ever before been accepted.

Coal, Cannel.—The production decreased from 69,747 short tons ($191,804) to 54,660 short tons ($146,488). Kentucky is the only State producing this kind of coal, which sells for about three times the average price of ordinary bituminous coal.

Coke.—The production decreased from 12,493,464 short tons ($19,018,276) to 10,359,584 short tons ($17,271,871). The coke trade being dependent chiefly on the demand from the iron furnaces, a decline in the production of pig iron means also a decline in the production of coke. Some progress was made in 1896 in the introduction of by-product coke ovens in the United States.

Cobalt Oxide.—The production increased from 6400 lbs. ($8640) to 12,825 lbs. ($16,672). This oxide is made by only one works in the United States, the raw material being chiefly speiss from Mine La Motte and Canadian nickel-copper matte.

Copperas.—The production decreased from 14,118 short tons ($69,846) to 11,170 short tons ($52,662). This vitriol is produced in the United States entirely as a by-product by the wire-drawers, who recover it from their waste pickling-liquors.

Copper Sulphate.—The production increased from 45,000,000 lbs. ($1,750,000) to 48,732,840 lbs. ($1,949,313). A good deal of this vitriol is made as a by-product by gold and silver refiners.

Chrome Ore.—The production decreased from 1553 long tons ($16,795) to 702 long tons ($7775). This was entirely the production of California, which State has large deposits of the ore. For the more part, however, they are remote from railways and freight to market is so heavy that there is not much chance for an increased production in competition with the foreign chrome ore that can be brought more cheaply to the Atlantic coast.

Feldspar.—The production in 1895 was 22,195 long tons ($104,082); in 1896 it was four tons greater, but the value increased to $112,829. The output of feldspar is used almost entirely by the potters of East Liverpool, Ohio, and Trenton, N. J.

Fluorspar.—The production was 4000 short tons ($24,000) in 1895 and for 1896 it is estimated at the same figures. This substance was produced in 1896 by only one mine, which is situated in Southern Illinois.

Gilsonite.—The production increased from 1150 short tons ($34,500) to 1700 short tons ($54,500). Included in these figures is the output of bitumen extracted from the asphaltic limestone, the so-called "litho-carbon," in Texas. The gilsonite proper is entirely the product of Utah.

Graphite.—The production of crystalline graphite increased from 377,450 lbs. ($17,286) to 405,006 lbs. ($18,225). This was as in previous years entirely the production of Ticonderoga, N. Y. The production of amorphous graphite in Rhode Island decreased from 840 short tons ($4700) to 574 short tons ($3850).

Gypsum.—The production decreased from 298,572 short tons ($974,219) to 231,649 short tons ($739,799).

Iron Ore.—The production decreased from 15,988,000 long tons ($27,979,000)

to 14,650,000 ($28,567,500), the decrease corresponding to the falling off in the production of pig iron. It is noteworthy that the average value of iron ore per ton was higher in 1896 than in 1895, while that of pig iron was lower.

Lime.—The production in 1896 is estimated at the same figures as in the previous year, namely, 60,000,000 bbls. ($30,000,000). This is equivalent to 6,000,000 short tons, which indicates a consumption of about 12,000,000 short tons of limestone for this purpose.

Magnesite.—The production decreased from 2200 short tons ($14,700) to 2067 short tons ($13,435). California is the only State which produces this mineral. The high freights from its deposits to the markets of the Atlantic coast prevent the growth of the industry, the supply being limited to the local demands.

Manganese Ore.—The production decreased from 170,509 long tons ($328,107) to 162,526 long tons ($339,083). These figures include the high-grade manganese ores of Virginia, Georgia, and Arkansas, the lower grade manganese ores which are shipped from Leadville, Colo., to the steel works of Colorado and Illinois; also the franklinite residuum of New Jersey and the low grade manganiferous iron ores of Michigan and Wisconsin. Not included are the silver-iron-manganese ores of Colorado, which are used as fluxes in lead smelting.

Mica.—The production of ground mica increased from 740,000 lbs. ($31,956) to 877,000 lbs. ($37,711); that of sheet mica from 6200 lbs. ($6400) to 8000 lbs. ($8405). The domestic mica industry is not profitable, and our market continues to be supplied chiefly by the importations from India and Canada.

Mineral Paints.—The production of iron oxides, ochers, etc., increased from 47,084 short tons ($1,086,767) to 54,153 short tons ($973,268). The decrease in the value of the product is due chiefly to the lower price of white lead. The production of vermilion decreased from 118 short tons ($118,190) to 96 short tons ($94,677). The production of white lead increased from 92,939 short tons ($8,768,656) to 95,068 short tons ($7,802,267). This is produced chiefly from pig lead. The production of zinc oxide decreased from 22,690 short tons ($1,588,300) to 16,799 short tons ($1,259,925). This is entirely the production of the three works of New Jersey and Pennsylvania.

Mineral Wool.—The production decreased from 6989 short tons ($74,962) to 5953 short tons ($61,714). This substance is produced chiefly from blast-furnace slag, but one works makes it by melting granite.

Monazite.—The domestic monazite mining industry came nearly to a standstill in 1896, the production falling from 1,900,000 lbs. ($114,000) to 17,500 lbs. ($875). The reason for this is that Brazilian monazite can be delivered more cheaply in the European markets than the American.

Natural Gas.—The value of the output is estimated to have declined from $12,000,000 to $10,000,000. Since a large amount of natural gas is burned without measuring, it is obviously impossible to arrive at exact statistics.

Petroleum.—The production increased from 52,961,943 bbls. ($47,624,913) to 61,396,394 bbls. ($56,963,137). Notwithstanding this great increase in production, the average price of petroleum was higher in 1896 than in the previous year. The more part of the increase is attributable to the Ohio fields, whose oil is more and more taking the place of the Pennsylvanian in the market.

Phosphate Rock. — The production decreased from 1,098,017 long tons

($3,296,757) to 863,754 long tons ($2,591,262). The falling off was especially large in South Carolina; that it was not more in Florida is probably due to the existence of large stocks at the various mines, since our reports showed a general suspension of operations toward the close of the year. The production of Tennessee did not increase. The phosphate industry of the United States is now in a very unprofitable condition, owing to the competition of Algerian phosphates and Thomas slag in the European markets.

Marl.—The production increased from 217,700 long tons ($587,790) to 155,000 long tons ($418,500). Owing to the fact that this substance is largely dug by farmers for their own consumption, returns for it are hard to secure.

Precious Stones.—The production decreased from $250,000 in 1895 to $200,000 in 1896. In both years these values represent chiefly the output of sapphire and ruby from Montana and turquoise from New Mexico.

Pyrites.—The production increased from 107,731 long tons ($342,587) to 117,782 long tons ($297,583). The high price of brimstone, which prevailed in 1896, increased the use of pyrites for acid-making.

Salt.—The production of salt evaporated from brine decreased from 12,091,265 bbls. ($5,643,383), equivalent to 1,535,591 metric tons, to 11,156,270 bbls. ($4,828,179), equivalent to 1,416,846 metric tons. The production of rock salt increased from 2,184,589 bbls. ($828,954), equivalent to 277,443 metric tons, to 2,198,303 bbls. ($711,919), equivalent to 279,184 metric tons. A large part of the salt production of the United States is now consumed in the manufacture of soda ash, etc., and since this branch of the chemical industry is growing rapidly the probability is that the salt production of the country will increase also.

Silica, Sand, and Quartz.—The production increased from 523,640 long tons ($553,128) to 720,399 long tons ($1,076,038).

Slate.—The production of roofing slate decreased from 752,606 squares ($2,494,213) to 699,100 squares ($2,260,862). The value of slate manufactures increased from $392,877 to $467,575. The domestic slate trade was depressed in 1896 and the production would have fallen off very much more had it not been for the great increase in the export trade, which suddenly developed.

Soda.—The production of natural soda increased from 1900 short tons ($47,500) to 3000 short tons ($65,000). This was entirely the output of California. The great deposits of Wyoming are too remote to be of much immediate value. The production of soda ash, etc., from salt declined from 167,000 metric tons ($3,841,000) to 158,975 metric tons ($3,656,425).

Stone.—The production of limestone for iron flux decreased from 4,723,154 long tons ($1,889,262) to 4,311,563 long tons ($1,724,625), or in the same proportion as the falling off in the production of pig iron. The value of the marble product declined from $2,888,114 to $2,600,429. Of onyx marble there was produced in 1896 3000 cu. ft. ($24,000) against 1200 cu. ft. ($12,000) in 1895. In both years the output was entirely from California. The production of oölitic limestone, entirely that of Indiana, decreased from 4,790,422 cu. ft. ($1,005,192) to 4,040,999 cu. ft. ($755,891). Of stones not included in the foregoing the output in 1896 is estimated at $27,219,484 against $26,269,943 in 1895.

Sulphur.—The production increased from 1650 long tons ($41,250) to 2800 long tons ($64,200). Utah and Louisiana were the only producers.

ABRASIVES.*

UNDER this caption are included corundum and emery, garnet, quartz, carborundum, diatomaceous earth, tripoli, pumice, rouge, crushed steel, boart (carbonado), and abrasive stone (whetstone, buhrstone, millstone, etc.). Some of these substances have other uses, especially quartz and diatomaceous earth, which are of more importance than their use as abrasives, but for convenience they will be considered in this group.

The production of abrasives in the United States is given in the following table:

PRODUCTION OF ABRASIVES IN THE UNITED STATES.

Commercial Names.	1895.			1896.		
	Tons of 2000 lbs.	Value.		Tons of 2000 lbs.	Value.	
		Total.	Per Ton.		Total.	Per Ton.
Carborundum	113	$67,800	$600	595	$365,612	$614
Corundum	385	53,900	140	400	56,000	140
Emery	1,700	119,000	70	1,550	108,500	70
Garnet	2,065	93,350	45	2,800	94,500	34
Tripoli (a)	1,783	26,049	15	2,425	31,028	13
Totals	6,046	$360,099	.....	7,770	$655,640	.....

(a) Includes infusorial earth.

The above table does not indicate exactly the consumption of abrasives in the United States, since a considerable part of the tripoli produced (with which is included infusorial or diatomaceous earth) is used for other purposes, such as non-heat-conducting covering for steam pipes and boilers. In Europe a good deal of this substance is employed in the manufacture of dynamite, but very little is consumed for this purpose in the United States.

The United States imports neither corundum nor garnet, but receives a good deal of emery from Turkey and Greece. The amount and value of these importations are shown in the following table:

IMPORTS OF EMERY INTO THE UNITED STATES.

Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.	Year.	Grains.		Ore or Rock.		Other M'f'es.	Total Value.
	Pounds	Value.	Long Tons.	Value.	Value.			Pounds	Value.	Long Tons.	Value.	Value.	
1889..	503,347	$18,937	5,234	$88,727	$111,302	$218,966	1893..	516,953	$20,073	5,066	$103.875	$3,819	$127.767
1890..	534,968	20,382	3,867	97,939	5,046	123,367	1894..	597,713	18,645	2,804	51,487	1,847	71,973
1891..	90,658	3,729	2,530	67,573	.....	71,302	1895..	678,761	25,066	6,803	80,386	27,586	133,038
1892..	566,448	22,586	5,280	95,625	2,412	120,623	1896..	751,464	26,520	6,289	119,667	1,971	148,158

*For further data relating to infusorial earth and tripoli and the manufacture of carborundum refer to Vol. II.; for the same and corundum, emery, crushed steel, rouge, and garnet refer to Vol. III.

PRODUCTION OF WHETSTONES, SCYTHESTONES, AND GRINDSTONES IN THE UNITED STATES.

Kind of Stone.	Measures.	1894.		1895.		1896.	
		Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Oilstones, etc.:							
Hindostan (finished product)	Pounds	300,000	$9,000	300,000	$6,250	275,000	$10,000
Indiana sandstone (finished product)	Pounds	150,000	3,375	100,000	7,500		
Arkansas stone ("rough rock")	Pounds	100,000	4,000	460,000	10,800		
Washita stone ("rough rock")	Pounds	250,000	3,125	203,200	10,614	a340,000	50,000
Scythestones:							
New Hampshire (Indian Pool, etc.)	Gross	11,000	30,250	12,000	33,000	15,000	35,000
Vermont (Lamoille, etc.)	Gross	4,700	16,450	3,500	12,250	907	3,176
Ohio and Michigan grit	Gross	7,000	19,250	4,814	12,889	4,129	7,025
Grindstones:							
Ohio and Michigan	Short tons.	29,989	257,596	36,389	290,378	b31,301	294,338

(*a*) Finished product in 1896. The product in 1894 and 1895 was "rough rock." (*b*) Michigan had no output in 1896.

Carbonado.

Carbonado is a black variety of the diamond. It is an opaque, massive mineral with a crystalline, sometimes granular or compact structure, resinous to adamantine luster, and hardness equal to or slightly in excess of the crystalline diamond. It is found chiefly in the Province of Bahia, Brazil. It finds its chief use in gem-cutting and in the bits of diamond-drills. H. Moissan, in *Comptes Rendus*, CXXI., pp. 449, 450, describes the huge specimen of carbonado which was found in a diamond-bearing district of Brazil on July 15, 1895. It weighed 630 gms., or 3073 carats, and was the largest piece ever found, the next largest on record being one of 1700 carats. It is of a good black color, its surface in places smooth, in places pimply. Carbonado of good quality is worth about $12.65 per carat. Carbonado is often referred to in the trade as "bort," which is properly a distinct variety of crystallized carbon, standing between the diamond and carbonado; it is employed for the same purposes as the latter.

Carborundum.

The Carborundum Company reports that its works produced during 1896, in round numbers, 1,191,000 lbs., of crystalline carborundum. Consideration at the present is given to the production in crystalline form only, but another important industry into which carbide of silicon promises to be a valuable adjunct is under consideration, it having been demonstrated by experiment that carborundum can be used, and will in all probability take the place of ferrosilicon in the manufacture of steel. Professor Luehrmann, of Germany, recently wrote an article on this subject, indicating that in this use of carborundum there will be in Germany alone approximately 2500 tons consumed annually, provided its cost does not exceed 6c. per lb. It may be used for this purpose in an amorphous form, and the Carborundum Company states that it is prepared to furnish it at a price slightly under this figure.

The Carborundum Company has now decided to duplicate the present plant at once, which will require an addition of 1000 electrical horse-power. Although it is not likely that this power can be secured for several months yet, preparations are already being made for building the new electrical furnaces. It is also reported that a new company will soon begin the erection of a factory at Niagara Falls for the manufacture of carborundum paper and cloth.

Corundum.

By Charles N. Jenks.

Prior to 1871 commercial corundum was obtained from various portions of the Old World, principally from the gem localities of Burma, Siam, and Ceylon; occasionally from the Kashmir Himalayas, from the Malabar coast, from the Ourals, China, Japan, Siberia, Syria, Armenia, Hindostan, Afghanistan, Bohemia, Austria, Sweden, France, and from all the emery-producing districts of Asia Minor and the Grecian islands. From these and other foreign localities corundum found its way into the market, chiefly in the form of hexagonal crystals and crystalline aggregates, in quantity sufficient for gem cutting and laboratory tests, and to a very limited extent as an abrasive agent where great efficiency was required and expense was not considered. For certain uses, such as making phonograph points, gem cutting, etc., selected opaque or translucent crystals, Ceylon or Burma preferred, still command as high as $6 per oz. Many of these placer crystals have been rejected as gems in the sortings, or were picked up by the natives without any attempt at regular mining. The total production from these sources has probably never exceeded 100 tons per annum, and has usually been much less.

Prof. F. Zirkel, an eminent German authority, in enumerating (1893) the modes of occurrence of corundum as a rock constituent in the Eastern Hemisphere, says corundum in small fine-grained aggregates is the chief constituent of emery wherever found; otherwise it occurs occasionally in grains or crystals of various sizes and colors in granites, gneisses, granular limestones, and dolomites; also in the amphibolites, in chlorite schist, in graphite, and in several basalts, often with spinel, rutile, and sillimanite; also in certain localities as a contact product of the diorites. It is observed in altered foreign inclusions or as real accessory masses in certain eruptive rocks, occurring with cordierite, spinel, andalusite, and tonalite, and in a contact product of quartz-mica-diorite in quartz phyllite and in kersantite.

Prof. J. Volney Lewis of South Carolina, in his admirable work on *Corundum and the Basic Magnesian Rocks of Western North Carolina* (1896), says: "Besides these occurrences as a rock constituent, corundum is found in large quantities in feldspar veins and associated with chlorites in the peridotites and serpentines of the Atlantic States of America." He also says, "Except the occurrences in granular limestone, in graphite, and in association with volcanic rocks, all the various modes enumerated above have been observed in North Carolina."

Besides the above the present writer has also noted one association of corundum with graphite at the Bad Creek sapphire mine in Transylvania County, N. C., specimens of which he has preserved as being the only association of this kind so far noted in North Carolina.

This entire subject has been so fully treated from a geological standpoint by such eminent authorities as Smith, Shepard, Genth, Chatard, Julian, King, and the two previously quoted, that the writer will attempt no more than a brief summary of their discoveries and deductions, together with certain corrections of fact in connection with the earlier history of corundum mining in America, confining himself for the most part to the economic features of corundum and emery.

Corundum has been found in the United States in Alabama, Arizona, California, Colorado, Connecticut, Delaware, Georgia, Maine, Maryland, Massachusetts, Montana, New Jersey, New Mexico, New York, North Carolina, Pennsylvania, South Carolina, and West Virginia and also in Canada. In these States about 410 occurrences of corundum are known to the writer, of which 264 are in North Carolina. The first reported occurrence of corundum in America was a small hexagonal crystal, said to have been found in the Laurens district in South Carolina in 1819 by one John Dickson of Columbia, but the history of corundum mining begins in 1871 at Corundum Hill in Macon County, N. C., where the work was undertaken by Col. Charles W. Jenks of Boston. The work was inaugurated, not with the expectation of finding gems, as has been erroneously stated, but to find pure crystal corundum in quantity, an abrasive more efficient than emery being needed in an industry in which Colonel Jenks and his associate were directly interested. The washing for gems, which were accidentally discovered, was merely incidental to the production of a pure commercial corundum. The two mines he then purchased and opened have, under different managements and for over 25 years, furnished the American market with four-fifths of the entire product of corundum.

During the first work at Corundum Hill and Laurel Creek about 200,000 lbs. of pure corundum, mostly crystal and lump, were put on the market. For several years after the Hampden Emery and Corundum Company of Chester, Mass., obtained control there was an annual output of nearly 1000 tons. A member of the company lately informed the writer that its largest yearly output during this period of heavy production was 1300 tons. From 1893 to 1895, inclusive, the output of these mines has not much exceeded 300 tons of finished product per annum and in 1896 it dropped to 150 tons. This has not been due to any scarcity of the mineral, but on account of trade conditions.

Prospecting for Corundum.—In the States mentioned as furnishing corundum and emery only three, in the opinion of the writer, have all the necessary conditions for extensive deposits. These are North Carolina, Georgia, and Massachusetts. The Massachusetts deposits will be referred to subsequently under the caption "Emery." As the writer does not profess an accurate knowledge of the subject from a geological standpoint, the occurrence of corundum and the associate minerals, and the peculiar conditions in the peridotite formations necessary to heavy deposits of this mineral, will be briefly presented from the standpoint of the prospector and miner.

This area of heavy deposits begins in Raybun County, Ga., near the North Carolina line, and, following the course of the peridotite rocks north 45° east, includes portions of Transylvania, Jackson, Macon, and Clay Counties, N. C., being about 50 miles in length and about 15 miles in width, and comprising about 750 sq. miles. The presence of corundum is indicated by the peridotites, or, in local parlance, serpentine chrysolite or olivine rocks. Two great systems of dikes of these olivine rocks traverse the crystalline schists and gneisses of the Appalachian range. First emerging from the latter formation in Central Alabama, they outcrop at intervals along their course (north 45° east) for a distance of over 800 miles, and until Trenton, N. J., is reached, where they disappear under the younger sedimentary rocks to the northward. These systems in Georgia and

North Carolina occur, one on the north side of the Blue Ridge, the other on the south side, and have a strike parallel to the main ridge, following closely in most cases its windings at an average distance of 6 miles from its crest, which they sometimes traverse for a short distance. Each system consists of a number of parallel dikes varying from one to several miles apart, and outcropping at intervals of a few miles. These outcrops usually occur in groups, varying in the number of their exposures; in the great concentration of systems in Jackson and Transylvania counties, N. C., there are no less than 68 discovered outcrops, 40 on the southern system and 28 on the northern, all within a radius of 10 miles of Sapphire; in the Ellejah and Corundum Hill group, Macon County, there are 22 exposures. Some single outcrops are hundreds of acres in extent, although not greatly exceeding 1000 ft. in width at any point. They consist mostly of scattered lenticular-shaped boulders, frequently of gigantic proportions.

From the main dikes on either side of the Blue Ridge run occasional cross and blind dikes. The former in some localities form networks with either system; the latter run parallel with the main dikes for short distances, or branch off at right angles, terminating abruptly. The dike systems are sometimes extremely puzzling even to the experienced prospector, but a knowledge of their peculiarities is of the utmost importance, since thousands of dollars have been wasted upon the cross and blind dikes which carry no deposits of value. Although no reference has ever been made to it in corundum literature, it is a significant fact, as a guide to the prospector, that all large deposits of either corundum or emery in either hemisphere have been found on or near the crests of the main chain of mountains and never in the lowlands. The dike systems do not, however, cross the main chain except at rare intervals. One or two instances occur where a single dike will reach the crest, but in these cases perhaps after looping around a high peak it will return to its proper side. A study of these dike systems from a geological standpoint (such as has never been made) will doubtless throw a great deal of light on a subject that has ever been puzzling to the prospector.

At the great concentration of dikes previously alluded to there are no less than seven parallel dikes, three from the northern system and four from the southern; and so closely do they approach each other that at one point it is but 9 miles across both systems.

There are no rules to be laid down in corundum mining and no written information of much value to the prospector. An eye trained by long experience and by careful comparisons of the many different outcrops of the peridotites, and the occurrence of corundum with its more frequent associate minerals, can tell by surface indications, with a reasonable degree of certainty, of the existence of corundum and if it is likely to be found in quantity. First, the quality of the olivine rocks will determine if it is likely to exist at all; and second, the character and quality of the associate minerals will determine the probable structural peculiarities of the corundum, and its consequent efficiency and value. They also indicate if the product will be easily mined and readily separated from its impurities. Its occurrence in true vein, or otherwise, can also be determined, since certain associate minerals and structural peculiarities in the wall of the vein proper will be present in no uncertain amount in the *débris* fallen down. The location of the olivine rocks and their contacts with the gneiss have a most important bearing with the

experienced prospector. No two outcrops that the writer has visited are exactly alike; each one has individual peculiarities, and it is not uncommon to find adjacent deposits of corundum whose associate minerals are wholly unlike in appearance, with an entirely different occurrence, necessitating radically different methods of mining and refining.

Next to the diamond, corundum is the hardest substance known in nature.* Its specific gravity varies considerably, but the mean is about 3.931. In color it ranges from white to black. Klaproth gives the following analysis of a Bengal product: Alumina, 89.5%; silica, 5.5%; iron, 1.25%; loss in decomposition, 3.75%. Analyses of specimens from the Carnatic, Malabar, China, and Ava by this and other chemists differ but slightly from the preceding. Prof. J. Lawrence Smith gives the following analysis of corundum from Asia: Alumina, 87.32%; silica, 2.6%; water, 3.74%; magnetic oxide of iron, 3.12%; lime, 1.00%; and gives its effective hardness at 57 as compared with the blue sapphire of India at 100. From a specimen of corundum from India he obtained: Alumina, 93.12%; magnetic oxide of iron, 0.91%; lime, 1.02%; water, 2.86%; silica, 0.96%; with an effective hardness of 58. He comments upon the low degree of hardness as compared with the blue sapphire as follows: "The quantity of water found to exist in corundum coming from different localities is variable, and it would appear that, all other things being equal, those containing the least water are the hardest." He further indicates the probability of association in "intimate mixture" of diaspore in one at least of these specimens, which gave over 3% of water, though imperceptible to the eye or glass. Diaspore stands only at 7 in Mohs' table, contains as high as 85% of alumina, and passes imperceptibly into corundum. Consequently the comparative softness of these specimens may have been due to the presence of diaspore. Analyses of pure corundum from different localities in North Carolina show that it does not vary much from blue sapphire, an average of three specimens standing as follows: Alumina, 94.06%; magnetic oxide of iron, 2.28%; silica, 2.56%; loss in decomposition, 1.10%. Careful tests by the writer, employing the same methods that were used by Professor Smith, show in three samples of blue sapphire from North Carolina (crystals) an effective hardness of 98 as against 100 from gem sapphires obtained from Burma, Ceylon, and Siam. The latter were pure transparent gems; the former, semi-transparent placer crystals. Three samples of opaque corundum (vein) crystals in the same series of tests stood at 97. Three samples of granular corundum from the Socrates Mine, Sapphire, N. C., stood at 94; sand corundum from three localities, 72; block or massive, three samples, two from North Carolina and one from Georgia, 79.

The Socrates corundum, when pure and made in wheels under the proper conditions, will do three times the work of any foreign emery the writer has tested, and twice that of the best Chester emery, and up to date more than any other corundum or abrasive product. This statement can be verified by the highest authority in the trade.

In the reduction of corundum from different localities it is observable that the same pockets or veins usually furnish a uniform product which crushes with the same fracture and into the same shaped grain, while another vein, or pocket in

* For the physical properties of the mineral the reader is referred to the article on "Sapphire" in another part of this volume.

the same vein, may differ entirely in both respects. Some corundums, notably the block or massive, crush rapidly to fine sizes on account of their laminated structure, and great care must be exercised in refining, since each process has a tendency to create finer particles. Sizes or numbers finer than 60 are much less valuable than coarser, being, by present methods of manufacture, far less efficient.

This question of structure and fracture is one of great importance to every corundum producer, since if the proportion of fine numbers be too great the product will be much harder as a whole to market. Any corundum that with ordinary care in the reduction processes gives over 20% of its weight in numbers finer than 60 will hardly prove economical unless it be absolutely pure, and it will in no case prove as economical or efficient when in wheel form as a more cohesive grain, since, although a rapid cutter, it will break down rapidly in comparison with other products, even under moderate pressure.

The Reduction of Corundum.—Three types of corundum are mentioned by various writers upon the subject, viz.: 1, sand; 2, block; 3, crystal. I would add a fourth type as an intermediate product of some mines between sand and crystal corundum, viz., granular corundum. Other writers usually include granular corundum under the head of sand, but sand corundum is, properly speaking, composed largely of very fine crystals or fragments of crystals, while the granular form (consisting of grains) is, as Lewis has aptly described it, "crystalline granular aggregates." These grains vary in size from that of one's fist, or even larger, to very fine particles, but with a structure and with properties of disintegration peculiarly their own. Perfectly shaped crystals are rarely found in these aggregates. This occurrence, in the opinion of the writer, furnishes the most economical vein to work, since it is readily separated from its impurities. It crushes less to fine numbers, and makes easily the best all-round wheel in the market. A corundum reduction-mill has therefore four separate products to handle, each requiring somewhat different treatment. I shall make, the following classification in the order of value: 1, granular; 2, crystal; 3, sand; 4, block or massive.

The granular corundum is seldom accompanied in the vein by either quartz or garnet, the ordinary impurities being its matrix of soft vermiculities or chlorites. It is usually of a grayish-white to cream color. For the reduction of this product the following processes are necessary:

1. Washing or hoeing in wide troughs set at a slight angle through rapidly running water, or, what is equally as effective, sluicing for some distance, the majority of impurities in either case being detached.

2. Drying and screening. The product is then dumped upon an inclined plane and allowed to drain sufficiently for sieving. All of the product that refuses to pass through a sieve with 14 meshes to the inch is crushed between iron rolls.

3. Muller process. This is by far the most rapid and thorough, as well as the most economical, process of cleaning all but the refractory products. It consists briefly of a shallow circular wooden tub 5 ft. in diameter with a cast-iron bed, within and upon which revolve two heavy wooden rollers about 12 times a minute, and through which a current of water is constantly flowing. This current is regulated according to the nature or specific gravity of the impurities. A raised platform in the center is provided with outlets through which the impurities pass.

Iron hoes set in the roller-frames precede them. The product is thus alternately stirred by the hoes and scoured by the heavy rolls following. The corundum, from its greater specific gravity, remains in the tub. This "mulling" process is continued from three to five hours, which is sufficient to clean any but a very refractory product. This muller, in one form or another, is also employed in many emery mills and wherever gravity processes are employed. It costs set up about $200. It requires from 3 to 5 horse-power to operate each machine when loaded, and one man will tend eight machines. Great care should be exercised in regulating the current, otherwise much corundum sand will be lost; also to avoid rounding the grains of corundum, which will rapidly take place after a certain point is reached in the process, and greatly impairs their cutting edges. This defect will be at once apparent in the wheel, since it will not "bite" freely and will soon wear uneven. This rounding of the grain is a common fault with both corundum and emery products and should be carefully guarded against.

4. Drying. After draining as before, the drying is greatly facilitated by the hot-air blast usually employed in similar processes, or by shallow flat iron pans heated by steam or hot air; or by the revolving cylinder. In the absence of these improved methods, the product may be fed from a hopper into belt elevators and carried to the top of a chimney 20 to 40 ft. high, whence it is dropped through the hot air upon an inclined heated plane of soapstone, down which it slides into a receptacle and before reaching the fire-box. This last method is employed by the Hampden Emery and Corundum Company at its North Carolina and Georgia mills, and was formerly employed by the writer at the Sapphire mines. It is simple, economical, and efficient.

5. Grading. In the manufacture of grain corundum and emery 24 sizes are usually made. The sieves employed are of brass or silk for the fine numbers, and usually of sheet iron or brass for the coarser numbers. They are either flat or circular. The silk sieves are very durable and economical. The corundum is first divided in a No. 60 sieve, and the fine sizes are worked one way, the coarse another, each sieve through the series being coarser than the one preceding it. The sieves are so arranged and adjusted that the corundum, which is fed evenly and slowly from the point of division, is (where flat sieves are employed) carried over each by a slight forward motion or jar, dropping the grains slightly after leaving each sieve to the next. Under each sieve is a bin numbered to correspond with the mesh of the sieve above it.

6. Magnetic separation. Almost every corundum product contains from 3 to 20% of magnetic iron, which is readily and rapidly separated by the ordinary magnetic separator.

7. If the product be very dirty, or if the processes 4, 5, and 6 have scoured off the few impurities left from the muller process, it is frequently advisable to again blow the coarse and medium numbers. For this operation great care must be exercised to prevent loss. The elutriation process is the only one that can be profitably employed to clean the finer numbers; this will be described later.

Crystal Corundum.—The preparation of crystal corundum is almost identical with that of the granular, the gangue being the same or similar. The crystals of much size will need to be first crushed (rolled if possible), and sieved before rolling again. Care must be taken in the crushing, or the crystals will go too largely to

fine numbers. The muller process is sometimes omitted when the product is very pure, the process of hoeing the second time often proving sufficient; otherwise the steps are the same.

Crystal corundum makes a harder and generally a more durable wheel than any other variety, although it does not cut as rapidly as the granular or some of the block varieties, but it is a more economical product than the latter, and cuts and wears far more evenly. For tool grinding strictly it is perhaps as good as the granular, all things considered. Corundum crystals are of all colors, from white to bluish-black. The prevailing colors are gray and reddish-brown.

Sand Corundum.—This product is also usually found imbedded in a vermiculite and chlorite gangue, but is generally accompanied with more quartz or garnet, or both, than the two products previously described. These cannot be separated economically by gravity processes, their specific gravity being so nearly the same as that of corundum. It is also apt to be accompanied by other somewhat heavy impurities, such as decomposed feldspar actinolite and tournaline. When garnet and quartz are the only heavy impurities, and if they do not occur in too great quantity, the product will make a good abrasive, provided none of the other impurities contains water or iron. A product of 50% sand corundum with not over 10% of vermiculite and chlorite (yellow or green), the remainder being garnet or quartz, will prove superior to any emery on the market. The Hampden Emery and Corundum Company's North Carolina product has been for long periods of this character, and the writer can state with confidence, from actual wheel tests, that it will cut 50% faster than a Chester emery-wheel of the same size and grain. Sand corundum varies greatly in color, but the prevailing colors in the vein are white, red, blue-gray, and gray, and some shades of brown.

Block or Massive Corundum.—This varies in color from blue-gray to gray-pink, red and blue, brown, blue-black, the prevailing color being gray. Generally speaking, it is the least efficient and economical of all corundum products. Owing to its generally laminated structure, block corundum crushes more readily than the crystal or granular varieties, and far more easily than emery. A few blows of the crusher is sufficient to break any lump that will enter it, and if great care is not employed much of the product will go to powder. Sometimes this product occurs nearly pure and the blocks are of great size. At Laurel Creek, Ga., blocks of great purity weighing upward of 5000 lbs. occurred. Owing to its purity, the corundum from this locality was in great demand for "vitrified" wheels, but the mine was closed on account of the peculiar condition of the abrasive trade. Usually the impurities in massive corundum are extremely refractory. The primitive fire process is generally employed in remote districts in breaking the masses to a size convenient for the crushers. Having been heated on a bed of burning wood, water is turned on and the pieces crack readily, separating much of the lighter impurities. If possible this product should be reduced to the required 14-mesh size on the ground and sluiced to the mill. Should the product at this point be above 50% corundum and with no other impurities save quartz and garnet or iron, it will pay to handle; but if other impurities, as is frequently the case with this variety, interpenetrate the individual grains, no gravity process will suffice and it will prove almost worthless as an abrasive. The only recourse is to the chemical processes, and these are not economical in regions remote from railways. In cases

where the impurity does not interpenetrate the small grains, a scouring machine called the "augur" will frequently remove the more refractory material. This machine was invented by Downs, a blacksmith in the employ of the Hampden Emery and Corundum Company at Corundum Hill, N. C. It consists of an open box, 5½ ft. long and 2 ft. wide, inside measurements, made of 2-in. oak. This is set on a strong frame, bolted to the floor, and from floor to opening in the box is 3 ft. Through the box lengthwise pass two bars of round iron, the revolving shafts, 2½ in. in diameter, which are set 3 in. above the bottom and about 2 in. from the sides. These project from the box about a foot from either end, where they rest in journals. Pulleys are attached to both shafts at opposite ends. Inside the box are four spiral screws, or augurs, made of 2½-in. flat steel ½ in. thick, which are shrunk and bolted upon the revolving shafts, two on each shaft. The shafts revolve rapidly toward each other, and are so arranged that the ore when shoveled in wet at the top is caught by the screws and worked from the center toward each end, at which there are two openings, regulated by a cut-off on either side. Near the orifice a shoe of steel is bolted to the end of each augur, the orifice being protected by sheet steel on the inside. When run at high speed all but the cleanest product is beaten into mud, which is washed out in the troughs and the operation repeated if necessary.

The writer has employed various makes of jigs, tumbling barrels, gravity machines, etc., but has decided in favor of the "muller" or "chaser" as by far the most efficient and economical. Any corundum product that cannot be separated readily from its impurities by the processes described is of doubtful value unless the mine be very near the railway, and where the chemicals necessary for its further treatment can be readily and cheaply obtained. The method to be employed in this event depends entirely upon the nature of the impurities, and space will not permit an extended reference to this system of their elimination.

Fine sizes may be readily and economically cleaned by elutriation and to a high degree of purity. Of several processes of elutriation the following is the one generally used by the optical and other trades. Several metal cylinders, sometimes as many as a dozen, about 3 ft. high and from 3 in. to 40 in. in diameter, are connected near their upper edges by pipes. The whole series is filled to the top with clean water. The corundum or emery to be cleaned is thoroughly moistened and introduced into the smallest vessel. A current of water is turned through the connected cylinders, and the corundum settles in each according to size of grain, the velocity of the current decreasing towards the cylinder of largest diameter. With the sediment in each cylinder there is more or less of impurity. This is separated by draining off the water and drawing the cake of sediment from the bottom, allowing it to harden thoroughly. When it can be handled an expert takes the cake in hand, and from long experience is able with a knife to exactly divide it at the point where the impurities and the corundum or emery unite.

These fine numbers when pure sell readily at 10c. per lb. The present prices for the standard Hampden Emery and Corundum Company's product, best North Carolina, when in kegs is 7 to 10c. per lb. for sizes No. 46 and coarser, and 4 to 7c. for No. 54 and finer, with some concession to large and regular customers. The average of all sizes at present is 7c. Selected crystals are in good

demand in certain quarters at a higher price than grain corundum, however pure. At present the supply is not equal to the demand and the writer is in a position to state positively that at least 2500 tons could be sold annually in this country alone, at prices above quoted, if it could be furnished of good quality and purity.

EMERY.

By CHARLES N. JENKS.

EMERY is an intimate mixture of corundum with magnetite or hematite. The chief European sources of this mineral are certain districts in Asia Minor and the neighboring islands of the Grecian Archipelago. It is also found in Persia, Sweden, Spain, Saxony, Poland, Jersey, Guernsey, in the Andes of South America, and in Greenland. In the United States several occurrences have been noted, but only two deposits of any value have been uncovered. These are at Chester, Mass., and at Peekskill, N. Y.

For many centuries the Island of Naxos in the Grecian Archipelago furnished almost the entire supply. In 1846, at the request of the Turkish Government, Dr. J. Lawrence Smith made a careful examination of the Turkish deposits, and found two of great value, besides locating others. Operations were almost immediately begun under the auspices of the government, and soon the price of emery, which had been at $140 per ton (best Naxos), fell to $50 @ $70 in consequence of the new supply.

In color emery varies considerably. The Naxos ore is readily distinguished by its uniformly gray or blue-gray color, its micaceous specks, and glistening luster; the Turkish, by its generally darker and more metallic appearance. The Chester, Mass., emery in some specimens has a peculiar greenish color, varied by gray specks, and in others it is dark blue. The Peekskill, or "Westchester," emery could readily be mistaken (some specimens at least) for either Turkish or Chester; but it is much finer-grained than either, and is, generally speaking, of a bluish-gray to dark blue color. English-manufactured emery is often artificially colored to a uniform tint, and is sometimes adulterated with ground iron slag. Some buyers even insist upon this coloring and adulteration, and if accustomed to a particular shade and appearance will reject a far better article of a different color as impure.

Emery from all localities is frequently incrusted with magnetic iron, and this crusting also frequently contains pyrites or mica. Mica sometimes penetrates the entire mass, giving the ore a silvery luster. Its specific gravity varies from 3.75 to 4.28. The following analyses are by J. Lawrence Smith and Thomas M. Chatard:

No.	Analyst.	Source.	(*a*) Hardness.	Al_2O_3.	Fe_3O_4.	SiO_2.	CaO.	MgO.	MnO_2.	H_2O.
1	Smith.	Naxos.	44	69.46	19.08	2.41	2.81	.....	.....	5.47
2	"	"	46	68.53	24.10	3.10	0.86	.....	.....	4.72
3	"	Gamuch.	47	77.82	8.62	8.13	1.80	.....	.....	3.11
4	"	Kulah.	53	63.00	30.12	2.36	.50	.....	.....	2.36
5	"	Samos.	56	70.10	22.21	4.00	.62	.....	.....	2.10
6	"	(*b*) Average.	48.7	67.97	23.11	11.40	1.14	.....	.....	3.29
7	"	(*c*) Chester.	40	50.02	44.11	3.25	.....	.....	.....	.....
8	"	(*d*) "	39	51.92	42.25	5.46	.....	.....	.....	.....
9	Chatard.	"	.....	71.61	21.25	3.87	Trace.	1.43	0.03	1.56

(*a*) As compared with the sapphire of India = 100. (*b*) Average of five samples of Grecian and five of Turkish. (*c*) Better quality stone emery. (*d*) Prepared grain.

Three samples of Chester emery, two obtained from the trade and one from the producers, subjected by the writer to the agate and glass test in the manner employed by Smith, gave an average hardness of 62, which is the highest figure ever obtained by this method from any emery product. Smith's tests of the Chester were made many years ago, when the mine was first opened.

Emery is, generally speaking, very cohesive (more so than most varieties of corundum), and it is extremely hard to reduce to grain form, requiring when in lump a succession of heavy direct blows to fracture it. This is due to its iron content, which binds the grains very firmly. Emery owes its abrasive qualities almost entirely to the corundum it contains. If it were deprived of its iron it would be a good corundum, but the mixture is so intimate that a separation by electrical or mechanical apparatus would be difficult if not impossible.

Naxos Emery.—In Naxos there are 17 deposits of emery, all of which have been worked. The most important mines are in the vicinity of the village of Bothis. The island is principally made up of Archæan gneiss and schist formations, the latter consisting of mica schists alternating with crystalline limestone. The emery occurs as lenticular deposits, of a maximum length of about 100 yards and a maximum thickness of 50 yards. They are closely associated with the limestones. They vary in position from horizontal to nearly vertical. In some localities the mineral occurs in strata from 1 to 2 ft. thick, which are crossed by two systems of joint-planes so that the mineral breaks into nearly cubical blocks. The floor of these deposits is invariably crystalline limestone, the roof crystalline dolomite, and over that is mica schist. The working of the deposits is conducted in an extremely primitive fashion, the rock being broken entirely by fire-setting. The regular use of explosives is impossible, since the rock cannot be drilled economically with steel. These deposits have been worked open-cast almost entirely, partly on account of the dangerous character of the dolomite roof and partly through the ignorance of the native workmen.

I am informed, by those who are in a position to know, that nc Naxos stone of first quality ever reaches this country, since the English, French, and German manufacturers have agents at Naxos, who buy all the best stone as fast as mined. This statement seems highly probable. No American manufacturer has an agent at Naxos, and the low prices obtained for all abrasive products in this country would hardly warrant the purchase of selected stone at a greatly advanced price, such as European manufacturers are compelled to pay for first choice at Naxos.

In the opinion of the writer the value of Naxos emery has been overrated, the Turkish being equally good, while that of Chester is much superior to either as an all-round abrasive. The Naxos emery contains a comparatively large percentage of water, and there is ground for distrusting any product containing a large percentage of water, whatever else it may contain.

The best Naxos emery left after the culling before mentioned can be purchased directly from the Greek Government and costs $31 @ $36 per ton delivered in New York or Boston, while inferior lots have been offered at $13.

According to Consul-General Stevens of Smyrna (1885) the emery mines of Greece are leased by the Greek Government to the highest bidder for periods of 12 years, the lessee being obliged to take out at least 1650 tons of emery per annum. The Naxos product is taken to the neighboring island of Syra in sailing-

vessels, whence it is reshipped. A good deal of it goes to Smyrna, the chief depot for the Turkish mines, where it is marketed.

Turkish Emery.—Smith, in his description of the occurrence of emery at Gamuch and at Kulah, says these localities "are those which afforded me the best means of studying the geology of emery, although in every instance I have found it associated with the old limestone overlying mica, slate, gneiss, etc. It is imbedded either in the earth that covers the limestone or in the rock itself, and exists in masses from the size of a pea to that of several tons' weight, generally angular, sometimes rounded, and when in the latter form they do not appear to have become so by attrition. Having had the means of studying the emery and rock in contact, I have come to the firm conclusion that the emery has been formed and consolidated in the limestone in which it is found, and that it has not been detached from the older rocks as granite, gneiss, etc., and lodged in the limestone at its period of formation."

Consul Stevens, previously quoted, says that emery is found in nearly all parts of Asia Minor, frequently in the remote regions of the interior. The principal deposits are in the districts of Thyra and Aidin, south of Smyrna and not far from the Ottoman Railway, where the chief mines are the Charnaud, Jackson, and Abbott. The ore is re-sorted in Smyrna. Another mine near Milassa is worked, the larger part of the product of which finds purchasers in the United States. It is of inferior quality, uncleaned in many cases, its grain is smooth, and it contains a large amount of magnetic oxide of iron.

The Turkish mines are worked under grants from the Imperial Government, usually for periods of 99 years. It is almost impossible for Europeans to obtain these, although this difficulty may be overcome by making application in the name of an Ottoman subject or jointly with him.

The chief mines in Turkey are opened by shafts and galleries, explosives being used in the extraction of the mineral. At the Jackson mine a steam-pump is employed, the ore lying at considerable depth and water being encountered. At one of the Abbott mines the emery is dug up from the red argillaceous earth of the surface. The ore from these mines is conveyed by camels to the nearest railway point, and thence forwarded to Smyrna, where it is sorted again prior to shipment. The large pieces are broken at the mine by heating and then cooling suddenly with water, being finally reduced by sledging.

Stevens said (1885) that the total shipments from Smyrna aggregate about 7000 tons annually. The latest information (1896) obtained by the writer, which, however, he has not been able to confirm, places this amount at over 14,000 tons.

Chester Emery.—The emery vein at Chester, Mass., occurs in a zone of chlorite schist lying between talc rock on the east and a hornblende schist on the west. The highest outcropping is near the crest of the ridge, 650 feet above the base of the mountain. The vein traverses the mountain on both sides of the Westfield River, following a general course of north 20° east, and dips 70° east. It has been traced for several miles. The Chester mine, on account of its nearness to railway and trade centers, and the character of the product, which is readily refined, continues to be worked even in the face of the strong foreign competition. This emery deposit is very large. It is owned by the Hampden Emery and Corundum Company, which has very complete and modern methods of mining, and well-

equipped reduction works at Chester. A wheel can be made of this product which will do nearly or quite a third more work in a given time than any other known emery, and with greater economy. The present annual output is about 1200 tons.

Peekskill or "Westchester" Emery.—These deposits occur about three miles from the town of Peekskill, N. Y., and about 750 feet above sea-level. On the north and northeast are outcrops of granite; on the south, the marble of Sing Sing and the gneisses (Laurentian?) at Spuyten Duyvil. The emery is directly associated with hornblende rock. The product is not as efficient as the average of the Naxos, Turkish, or Chester. A strong effort is being made to introduce it in the trade, and it is said that over 1000 tons were produced in 1896; whether all was marketed I am unable to say. With a good base this product will doubtless do more efficient work in a wheel than a good deal of emery that bears a better reputation.

Other Localities.—Several other localities are mentioned as furnishing specimens of emery; among them are Tallapoosa, Ga., and at two or more points in North Carolina. The writer has no personal knowledge of these deposits.

Manufacture of Grain-emery.—The leading manufacturers of grain-emery in the United States are the Washington Mills Emery Manufacturing Company, North Grafton, Mass.; the Ashland Emery Mills, Perth Amboy, N. J.; the American Emery Mills, Easton, Pa.; the Hampden Emery and Corundum Company, Chester, Mass.; and the Walpole, Diamond, Jackson, and New York mills, which are represented in the trade by the Levant Emery Company of New York. These concerns make over nine tenths of the total production of grain-emery in the United States.

The methods of preparing grain-emery are practically the same as those described in the reduction of corundum, although each emery-mill has certain fancied improvements over other mills that are guarded with jealous care. These consist mainly in labor-saving devices or attachments, but to all intents and purposes the processes are alike. These processes of preparing emery are expensive, and the percentage of loss in the separation is necessarily large. From the lump to the finished product in kegs it costs fully 1¼ @ 1½c. per pound. This added to the cost of the stone, 1c. (estimated) makes the total cost to the manufacturer 2¼ @ 2½c. per pound. The cost of marketing is also considerable, yet the competition is so strong that the best American article on the market can be bought for 3½ @ 4c. freight paid. This leaves a margin of not to exceed 1c. per pound, and probably less in some cases.

The present quotations on emery in the American market, or rather the prices at which emery can be purchased in keg lots, all numbers, are as follows: Chester, 3½c.; Turkish, 3½c.; Naxos, 3½ @ 4c.; Peekskill or "Westchester," 3c. In some cases the freight to destination, at these prices, is paid. The American Glue Company, with offices in New York and Boston, agents for "Wellington Mills" best selected Naxos, quote prices in keg lots as follows: 7c. for No. 54 and coarser, 5c. for No. 60 and finer, with a discount of 10% on large orders. It reports no deviation from these prices and ready sales. The Chester is, however, easily superior to this, the best obtainable Naxos grain.

The tariff on emery since August 17, 1894, has been as follows: emery-grains and emery ground, pulverized, refined, or manufactured, 0.8c. per pound; emery

wheels and files, 20% ad valorem; emery ore, free of duty. Prior to this the tariff was 1c. per lb. on emery grains, emery ore being free.

Stevens says: "Good clean lump stone can be contracted at Smyrna on board ship at $19.50@$20.70 per ton of 2240 lbs. The same quality, but in smaller pieces, at $3.65 per ton less." This is an average price of less than 1c. per lb. Another authority (Heap) gives the average price at Smyrna at $19.50 per ton of 2240 lbs. From other reliable sources in the trade, both here and abroad, the average cost of lump stone is estimated at about 1c. per lb. in Boston or New York. The cost of shipment from Smyrna, Naxos, or Syra varies from $1.22, $2.44 per long ton in steamer; by sailing vessels at merely nominal figures, being taken as ballast.

Garnet.

Garnet is mined in Warren County, N. Y., and Delaware County, Pa., four or five firms being engaged in its production. The mining is done entirely by open-cast work, and is consequently suspended during the winter. The total number of persons engaged in the industry is not more than 125.

Previous to 1882 garnet was imported into the United States from Europe; in that year a very hard class of the mineral in large quantity was discovered in the Adirondack Mountains of New York; its quality being superior to that of the European mineral, the latter was displaced in this market and American garnet has even been sent to Europe. The Adirondack garnet is of hardness 7.5 to 8.0, or nearly that of topaz. It occurs in laminated pockets scattered through beds of a very compact hornblende-feldspar rock, the size of the pockets ranging up to 1000 lbs. yield, though the most are not more than 5 or 6 in. in diameter. The numerous cleavage planes of the mineral prevent its use as a gem, but make it the more desirable for abrasive material, affording sharp, cutting edges to the crushed material. The mineral is found in numerous localities, but only in a few where it occurs in sufficient quantity to be workable and has the requisite hardness, ordinary garnet being only 6.5 to 7.5 on the scale.

The garnet of Delaware County, Pa., occurs in aggregates of small crystals in quartzose gneiss. It is not obtained so pure as the Adirondack garnet, and consequently has to be cleaned by washing, which is done in the same manner as corundum washing.

A good deal of common garnet, in huge crystals, is found in North Carolina, but these are of inferior quality for abrasive purposes, and have not yet found a market.

The value of Adirondack garnet, clean product, is $40 per ton of 2000 lbs. f. o. b. cars. Other kinds of garnet are quoted at a lower figure.

Garnet and garnet paper are exported from the United States to Europe.

Pumice.

Pumice is a volcanic glass, closely related to obsidian. The term "glass," however, is used as a petrological classification, since the substance is apparently a frothy, cellular scoria. It is in fact a variety of lava to which an excessively cellular structure has been imparted by expanding steam-bubbles.

But it passes from this condition gradually into the heavy, vitreous obsidian, which is of a similar chemical composition. It is found in many places, especially on the shores of the Tyrrhenian Sea, but that of commerce comes almost exclusively from the island of Lipari.

The best stone is obtained from the Monte Chirica, in the northeast of the island, the district covering an area of about 3 sq. miles. It occurs in strata underlying a mantle of white tuff, and is extracted by underground mining, the workings sometimes going to considerable depth. The deposits are, however, very irregular, both in extent and in quality.

The number of openings is estimated at 250, and the number of persons engaged in the industry at 1000, of whom 600 are miners. The rock is brought to the surface in blocks, or in baskets, and carried to Canneto, whence about one-fourth goes to Lipari for further manipulation. At the latter place it is allowed to dry a month, and crumble into *lisconi*, after which it is sorted into *grosse* (lumps down to the size of the fist), *correnti* (medium-sized pieces), and *pezzame* (from 2 in. downward). It is then further subdivided according to quality. The *grosse* are sorted into three superior grades, namely, *fiore*, *quasi-fiore*, and *mordente*, which are ready for shipment. The inferior grade is then dressed with files to remove asperities of surface and test whether the stone is not too friable for use. It is next classified into *bianche*, *dubbiose*, and *nere*. Large pieces of inferior pumice, known as *rotonde*, are never trimmed. Besides this there is an entirely different variety, the so-called *alessandrina*, which is cut with hatchets into brick-shaped pieces and used for smoothing oil-cloth; and a heavy dark stone, *bastardone* (always trimmed), and several less important varieties.

The *correnti* are generally exported without dressing. The *pezzame* are usually, but not always, ground to a powder, of fineness according to the work for which it is required.

Good pumice is of approximately the following composition: Silica, 73.70%; alumina, 12.27%; potash, 4.73%; soda, 4.52%; oxide of iron, 2.31%; water, lime, etc., 2.47%—total, 100. The quality is chiefly a matter of texture, an essential condition being homogeneity and freedom from included crystals. The stone must be neither too brittle nor too hard.

The pumice deposits of Lipari are very extensive, and if improved methods of mining were introduced the output could be increased very much.

The stone washed up by the sea is seldom collected nowadays. Attempts have been made to manufacture bricks out of the *pezzame*, but so far no satisfactory cement has been found. A menace to the industry lies, however, in the manufacture of an artificial pumice in Germany, which is said to give good results.

The French market demands the best stone, and will not take filed material, which is, however, accepted in the English market. The largest exportation is to England, with France, America, Austria, Germany, and Belgium following in order named. Large lumps, known as *testoni*, are sent to Trieste to be ground there. Almost all the *alessandrina* finds its way to England and America. A good deal of inferior pumice goes to St. Petersburg and Odessa.

The values of the various grades at London is as follows: *Fiore*, $24.35 per

cwt. (112 lbs.); *quasifiore* and *mordente*, $9.25; first pick, $4.40; second, $3.40; third, $2.90; *bastardone*, $2.70; *alessandrina*, $2.45; *pezzame*, $19.50 to $24.35 per ton (2240 lbs.).

Prices rose considerably about 1888 when a syndicate was formed, with a capital of 500,000 lire, to rent the municipal pumice lands for 92,000 lire per annum. It soon failed, however, owing to mismanagement, since when prices have declined.

Rouge.

Rouge, which is used for polishing metal, is ferric oxide prepared by calcining copperas. Some is also made directly from steel by a new process which is in use at Pittsburg.

Silica.

Silica is used as an abrasive in three forms, viz., quartz, diatomaceous earth, and tripoli, each of which is produced in the United States.

Quartz in the form of sand is used extensively in the manufacture of sandpaper. The white, milky variety is also crushed fine and used for the same purpose, appearing in the trade as "crushed flint." The uses of quartz as an abrasive, however, are far less important than its uses for other purposes. A large amount, under the name of flint, is consumed in pottery making. Quartz crystal is mined in Connecticut for reduction to powder, in which form it is mixed with oil and applied to smooth wooden surfaces; the oil carries the quartz into the pores of the wood, and makes it capable when dry of taking a high polish. In 1894 the production of quartz for this purpose amounted to 6024 short tons, value $18,054; in 1895 it was 9000 tons, value $27,000. In North Carolina several thousand tons of quartz have keen produced for use in absorption towers for sulphuric acid works. But by far the most important use of quartz is (chiefly in the form of sand) in glass-making.

In the production and consumption of infusorial earth and tripoli in the United States in 1896 there was no new development.

Steel.

Crushed steel and steel-emery are artificial abrasives made of highly tempered steel. They are comparatively new products, but are likely to replace natural abrasives for a good many purposes.

ALUM.

THE only aluminum-bearing materials used for the production of alum and sulphate of aluminum in the United States are bauxite and cryolite. Besides the preparation of small quantities of aluminum chloride and minor chemicals, which may be ignored, the only other purpose for which these ores are used is the production of aluminum. Consequently it is possible to calculate from the consumption of bauxite and cryolite and the production of aluminum the production of alum and sulphate of aluminum. On this basis the production in 1895 was 117,900 short tons, expressed as crystallized alum, and in 1896 it was 77,700 short tons. The proportion between the makes of alum and sulphate of aluminum in 1895 is estimated at 1 : 2.5; in 1896 at 1 : 3. At the latter ratio the production of 1896 was approximately 14,000 tons of crystallized alum and 42,000 tons of sulphate of aluminum.

Alum and sulphate of aluminum were made in 1896 at eleven works; five in Pennsylvania, four in New York, and two in Massachusetts. Aside from the general dullness of business there were no features of especial interest in the industry other than the continued tendency to make sulphate of aluminum instead of crystallized alum, and the further displacement of imported bauxite by the domestic product. It is recognized that the American bauxite, although not so high in alumina as the French, gives an equally high yield, and is free from deleterious impurities.

The average price for alum in 1896 was $1.50 per cwt., which was the same as in 1895. Sulphate of aluminum was quoted at 90c.@$1.80, the price varying according to quality and tenor in alumina.

STATISTICS OF ALUM IN THE UNITED STATES.

Year.	Production. (a)			Imports. (b)			Year.	Production. (a)			Imports. (b)		
	Short Tons.	Value.	Per Pound	Short Tons.	Value.	Per Pound		Short Tons.	Value.	Per Pound	Short Tons.	Value.	Per Pound
1891..	74,000	$2,220,000	$0.015	2,326	58,864	$0.013	1894..	72,000	$2,160,000	$0.015	919	$30,832	$0.016
1892..	86,000	2,580,000	.015	2,070	59,336	.014	1895..	117,900	3,537,000	.015	1,492	46,815	.016
1893..	96,000	2,880,000	.015	2,286	73,806	.016	1896..	77,700	2,331,000	.015	2,763	86,372	.016

(a) Calculated as crystallized alum. (b) Includes alumina, alum, alum cake, sulphate of alumina, aluminous cake, and alum in crystals or ground.

ALUMINUM.

THE production of aluminum in the United States remains in the hands of one concern, the Pittsburg Reduction Company, which during 1896 put in operation its new plant at Niagara Falls, known as the lower works. The litigation which stopped the operations of the Cowles Company is not yet definitely settled, however. The statistics of the industry for the past five years are given in the following table:

PRODUCTION, IMPORTS, AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES.

Year.	Production.		Imports.		Consumption.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1892.....	295,000	$191,750	43	$51	295,043	$191,801
1893.....	312,000	202,800	7,816	4,683	319,816	207,483
1894.....	817,600	490,560	5,303	2,524	822,903	493,084
1895.....	900,000	495,000	25,294	7,814	925,294	502,814
1896.....	1,300,000	540,000	698	591	1,300,698	540,591

In Europe the new works of the British Aluminum Company at Foyers, Scotland, were put in operation in June, 1896, and plans are already being made for increasing their capacity. Besides the Foyers works the company is now making alumina at Larne, Ireland, electrolytic carbons at Greenock, Scotland, and metal castings, sheets, rods, tubes, etc., at Milton, Staffordshire. This company has made no report of its output in 1896. That of the French and Swiss works was about the same as in 1895.

The Neuhausen Company has in process of erection a 10,000 horse-power plant at Rheinfelden, near Basle, Switzerland, which it expects to have in operation this year (1897). It has also acquired a water-right in Austria. In Norway a syndicate of German and American capitalists has purchased rights at the Falls of Sarpsfos, on the Hafslund estate, between Christiania and Göteborg, where it is intended to have aluminum works in 1898.

The capacity of aluminum works now in operation, and their estimated capacity in 1898, is shown in the following table:

Company.	Location.	1897.		1898—Prospective.	
		Horse Power.	Daily Capacity.	Horse Power.	Daily Capacity.
Pittsburg Reduction Company	New Kensington, Pa.	1,600	2,000	1,600	2,000
	Niagara Falls, N. Y.	1,600	2,400	7,100	9,400
Alum. Ind. Act. Gesell.	Neuhausen	4,000	5,000	4,000	5,000
	Rheinfelden	.....	.....	6,000	8,000
Soc. Elec. Met. Française	La Praz	2,500	3,000	2,500	3,000
Soc. Ind. de l'Alum	Saint Michel	2,000	2,500	2,000	2,500
British Aluminum Company	Falls of Foyer	3,000	4,000	3,000	4,000
Proposed works	Falls of Sarpsfos	.....	.....	5,000	6,500
Totals	.....	14,700	18,900	33,200	42,900

At the beginning of 1896 the American scale of prices was as follows: No. 1, 98% pure, ingots for milling, 50@55c., according to size of order; ingots for remelting, 48@53c.; No. 2, 94% pure, ingots for remelting, 42@50c. These rates held until July, when for No. 2 was quoted: Ingots for remelting, 38@43c., and ingots from scrap, 35@40c. In November the Pittsburg Reduction Company announced a reduction in prices of from 5c. on the lower grades to 11c. on the higher, the new schedule (which was maintained to the close of the year) standing thus: No. 1 (guaranteed over 98% pure) ingots for remelting, 42c. in small lots, 39c. in 100-lb. lots, 37c. in ton lots; No. 2 (guaranteed over 90% pure with no injurious impurities) ingots for remelting, 34c. in small lots, 33c. in 100-lb. lots, 31c. in ton lots; nickel-aluminum casting metal (pure aluminum alloyed with less than 10% nickel and other hardening ingredients), 40c. in small lots, 38c. in 100-lb. lots, 35c. in ton lots; special casting alloy (containing over 80% pure aluminum, used in place of brass), 35c. in small lots, 30c. in 100-lb. lots, 27c. in ton lots; castings, 45c. per lb. upwards, and special rates for bars, rods, shapes, etc.

No important improvements in the metallurgy of aluminum have been put on record since the publication of the last volume of THE MINERAL INDUSTRY; nor have there been any new applications of the metal in the arts, which indicate use for a large production. In the manufacture of household utensils and small ware, however, consumption of aluminum is undoubtedly increasing rapidly. The French military authorities are now using it extensively for drinking and mess-vessels, which are made from plate without the use of solder. These vessels are found to resist well ordinary wear and tear, the action of fire, foods and cooking liquids not left in contact with the metal for long periods. Experiments on this point have been very exhaustive, and as to the adaptability of aluminum for saucepans, spoons, cups, etc., there should no longer be any doubt. About 15,000 sets of aluminum cooking utensils were used by the French troops in the Madagascar campaign. Aluminum helmets, however, have proved unsatisfactory, wearing out too quickly. One concern has introduced aluminum gold-pans, or bateas, in sizes from 12 in. diameter upwards, which are very convenient and show "colors" fully as well as the ordinary, dark Russia-iron pan. The Pittsburg Reduction Company substituted aluminum for copper to conduct current to its new works at Niagara from the central power station, the distance being about 350 ft.

Professors Richards and Scott have made the following determinations of the electrical conductivity of aluminum:

Sample.	Ohms per 1,000 ft. O 1 in. Diam. at 15° C.	Per Cent. Conductivity at 25° C.	Per Cent. Variation per Degree Between 15 °C and 80° C.
Copper	101.83	100.00	0.388
Aluminum No. 1, 99½%	161.40	63.90	0.385
Aluminum No. 2, 99%	163.80	62.17	0.385
Aluminum No. 3, 98%	181.30	56.17	0.360
Aluminum No. 4	174.10	58.48	0.361
Aluminum No. 5	185.10	55.01	0.359

A French test lately showed commercially pure aluminum to have a conductivity of 62%, taking copper at 100, which agrees with the results obtained by Professors Richards and Scott.

On the French state railways it is stated that a number of passenger cars are to be tried in which all brass, copper, and iron parts, with the exception of axles, wheels, bearings, and springs, brake beams and couplings, are to be replaced by aluminum. The weight of a car provided with aluminum fittings is 1½ tons less than that of the old coaches. As an ordinary train in France consists of 20 vehicles, the weight of the train would thus be reduced by 30 tons.

The subject of aluminum alloys received the usual amount of investigation, without, however, reaching any result of unusual interest. According to a writer in *Engineering* (London), the properties of aluminum alloys depend on the purity of the metal employed, and also on the homogeneity of the alloy. It has been found that aluminum absolutely free from sodium, alloyed with 6% of tin, was attacked by water with disengagement of gas, while in presence of a considerable amount of tin the decomposition is much stronger; hence tin solder for aluminum should be absolutely rejected.

F. G. Andrews describes the properties of various aluminum alloys.* The aluminum-gold alloys are of remarkable color, but of no value except for decorative purposes. Of the aluminum-silver alloys those with 4 to 8 parts silver are almost silver-white, harder than aluminum and not so brittle, take a good polish, and are useful for medals, jewelry, and decorative work. The alloy of 66% Cu, 24% Ni, and 10% Al takes a good polish and has the color of 10-carat gold. That of 55% Cu, 33% Ni, and 12% Al has a golden-brown color, while that of 72.5% Cu, 21.25% Ni, and 6.25% Al is similar but richer and deeper in tint.

R. I. Roman† claims that an alloy of 98% Al, 1% Ni, and 1% W has all the qualities of aluminum as regards low specific gravity, ductility, color, and resistance to corrosion, and in addition has greater tensile strength and elasticity, takes a finer polish, and can be more easily machined. The same inventor also claims‡ that 99% of aluminum melted with 1% calcium carbide, and subsequent removal of calcium, gives a metal with about 0.1% carbon, which has considerable hardness, greater elasticity, and a lower melting point than pure aluminum.

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* *Dingler's Polyt. Jour.*, 1896, p. 299, 24. † English Patent No. 21,186, Nov. 8, 1895.

‡ English Patent No. 17,250, Sept. 16, 1895.

An alloy of 90% tin and 10% aluminum melts at 390°, has a permanent brilliancy, and can be strongly soldered to glass.

While aluminum has not found an important application in the metallic state it has come to play a valuable part in metallurgy. It now partly replaces manganese in the manufacture of iron and steel, wholly replaces magnesium in making nickel, and is used in the production of brass. Recently it has been employed to reduce metals where carbon fails, as in the Greene-Wahl process for manganese, and now H. Moissan suggests a further utilization of this idea in the incorporation of tungsten, molybdenum, uranium, vanadium, titanium, silicon, and boron (which are difficult to reduce and also to fuse with other metals) in iron, steel, copper, and the useful alloys.* He proposes to reduce the oxide of the desired element by aluminum, forming an aluminum alloy which fuses at the heat of the reaction. The alloy will be added to the iron or steel while molten, and the excess of aluminum removed by refining with ferric oxide or copper.

The analysis of aluminum and its alloys is described by H. Moissan in *Comptes Rendus*, Vol. CXXI., pp. 851–856, and *Journal of the Society of Chemical Industry*, 1896, p. 136.

*H. Moissan, *Comp. Rend.*, Vol. XXII., pp. 1302–1303; H. Le Chatelier, *Bull. de la Soc. Cl. Encouragement*, Vol. CXIX., pp. 1196–1202.

ANTIMONY.

THE sole producer of antimony in the United States is Mathison & Co., whose works are at Chelsea, Staten Island. They smelt some ores derived from California, but chiefly ores imported from foreign countries; the most regular supply being Europe, though odd parcels are obtained from Australia and South America. There are good deposits of antimony in Nevada, California, Montana, Arizona, and elsewhere in the United States; but at the present value of the ore, viz., $40 to $50 per ton, there are very few which can pay freight to smelter and have a margin for mining expense. The small amount of ore that is produced in California is shipped to New York per sailing vessel, whereby comparatively low freights are obtained; yet even with cheap ocean freight transportation to the seaboard is so expensive that the production of the State has been falling off.

PRODUCTION AND IMPORTS OF ANTIMONY IN THE UNITED STATES.

Year.	Production.				Imports.				
	Metal.		Ores.		Crude and Regulus.		Ores.		Total Value.
	Pounds.	Value.	Pouads.	Value	Pounds.	Value.	Pounds.	Value.	
1892.....	400,000	$36,000	1,700,000	$51,000	3,950,864	$392,761	192,344	$7,338	$400,099
1893.....	700,000	63,000	1,700,000	41,000	2,780,432	243,341	116,495	5,253	248,594
1894.....	440,000	39,000	330,000	9,075	1,205,752	79,265	375,468	18,068	97,333
1895.....	932,000	70,332	2,166,000	37,905	3,449,901	251,850	668,610	31,893	230,946
1896.....	1,226,000	85,700	300,000	4,750	3,583,390	117,092	1,180,828	21,392	226,634

This table does not include the production of antimony in the form of antimonial lead, which is very important and is included under the caption "Lead" further on in this volume.

Following is a list of the antimony deposits of California, described in the report of the State Mineralogist published in 1896:

Antimony, ¾ mile north of Havilah, Kern County. The ore, stibnite, occurs in irregular masses in porphyry. Opened by an adit 70 ft. long, and winze 60 ft. deep.

Bousby, on Antimony Mountain, near the head of San Emidio Cañon.

Erskine Creek, several small mines which yield stibnite and native antimony.

Grace Darling, an old location near Erskine Creek.

Padre, in Tailhold Cañon, located in 1892.

San Emidio, in San Emidio Cañon; ore averages 40% Sb.

All of the above are in Kern County. Other undeveloped deposits exist in a granite formation in the mountains east of San Emidio Cañon.

Crowell, 5 miles southwest of South Riverside in Riverside County. An occurrence of stibnite in irregular bunches.

WORLD'S PRODUCTION OF ANTIMONY ORE, (*a*) IN METRIC TONS.

Year	Austria.		France.		Hungary.		Italy.		Japan.		Mexico.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
1891.....	333	$20,835	5,316	$133,518	.....	.....	782	.....	2,268	.....	2	.....
1892.....	97	5,646	5,103	135,200	853	$34,937	621	$45,672	2,814	.....	.....	.....
1893.....	441	24,339	7,171	130,000	881	32,160	1,193	40,402	1,649	.....	9	.....
1894.....	686	32,843	6,144	81,231	1,265	33,992	1,504	45,059	.....	.....	35	.....
1895.....	695	26,688	.....	.....	1,240	31,838	2,241	40,254	.....	.....	.....	.....

Year.	New South Wales.		New Zealand.		Portugal.		Queensland.		Spain.		United States.		Victoria.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
1891 .	930	$110,285	421	$24,750	.....	.....	220	$18,125	693	$20,805	635	$45,500	67	$5,940
1892....	740	73,400	370	24,500	.....	.....	26	1,390	395	12,467	744	51,000	299	11,390
1893 ...	1,803	125,460	336	17,335	666	$51,789	30	1,440	88	2,784	771	41,000	89	2,155
1894....	1,250	91,227	45	3,805	803	47,698	28	1,400	25	700	150	9,075	36	890
1895....	489	36,225	55	7,430	.....	.....	.....	.....	44	700	982	37,905	.....	.....

(*a*) The foreign statistics are derived from the official reports of the several governments; those for the United States were collected specially for THE MINERAL INDUSTRY.

The Assay of Antimony Ores.—Dr. John Clark calls attention to the inaccuracy of all fire-assay methods of estimating antimony,* and refers to Carnot's investigation of the subject,† wherein it was shown that results were rarely within 8 or 9% of the truth. Dr. Clark does not agree with Carnot's statement that results are uniformly low, owing to volatilization of antimony, and finds that such loss is apt to be compensated by iron and sulphur in the button. Thus one sample of ore containing 46.80% Sb by wet assay was reported at 49.0% and 52.0% by two chemists using the fusion method.

Dr. Clark recommends for antimony ores and alloys a modification of Mohr's method of titration with $n/10$ solution of iodine. I. In case of ore free from As and Fe dissolve in HCl, heat to drive off H_2S, add tartaric acid or Rochelle salt, make alkaline with $NaHCO_3$ and titrate. II. In presence of iron, dissolve in HCl, precipitate with H_2S, redissolve in HCl, and proceed as before. III. When ore contains arsenic, dissolve in HCl + Fe_2Cl_6 and distill off the arsenic; precipitate Sb with H_2S and proceed as per II. The arsenic distilled off may be condensed and estimated by titration with $n/10$ solution of iodine. IV. When ore or alloy contains tin, dissolve in HCl + Fe_2Cl_6, distill off As, precipitate Sb with metallic iron, redissolve (after washing) in HCl plus a little $KClO_3$, filter off insoluble impurities derived from the iron, precipitate Sb with H_2S, and proceed as per III.

Metallurgy.—A discovery of antimony ore carrying a notable amount of gold was reported at Randon, Nova Scotia. Ores of this character have been known for a long time in Portugal and Australia, and their treatment metallurgically has been a problem which is not yet solved. At the present stage of knowledge either the gold or the antimony must be sacrificed. Various processes for the recovery of both, some electrolytic and some purely chemical, have been proposed,

* *Jour. Soc. Chem. Ind.*, 1896, p. 255.

† *Annales des Mines*, Vol. I., p. 303.

and certain of them have been tried practically, but have failed through their cost or their complex character.

In the metallurgy of simple ores of antimony there has been no improvement; at least none made public. This business is in the hands of a very few concerns which employ few professional men, and there is consequently little inducement to exchange experiences in the columns of technical periodicals. The best references to the metallurgy of antimony in recent literature are the following:

1. THE MINERAL INDUSTRY, Vol. I., 1892, pp. 19–28; "Antimony, its Occurrence, Metallurgy, Uses, etc."; by Prof. J. F. Kemp.

2. *Ibid.*, Vol. II., 1893, pp. 13–24; "Antimony, its Ores, Occurrence, Metallurgy, Uses, and Production."

3. *Journal of the Society of Chemical Industry*, Jan. 30, 1892, pp. 16–19; "The English Process of Antimony Smelting," by Edward Rodger.

4. *Engineering and Mining Journal*, March 12, 1892; an abstract of the preceding.

5. *Annales des Mines*, Aug., 1892, pp. 163–191; "The Sale of Antimony Ore," by P. L. Burthe.

6. *Metallhüttenkunde*, by Carl Schnabel, Vol. II., 1896, pp. 444–486.

Electrolytic Antimony.—Electrolytic antimony is now beginning to appear in the market in Europe in the form of brittle, gray plates 2 mm. in thickness, of smooth, lustrous surface and silver-white color. The fracture is white and crystalline. An analysis of the metal as deposited showed: 0.008% Fe, 0.014% Na, 0.288% S, traces of Pb and Cu, and 99.690% Sb. After remelting: 0.0046% Fe, 0.0084% lead and copper, 0.0001% S, and 99.9869% Sb.*

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK, IN CENTS PER POUND.

Year.	Brand.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1891....	Cookson's.....	19.00	18.00	17.50	17.13	16.00	14.50	13.63	12.00	11.63	13.25	16.00	16.25	15.50
"	Hallett's.....	16.50	16.75	16.00	15.50	14.75	13.00	12.00	10.63	10.00	11.00	12.25	12.75	13.50
"	L. X.....	17.13	17.00	16.75	16.00	15.13	13.63	12.50	11.38	10.63	12.00	15.25	15.13	14.38
1892...	Cookson's.....	15.88	15.00	14.90	15.00	15.13	14.40	14.20	13.50	11.50	11.80	12.00	11.50	13.75
"	Hallett's.....	14.40	11.00	10.90	11.00	11.50	11.25	11.00	10.75	10.50	10.50	10.63	10.75	12.10
"	L. X.....	12.13	12.13	12.50	12.50	13.00	12.75	12.50	11.88	11.25	11.20	11.19	10.80	12.00
1893...	Cookson's	11.00	10.75	10.75	10.75	10.13	10.50	10.50	10.25	10.30	10.25	10.00	10.25	10.50
" ...	Hallett's	10.25	10.38	10.00	10.00	10.00	9.90	9.88	9.75	9.60	9.75	9.50	9.25	9.88
"	L. X.....	10.50	10.63	10.38	10.25	10.38	10.00	10.13	10.00	10.00	10.00	10.00	10.00	10.20
1894...	Cookson's.....	10.25	10.00	10.13	10.13	10.13	9.75	10.00	10.00	9.50	9.63	8.50	8.38	9.63
"	Hallett's.....	9.50	9.38	9.50	9.38	9.25	9.13	8.75	8.88	8.88	8.25	8.25	8.13	8.88
1895....	Cookson's.....	8.00	8.25	8.15	8.00	8.00	8.00	8.00	7.85	7.85	7.75	7.75	7.75	7.95
" ...	Hallett's.....	7.25	7.25	7.10	7.00	7.00	7.05	7.10	7.20	7.25	7.13	7.00	7.00	7.11
" ...	U. S. Star.....	8.25	7.50	7.38	7.50	7.35	7.50	7.50	7.75	7.50	7.50	7.50	7.50	7.56
"	Japanese.....	7.00	7.00	6.88	6.88	6.88	7.00	7.00	7.20	7.00	6.88	6.88	6.88	6.94
1896...	Cookson's.....	7.55	7.43	7.37	7.44	7.50	7.37	7.00	7.00	7.00	7.00	7.00	7.18	7.23
" ...	Hallett's.....	7.10	6.90	6.87	6.81	6.75	6.75	6.75	6.50	6.50	6.50	6.46	6.68	6.71
"	U. S. Star.....	7.25	7 09	7.00	7.00	7.00	6.91	6.75	6.75	6.75	6.75	6.75	6.93	6.91
"	Japanese	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75

PRICES OF ANTIMONY REGULUS PER TON IN LONDON.†

Month, 1896.	Highest.	Lowest.	Month, 1896.	Highest.	Lowest.	Year.	Highest.	Lowest.
January	£30½	£30	July...	£30	£30	1891	£72	£40
February.....	30	30	August	30	30	2	52	43
March.....	30	29½	September.....	29½	29½	3	43	37
April.....	30	30	October	29½	29½	4	37	32
May.....	30½	30	November	29½	29½	5	33	31½
June.....	30½	30	December	30	29½	6	30½	29½

* *Zeits. Elektrochem.*, 1896, II., pp. 524, 525.

† From James Lewis & Son's Monthly Report.

The New York Antimony Market in 1896.

The production of antimony in the United States during 1896 was 596 short tons, which is again an increase over 1895. A large part of the antimony refined came from foreign ores.

Almost throughout the year the market had a dull and declining tendency, and sales were very difficult to effect. We mentioned in our last year's report that the old English standard brands have more or less lost their hold on the consumers, mainly on account of the very stiff prices often asked for them and the great and increasing competition of new brands, which were also found to be of good quality, so that the cheaper prices at which they were offered induced a great many of the consumers to go over to them. This has been, to a less degree, the case with Italian and Hungarian antimony, but much more marked with Japanese, which has been found of excellent quality, and which, in larger quantities, has regularly been imported, and commands now a ready sale to the detriment of the English brands. It does not appear that consumption has materially changed, and we are probably not wrong in saying that it was not much larger in 1896 than in 1895.

At the opening of the year Cookson's antimony was held at 7¾c., United States French Star at 7¼c., and Hallett's at 6⅞c., but from month to month prices sagged off slowly, and by the end of June the finer grades especially had suffered on account of the competition of the Japanese, as well as of the United States antimony, which was greatly pushed for sale, and the prices then stood Cookson's 7⅛c., United States French Star 6⅞c., and Hallett's 6⅝c.

During the depression of business in the summer months a further decline took place, which culminated at the end of October, at which time the following rates had been established: Cookson's 6⅝c., United States French Star 6½c., Hallett's 6¼c. A somewhat better consumptive demand sprang up after the election had been decided, and the strenuous resistance made by the producers to further work at these figures, which evidently had proved very unremunerative, resulted in a slight improvement in the market, which was carried through, and at the end of the year the prices ruling were as follows: Cookson's 7@7⅛c., United States French Star 6⅞@7c., Hallett's 6⅝@6¾c.

The London Antimony Market in 1896.

The range of values during 1896 was about co-extensive with that for 1895, but the general tendency was toward lower prices than those which ruled in 1895, and which latter lay between the extreme limits of £33 and £31 10s.

January, 1896, opened with the article somewhat neglected at £30@£30 10s., and eased off during February to the extent of about 10s. per ton, recovering to the opening figure by the beginning of May. No change of consequence occurred until August, when the demand was particularly weak, and when the quotation sank to £29 10s. In September the quotation was still lower, and at the close of October it was quoted at £28 10s.@£29, the lowest point of the year, November bringing the value up again to £30@£31. Nothing fresh transpired during the month of December, and the market closed unchanged at about the same prices as those ruling during November.

ARSENIC.

THE world's supply of arsenic and arsenic compounds at the present time is obtained from Germany and England. Small quantities are produced in Italy and Spain, but these are consumed at home. None is made in the United States, although there is here an abundance of mispickel, which is the principal ore of arsenic, and in Canada there are large deposits. Most of these ores carry a little gold, and both elements of value—the arsenic and gold—are easily recoverable. The article immediately following describes the process employed for this purpose at Bovisa, Italy.

Besides the arsenic ore proper it would be possible to obtain arsenic as a by-product in many existing metallurgical processes. There are many gold and silver-bearing ores beneficiated in the Western United States which contain a little mispickel. It may be in very small percentage of the crude ore, but in the course of treatment it becomes concentrated in certain products which might be easily made available as sources of arsenic. Thus in roasting an accumulated lot of speiss at the Arkansas Valley Smelting Works, Leadville, Colo., several years ago, a good deal of trouble was experienced through poisoning the workmen by the rich arsenical fumes that were given off, and it became necessary to reduce the percentage of arsenic in the roasting charges. With such resources as these the United States ought to make its own supply of arsenic and arsenical compounds. The methods of arsenic-winning employed in Cornwall were fully described in THE MINERAL INDUSTRY, Vol. III.; those of Saxony in Vol. IV.

ARSENIC PRODUCTS IN GERMANY AND ENGLAND. (*a*)

Year.	Prussia. (*b*)		Saxony. (*c*)		Total Germany.		England.	
	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.	Metric Tons	Value.
1891.....	812	$41,379	1,342	$96,827	1,982	$138,206	6,147	$292,965
1892.....	592	22,739	1,170	86,388	1,667	109,127	5,196	218,430
1893.....	709	31,939	1,075	87,649	1,793	119,588	6,170	288,470
1894.....	1,147	57,377	1,084	108,511	2,376	165,888	4,877	243,070
1895.....	1,788	98,363	1,229	108,826	3,005	207,189	4,875	260,990

(*a*) Includes arsenious acid, red, yellow and white glass, and metallic arsenic.
(*b*) From *Zeitschrift für das Berg- und Hüttenwesen im Preuss. Staat.*
(*c*) From *Jahrbucher für das Berg- und Hüttenwesen im Königreiche Sachsen.*

The Gold-Arsenic Works at Bovisa, Italy.

By F. Clerici.

The works at Bovisa (Milan), belonging to the Societa Anonima Ingeniere L. Vogel, are among the most complete of their kind in Europe. The principal object of the company is the production of mineral superphosphate. The phosphates, amounting to about 75,000 tons yearly, are mostly imported from South Carolina, and about the same quantity of sulphuric acid is produced on the premises for converting the phosphates into superphosphates.

For the production of sulphuric acid about 25 tons per day of iron pyrites are burnt in *Etagen-Ofen*. Of these 25 tons 15 tons come from the mines of Agordo (Udine), containing from 2 to 3% of copper, which is saved by lixiviation after roasting. The remaining 10 tons come from the Cani mine, in the neighborhood of Mt. Rosa; this mine is one of the group belonging to the Pestarena mining camp, and has been leased by the Societa Anonima L. Vogel. Worked for free gold since the time of the Romans, this mine constantly gave good returns up to the beginning of this century, when the ore having suddenly become very refractory the mine was shut down. It was re-opened about 25 years ago by the Pestarena Mining Company, and a large mill was erected for amalgamation with arrastras. But the ore carried too much iron pyrites and arsenic and no satisfactory results could be obtained.

In 1887 the mine was leased by the Societa Anonima L. Vogel, and I was sent as manager to see how the mine and the ore could be treated to best advantage. The problem was very complex, and I saw that the only way to make the concern pay was to utilize all that the ore contained—sulphur, arsenic and gold. The plan adopted, and which has been carried out ever since, is as follows:

The ore, containing enough iron pyrites to burn by itself in the *Etagen-Ofen*, is sorted out and sent to the works at Bovisa (distance, 5 miles by wagon and 65 miles by railroad): the poorer ore to be treated at the mine with cyanide.

The ore going to Bovisa contains an average of 34% sulphur, 10 to 12% arsenic, 0.6 to 0.7 oz. gold and 2.5 ozs. silver per ton. This ore is subjected to a first roasting in *Etagen-Ofen*, where it is freed from the sulphur and arsenic. Attached to the roasting furnace are two series of large leaden canals (labyrinth) through which the sulphurous and arsenious acids are made to pass. The sulphurous acid after passing the labyrinth goes to the lead chambers to be converted into sulphuric acid; the arsenious acid condenses in the labyrinth and deposits itself in the form of a reddish-white slime.

These slimes are taken from time to time from the above-mentioned labyrinth and dried, in order to be subsequently subjected to a last sublimation, so as to obtain a white commercial product 99% pure. So far no difficulty was met with, but as soon as we began the sublimation in a kind of *Muffel-Ofen* our troubles began. The slimes coming from the labyrinth were found to contain some free sulphuric acid and some very fine particles of oxide of iron. Owing to the presence of these two substances we found it absolutely impossible to obtain by sublimation a pure product; the arsenious acid obtained had a faint reddish tint; its purity was never more than 90%; it was therefore not salable. It was evident that, before trying the sublimation, it was absolutely necessary to get rid of the

free sulphuric acid present. We devised a large agitator in which the slimes were placed and agitated with water, which was siphoned off after letting the slimes settle, and the operation was repeated three or four times. This treatment, although tedious and expensive, did not give the wished-for results, although a considerable improvement was shown. After trying a few more schemes unsuccessfully, a method was at last devised which was the simplest of all, and which, as is always the case, should have occurred to us at first. A large rectangular vat of bricks with a false bottom was built, with a quartz filter on its top, covered with perforated sheets of lead. The slimes coming from the labyrinth are thrown into this vat, and spread evenly all over the filter; then water is allowed to percolate slowly through the mass; the free sulphuric acid present being thus carried out through the false bottom, and recovered by pumping it back to the lead chambers.

The slimes thus washed gave the most satisfactory results, and in one single sublimation a beautifully white product containing 98 to 99% of pure arsenious acid was obtained. In England, and even at Freiberg, two sublimations are necessary to arrive at a commercial product.

Before this industry—new for Italy—began the arsenious acid used in that country was all shipped from England, but in a year's time the English product was completely shut out from the market, the Bovisa works supplying all demands in Italy, amounting to about 400 tons a year. The greater part of this product is used in glass manufacture and in tanning leather. The product is sold in Italy at 38f. per 100 kgs. (about 3.3c. per lb.). In this manner the sulphur and the arsenic of our ore are well disposed of.

Our study was then drawn to the recovery of the gold contained in the roasted pyrites. This product still contains 1.5 to 2% of sulphur and about 0.5% of arsenic. In order to eliminate these substances—injurious to the subsequent operation of chlorination—the roasted pyrites undergo a second thorough roasting in another *Etagen-Ofen* quite similar to the first one, but provided with a fireplace, where a lively fire is made in order to drive out the last traces of sulphur and arsenic. The pyrites thus dead roasted are cooled and then charged into large wooden rectangular vats, lined with lead, and holding 10 tons each. These tanks are provided with a false bottom, covered with a quartz filter, upon which the mass rests. A weak solution of chloride of lime, together with a weak solution of sulphuric acid, is allowed to percolate slowly through the mass, dissolving and carrying with it the gold, the clear solution passing through the false bottom to the precipitating tanks. One vat of 10 tons is completely lixiviated and washed out in three days.

The extraction of the gold ranges between 85 and 87%. We do not attempt to save the silver, as it would hardly pay the expenses of extraction. In this manner a mine, which, owing to the very refractory nature of the ore, was almost worthless 10 years ago, is now giving a very good profit.

ASBESTOS.

ASBESTOS was produced in the United States in 1896 in Georgia and Washington. In Georgia the Sall Mountain Asbestos Co. installed new machinery and made an increased output. Its product is chiefly short-fiber mineral. A small amount of asbestos was produced at Lyman in Skagit County, Washington. In Wyoming the McConnell Asbestos Co. continued development work on its property at Casper Mountain, Natrona County, but made no shipments. The Wyoming mineral is of the chrysotile variety, occurring in serpentine; the Georgian is true asbestos, and is found in association with soapstone. No production was reported in 1896 from Virginia, California, South Dakota, and Oregon, where deposits have been worked on a small scale at various times.

A new preparation now made at Danville in Quebec is called "asbestic." In its manufacture the lower grade or short-fiber asbestos is utilized. It is an asbestos-plaster intended for the walls of buildings in place of ordinary plastering. Experiments have shown that it will resist a high degree of heat successfully, and that woodwork properly covered by it will be protected against any ordinary fire. This material is being gradually introduced, and its production is becoming an industry of some importance.

Herr L. Frobeen, of Berlin, Germany, is making a new fireproof paper. By his process 95 parts of asbestos-fiber of the best quality are washed in a solution of permanganate of calcium and then treated with sulphuric acid, the fiber being thus bleached. Five parts of ground wood-pulp are then added, and the entire mass placed in an agitating-box, together with limewater and borax. After being thoroughly mixed the material is pumped into a reservoir, whence it is drawn off to the usual paper-making machinery. Paper produced in this way, it is reported, may be placed in a white-hot fire with impunity.

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.				Imports.		
	Short Tons.	Metric Tons.	Value.	Value per Metric Ton.	Manufactured.	Unmanufactured.	Total.
1890.....	71	64	$4,560	$71.25	$5,342	$252,557	$257,879
1891.....	66	60	3,960	66.00	4,872	353,589	358,461
1892.....	100	91	5,000	54.94	7,209	262,433	269,642
1893.....	120	109	6,000	55.00	9,403	175,602	185,005
1894.....	265	240	4,300	18.91	15,989	240,029	256,018
1895.....	664	602	11,837	19.66	19,731	225,147	244,878
1896.....	716	650	12,670	19.49	15,654	229,084	244,738

Asbestos in Quebec.

By Matthew Penhale.

In Quebec asbestos is found in the belt of serpentine rocks extending across the province from the St. Lawrence to the Vermont line. It is not found everywhere in this belt, and the deposits worked occur in four groups: at Black Lake, at Metford, and at Broughton on the line of the Quebec Central Railway, northeast from Sherbrooke, and near Danville on the Grand Trunk Railway. Some other deposits are known to exist, but as the existing mines fully supply the present demand, there is little prospecting, and the opening of new mines is rare. As the demand grows, however, it is probable that more work will be done in this direction.

Methods of Mining.—There has been no change in the methods of mining. With one exception all the Quebec mines are open pits or quarries of considerable extent. The asbestos is taken out in connection with the country rock or waste, a large quantity of which has to be handled. Blasting must generally be avoided, as it is apt to break up and destroy the fibers, and it is the length and fineness of these upon which the value of the asbestos chiefly depends. Most of the mines are supplied with hoisting-engines and derricks for lifting the mineral and rock out of the pits. Pumping is required only occasionally. The appliances are largely of the same kind as those found in quarries.

The exception in Quebec is the Broughton mine, which is owned by the Glasgow & Montreal Asbestos Company, but at present leased and worked by the United Asbestos Company. Here there is a well-defined vein of asbestos running northeast and southwest on the parting between the serpentine and slate. The working is altogether underground. Three shafts have been sunk and drifts on them have been run on the vein. The mineral is taken out by stoping, and is generally of a very fine quality, the fibers being 6 or 7 in. long. The mine is provided with hoisting and pumping machinery. At most of the mines the asbestos is found in bunches or pockets, and Broughton is the only instance of a regular vein. It may be mentioned that the vein carries also soapstone of very fine quality.

Preparation for Market.—The preparation of the asbestos consists in breaking up the rock in which it is enclosed and in freeing the fiber from adhering particles of rock and dust. This is done in rock-breakers and special dressing-mills. In nearly all the Quebec mines there has been a great improvement in methods during the past two years, and the asbestos is shipped in much cleaner and better condition than formerly. This was rendered necessary by competition, and has been carried so far that a ton of asbestos shipped now is equal in value to nearly twice the quantity a few years ago.

The largest and best cleaning plant, furnished with the latest improvements, is that of the Danville Asbestos and Slate Company, at the Danville mines. Here there is a large four-story mill. The crude material, lifted from the open mine by derricks, is delivered on the upper story of the mill to the crushers. It passes by gravity from floor to floor through the separators, the handling being almost entirely by machinery. It is finally delivered on the lower floor to the packers, separated into three grades according to fiber, and free from sand and grit. At this mill there is also machinery for working up the low-grade or short-fiber

asbestos into the asbestic or asbestos plaster mentioned above. The latter has only recently been introduced. Negotiations are pending for its use in some of the buildings for the Paris Exposition of 1900.

Production and Shipments.—The shipments, as obtained from the Grand Trunk and Quebec Central returns, were 10,380 tons (of 2000 lbs.) divided as follows:

From:	Tons.
Danville Mines	4,939
Thetford, Bell's Mine	1,886
Thetford, Johnson's Mine	1,632
Thetford, King Brothers' Mine	845
Thetford, Ward Mine	20
Black Lake, all mines	1,008
Broughton Mine	50
Total	10,380

This does not include the short-fiber asbestos shipped from the Danville mines in the form of asbestic. The total is the largest quantity ever reported in one year.

The values are not reported. The Canadian custom-house returns give the average value of asbestos exported at about $50 per ton, but this is probably too high, and the average received at the mines is not over $30 per ton.

The Danville mine, though comparatively new, is the largest shipper and was worked steadily throughout the year. At Black Lake comparatively little was done. The American Asbestos Company's mine was the only one worked steadily. The United Asbestos Company's and the Anglo-Canadian Company's mines were idle. The Glasgow & Montreal Company's mine was worked a short time, but the lessee failed to comply with agreements, and it was closed again. At Thetford the mines, with the exception of the Beaver, were at work. The Broughton mine was closed for most of the year and only reopened toward its close.

The deposit at Thetford is the largest body of asbestos yet discovered in Quebec. At Black Lake there are abundant deposits, and the production at both these places, and at Broughton also, can be increased largely when the demand is sufficient to warrant the extension of the workings.

The destination of the shipments was as follows:

To United States:	Tons.
Erie, Pa.	1,030
Franklin, Pa.	57
Lockland, Ohio	56
New York	3,926
Total	5,069
Bromeville, Que.	20
Quebec and Montreal	539
Other countries	4,752
Total	10,380

Other Countries:	Tons.
London	588
Liverpool	233
Glasgow	407
Havre	280
Antwerp	556
Rotterdam	809
Hamburg	1,809
Genoa	70
Total	4,752

The asbestos reported shipped to Quebec and Montreal was all intended for export. The mineral sent to New York was all for the H. W. Johns Company.

It is worthy of note that one shipment of Canadian asbestos was last year made to Italy, which formerly furnished the entire commercial supply. Germany is the largest foreign customer, most of the asbestos shipped to Rotterdam being for German points, as well as that sent to Hamburg. The United States is the best customer of the Quebec mines, taking about 49% of the whole.

The stocks at nearly all the mines are now large, and no considerable increase in mining can be expected for the present.

ASPHALTUM.

THE production of asphaltum and bituminous sandstone in the United States fell off heavily in 1896; asphaltic limestone showed an increase. Asphaltum was mined in California and Utah; asphaltic limestone in Utah, Texas, and the Indian Territory; bituminous sandstone in California, Kentucky, and the Indian Territory.

PRODUCTION OF ASPHALT AND BITUMINOUS ROCK IN THE UNITED STATES. (IN SHORT TONS.)

States.	1895.			1896.		
	Tons.	Value.	Value Per Ton.	Tons.	Value.	Value Per Ton.
Bituminous Sandstone:						
California	38,921	$121,586	$3.04	50,000	$125,000	$2.50
Kentucky	2,359	11,795	5.00	2,500	10,000	4.00
Utah	1,094	6,564	6.00	619	3,714	6.00
Totals	42,374	$139,945	$3.30	53,119	$138,714	$2.60
Asphaltic Limestone	a5,550	16,650	3.00	a3,119	8,714	2.80
Asphaltum: California	25,525	170,500	6.68	20,500	66,900	3.26
Gilsonite and Bitumen	a1,150	34,500	30.00	a1,700	54,500	32.06

(a) Product of Texas and Utah.

Utah.—The output of gilsonite, or glance pitch, is included under the heading asphaltum. This substance is mined in Utah in the neighborhood of Price and Fort Duchesne, and is practically the only asphaltum product of Utah which will stand the freight-rate on shipments beyond a comparatively circumscribed area. It is mined by several companies and shipped chiefly to St. Louis, where it is used in the manufacture of varnishes and similar products. The Gilson Asphaltum Co., which is the principal producer, is said to have shipped upwards of 1000 tons of the mineral from its mines near Duchesne in 1896. At the end of November, however, a serious explosion occurred in the mine, which wrecked the plant and interrupted the output.

The gilsonite deposits of Utah have been described by Geo. H. Eldridge in a monograph recently issued by the Geological Survey, of which the following notes are made: So far as yet discovered gilsonite (uintaite) is confined to the Uncompahgre Indian Reservation and immediate vicinity. The elaterite deposits lie chiefly in the southern part of the Uinta reservation. Ozokerite occurs in the vicinity of Soldier Summit, a station on the D. & R. G. Western Railway. Maltha is reported at a number of isolated points. The asphaltic limestones thus far exploited occur in the southern spurs of the divide between Strawberry and Soldier

creeks, about 7 miles northwest of Tucker. The asphaltic sandstones outcrop in many valleys, notably the Ashley and the Nine-Mile; and the bituminous shales and limestones are broadly distributed.

The larger veins of gilsonite are somewhat scattered, one lying about 3½ miles east of Fort Duchesne, a second in the region of Upper Evacuation Creek, and the three others of chief importance in the vicinity of White River and the Colorado-Utah line. The region is deficient in transportation routes. The product of the Duchesne and Cŭlmer mines is taken by wagon to Price on the D. & R. G. Western Railway, which costs $12 @ $15 per ton, to which has to be added railway freight to Chicago or St. Louis, $9 per ton. The mines on the Colorado-Utah line now ship their product by way of Meeker to Rifle on the D. & R. G. Railway in Colorado, a distance of 125 miles. The present value of the best grade of gilsonite in Chicago and St. Louis is $40 @ $50 per ton.

Texas.—Under the heading of asphaltic limestone is included the output of "mastic," which is prepared at Uvalde, Texas. The bitumen which is extracted by naptha is reckoned with the Utah gilsonite.

California.—In California the chief event of the year was an association between the Alcatraz Asphalt Co. and an English syndicate, whereby it is thought a market for the product will be opened in Europe. According to the *Engineering and Mining Journal*, Oct. 31, 1896, this company owns three great asphalt deposits known as Las Conchas, La Patera, and the Sisquoc Grant. The Las Conchas deposit, at Carpenteria, is very pure, producing a liquid asphaltum containing 95% bitumen. It covers about 75 acres, and is estimated to be 25 ft. thick. The works, which have a capacity of 75 tons per 24 hours, employ 30 men, while 30 men are also employed in the mine. The La Patera mine, located on the Don Ranch on the coast, 10 miles west of Santa Barbara, is worked by four shafts and a tunnel. Steam hoisting-works and pumps are used and 23 men are employed. The deposit is from 2 ft. to 12 ft. in thickness. The Sisquoc Ranch, 8 miles north of Los Alamos, contains 35,485 acres. Upon it are two deposits of asphalt. That known as the Brea is 10,500 ft. in length, with an average width of 500 ft. and depth of 300 ft. What is known as the Mesa deposit lies on a plateau 300 ft. above the valleys, which surround it on three sides, and at an elevation of 1900 ft. It is about 5000 ft. in length, with an average width of 600 ft. and depth of 100 ft. Refining works with a monthly capacity of 3000 tons are in course of erection. The plan in contemplation is to separate the asphaltum from the sand in the crude material by solution in naphtha. The solution will then be piped to Gaviota, a distance of 30 miles, where the naphtha will be distilled and pumped back for use again. This is certainly a novel idea, and its outcome will be awaited with interest.

The production of asphaltum in California in 1895 (according to the returns to the State Mining Bureau) was 25,525 short tons, valued at $170,500; of bituminous rock 38,921 tons, valued at $121,586. In 1896 the production of asphaltum was 20,500 short tons, valued at $66,900.

Kentucky.—Marshall Morris, in a paper read before the Engineers' and Architects' Club of Louisville,* stated that bituminous rock is found in the Chester group of the Subcarboniferous rocks along the eastern and southern edge of the

* *Engineering and Mining Journal*, Jan. 9, 1897.

Western coal-field of Kentucky, and particularly in what is known as the "third sandstone," in Owen's Geological Reports of Kentucky. It has also been found in the conglomerate sandstone of the coal measures, but under heavy cover, and has not been developed. Deposits are found in the counties of Breckenridge, Grayson, Edmonson, and Logan; but the Breckenridge Asphalt Company, having its mines near Garfield, in Breckenridge County, is the only one which has made a commercial success, though several hundred tons have been quarried and used in putting down small sample pavements by the Logan County Company. The Grayson County Company was first in the field, but its property has not been operated for several years.

As to its origin, geologists differ; but there is little doubt that the beds of bituminous rock were once oil-bearing sands, in which only the residuum, metamorphosed into asphaltum, now remains. The impregnated rock is found in the shape of pockets or basins, increasing in thickness from the rim to the centre. The greatest thickness is 10 ft.

The impregnation shades off gradually into the sandstone, or sometimes breaks off abruptly; it is most economically mined by stripping off the overlying sandstone and partially impregnated rock. The sandstone is a fine-grained white sand, nearly pure silica, and impregnated with as much as 12% bitumen. The average good material contains about 8%. The composition of the residue, after the bituminous substances have been removed, is as follows: Silica, 96.88%; sesquioxide of iron, 0.81%; alumina, 0.46%; lime, 0.34%; magnesia, 0.20%; soda, 0.81%; potash, 0.20%; combined water and loss, 0.25%; total, 99.95%.

The treatment of the material for road-making is similar to that of European asphalt rock, except that on account of the gummy nature of the bitumen and the sharpness of the sand it is much more difficult to disintegrate. The sand is so sharp that a set of chilled-steel rolls which the Buffalo Company first attempted to use were rendered worthless in less than 60 days. The rock is now ground in mills especially designed for the work, consisting of a horizontal plate to which raised lugs are attached, revolving at a high rate of speed; the rock being broken by impact, is carried by centrifugal force through a screen surrounding the mill. So sharp is the sand that these screens, though made of steel 1 in. thick, are worn out in grinding about 1000 tons of rock, and the lugs on the revolving plate have to be renewed every 24 hours. The powder, after leaving the mill and passing through a second screen, is carried by elevators to revolving heaters, and thence to the street, where it is spread and compacted with tampers, as in the case of the European rock asphalts.

The whole operation of grinding and heating is automatic; the rock is not touched by hand from the time it is loaded into elevators to be conveyed to the mills, until it is dumped on the street to be spread and compacted. The only skill required about the plant is in the selection of the rock, while in the case of compounds of Trinidad or other bitumen much depends on the care with which the heating or cooking is conducted, the same workmen being likely to turn out from day to day material of altogether different values as wearing surfaces for streets.

The Kentucky bituminous rock will make a good pavement by itself, as is shown by the pavement laid on Jefferson Street in Louisville, and by the first

streets laid in Buffalo, N. Y. The pavement so laid is soft, and is readily torn by the toe calks of horses, and indented by the wheels of vehicles. These marks disappear under continued traffic, but it was shown that at least 12 months was required to produce a permanent road, and that the appearance of the street in the meantime was in no manner conductive to popularity. Numerous experiments were made to produce a hard pavement; limestone was added to absorb the light oils, but the result of adding this dead material was to weaken the adhesiveness of the pavement. The best result was obtained by the addition of Vorwohle rock from Germany, the proportions found best by experience being 20 to 30% of the Vorwohle rock, and 80 to 70% of the Kentucky rock.

On January 1, 1897, there were laid in the United States, as nearly as could be ascertained, 27,468,915 sq. yds. of asphalt pavement, divided as follows: Trinidad asphalt, 21,527,415 sq. yds.; Bermudez, 697,500; Alcatraz (California), 916,000; Standard (California), 200,000; foreign rock asphalt, 603,000; Utah rock asphalt, 293,000; other asphalt in the Pacific States (estimated), 1,032,000; asphalt block, 2,200,000.*

France.—Asphaltum has recently been discovered at Crédo, near the Swiss frontier. The deposit lies midway between that of Seyssel in France and that of Val-de-Travers in Switzerland.

Mexico.—A large deposit of asphalt in the Sota de la Marina district, Tamaulipas, Mexico, has been described as follows in a recent U. S. Consular Report: There are about 20 wells, which have a continuous flow of mineral tar. The ground for over 20 miles square is covered with asphalt deposits. The near-by streams have the whole year round a fatty, oily substance flowing into them from the surrounding mountains, making the water unfit to drink. A concession for the working of these fields for 50 years has been obtained, and a company is now being organized in the United States to commence work, put machinery on the ground, and drill for oil. The district being situated near the coast, transportation can be effected by water. The river La Marina, which flows through the district, is navigable for vessels of 30 tons capacity.

THE WORLD'S PRODUCTION OF ASPHALTUM AND ASPHALTIC ROCK. (IN METRIC TONS.)

Year.	Austria.	France.	Germany.	Italy.	Russia.	Spain.	Trinidad.	United States.
1891	180	260,626	49,150	28,180	140 394	312	97,725	40,884
1892	78	224,000	53,279	34,580	150,000	668	113,200	42,675
1893	88	222,000	47,238	25,980	155,000	820	93,090	31,655
1894	116	230,603	55,981	60,493	160,544	985	93,965	35,098
1895	404	.....	59,563	45,713	.....	790	87,531	62,868

Ozokerite.—There was no production of ozokerite in the United States in 1896. The only domestic source of this substance is Utah, whence cost of transportation is so high that it cannot compete with the Galician mineral, which is freighted from Boryslaw to New York for 35 cents per 100 lbs. The Boryslaw mines were sold during the year to a Vienna syndicate, which aims to control the entire production, and consolidate and systematize their exploitation. The average market price of ozokerite in New York in 1896 was 6 cents per pound.

* Geo. W. Tillson, "Asphalt and Asphalt Pavements," *Proceedings of the American Society of Civil Engineers*, April, 1897.

BARYTES.

THE barytes industry suffered severely from the depression of business in 1896. In fact the paint trade as a whole was dull, and practically the only use for barytes is in paint-making. The American producers suffered not only from the decreased demand for paints but also from the encroachments of foreign producers, who have been shipping hither large quantities of both crude and manufactured barytes, on which the duties were reduced by the Wilson Act. The foreign barytes comes principally from Germany, being carried from Hamburg and Bremen to New York for very low freights, occasionally as low as $1.50 per long ton, while Virginia barytes cannot be transported to New York for less than $2.50 per ton, and the rate on Missouri mineral is, of course, much higher. The barytes industry has, moreover, to meet the competition of "lithophone," of which the importation into the United States is increasing. This is an artificial mixture of zinc sulphide and barium sulphate, prepared by adding a solution of barium sulphide to one of zinc sulphate, whereby zinc sulphide and barium sulphate, both insoluble white precipitates, are formed. This pigment is made only in Germany, and probably cannot be made to so good advantage elsewhere; at least not without an equally cheap supply of zinc sulphate.

Barytes, which was at one time regarded contemptuously as merely an adulterant for white lead, is now recognized as a legitimate pigment, specially valuable for certain purposes, and recommended by the most reputable dealers in paints. It is generally employed in conjunction with zinc-white, and when not used in improperly large proportion produces a paint which will undoubtedly withstand certain atmospheric conditions better than lead-white, and is nearly equal to it in covering power. Barium sulphate also forms a valuable base for colored pigments, being chemically inert, or practically so. There is no doubt that barytes is still sold under the name of "white lead" by disreputable concerns, but the fraud can be so easily detected that no one should be deceived. Thus if the paint is mixed with oil it is sufficient to burn off the oil and test the residue with nitric acid, in which zinc-white and lead-white are soluble, while barytes is insoluble.

The following table gives the production and imports of crude barytes in the United States. With respect to quantities this practically represents also the finished product, since there is no important loss in manufacturing.

Barytes is produced in the United States to a large extent by persons, farmers and others, who do not make a regular business of it, and the only reliable

method of arriving at the output is from the concerns buying it, which are few in number. Several of the latter have, however, refused to make us returns this year, wherefore we have been obliged to estimate their purchases, basing these estimates on the best information obtained indirectly. In order that possible errors may not be embodied permanently in the statistics we put on record the estimates as follows: Dingee, Weinman & Co., Lynchburg, Va., 6500 tons; Page & Krause Manufacturing Company, St. Louis, Mo., 3500 tons; Mepham & Klein, St. Louis, Mo., 5000 tons.

PRODUCTION, IMPORTS, AND CONSUMPTION OF BARYTES IN THE UNITED STATES.

(In tons of 2000 lbs.)

Year.	Production.			Imports.			Consumption.	
	Quantity.	Value Per Ton.	Value.	Quantity.	Value Per Ton.	Value.	Quantity.	Value.
1892.....	28,476	$5.00	$142,380	4,667	$4.88	$22,837	33,143	$165,217
1893.....	26,632	5.00	133,360	4,459	4.21	18,791	31,091	152,151
1894.....	23,758	4.00	95,032	2,720	5.80	15,826	26,478	110,858
1895.....	20,255	4.89	99,020	4,680	5.27	24,673	24,935	123,693
1896.....	29,500	4.00	87,600	3,333	7.38	24,619	25,233	112,219

During the hard times of 1896 two of the Virginia producers were obliged to suspend operations. The New Haven mill, which used to grind foreign barytes chiefly, also went out of business. Prices ranged very low, best Missouri floated mineral having been quoted in New York at $14.50@$15; while at one time Virginia mineral was offered at $13, which was probably below the actual cost of production. Toward the end of the year, however, there was an improvement and it became impossible to buy any kind of A1 barytes for less than $15 per ton of 2000 lbs.

BAUXITE.

By Wm. M. Brewer.

In many respects the conditions of the bauxite mining industry were the same in 1896 as in 1895. It is true that the output decreased slightly, but this fact was more than offset by the satisfactory results obtained from the export trade which was begun during the summer. The fact that Southern bauxite found such favor in Germany as to create a demand where it came in direct competition with the Irish and French ores suggests that the claims which have been made as to superior grade were well founded. A contract was recently closed with the Southern Bauxite Company for 3000 tons for export, from the Dykes district in Cherokee County, Ala., the mining of which will commence early in 1897.

The only new developments during 1896 were those started by the Alabama Mineral Land Company of Anniston, which did considerable prospecting work on a deposit of bauxite discovered almost within the city limits of the town of Anniston. By analysis the ore proved to be of equally good grade as that found in other portions of the belt in Georgia and Alabama, but the extent of the deposit has not yet been proved.

The Knox dolomite formation in which the bauxite deposits are found is somewhat greater in thickness in Calhoun County than in Cherokee and Cleburne counties to the northeast.

The discovery of this deposit in Anniston is in accordance with the conclusions arrived at by Dr. C. W. Hayes of the United States Geological Survey, as well as by Dr. Eugene A. Smith and Professor McCalley of the Alabama Geological Survey, who have always been of the opinion that there was probability of new discoveries of deposits of bauxite being made further to the southwest than had been made previous to 1896, the same geological conditions prevailing for a considerable distance in this direction from the deposits which were discovered in 1890 and 1891 in Cherokee County.

The fact has been clearly demonstrated that the bauxite industry is by no means of the purely speculative character which it was supposed to possess when first inaugurated. The deposits which have been worked, especially in Cherokee County, Ala., have proved to be more extensive than was generally thought would be the case.

It is interesting to note in connection with the discovery near Anniston, that it

has been claimed for some years that the first discovery of bauxite in Alabama was made in 1888 in Calhoun County near Weaver's Station on the East Tennessee, Virginia & Georgia Division of the Southern Railway, and that at that time the attempt was made to use it for a lining in one of the charcoal furnaces of the Woodstock Iron Company. No efforts have been made, though, to mine this ore on a commercial scale or indeed to even develop the discoveries made in any portion of Calhoun County, except at Anniston.

In estimating the supply of bauxite in Georgia and Alabama it must not be forgotten that the production hitherto has been almost entirely from two districts in Georgia, viz., the Ridge Valley and Barnsley, which embrace a total area of probably five or six miles square on the eastern border of Floyd County and the western border of Bartow County, and from one district in Alabama, namely, the Dykes in Cherokee County. When we consider this fact and, further, that the large area embraced by the Vann's Valley deposits and the Cave Springs deposits in Floyd County, Ga., has been hardly touched, it must be admitted that there is little cause to apprehend that the supply of ore will fail to meet the demands for many years to come. There are also deposits in Chatooga and Walker counties in Georgia which have been merely prospected, as well as in the district southwest of Cherokee County in Alabama not yet exploited.

The most important advance in bauxite mining industry in 1896 were the devices put in operation by the three producing companies, the Republic Mining and Manufacturing Company, the Southern Bauxite Company, and the Georgia Bauxite Company for drying the ore previous to shipment, which formerly was left to the sun. The Georgia Bauxite Company erected in 1895 a Colby-Davis roaster for this purpose, but it was found that several modifications and alterations had to be made before its work was perfectly satisfactory. The other companies have erected home-made kilns which, I am informed, serve in a very satisfactory manner the purpose for which they are constructed.

PRODUCTION, IMPORTS, AND CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.				Imports.		Consumption.	
	Alabama.	Georgia.	Total.					
	Long Tons.	Long Tons.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1892	4,900	4,900	9,800	$49,000	5,716	$57,948	15,516	$106,948
1893	7,063	3,965	11,028	55,205	5,103	28,217	16,131	83,422
1894	8,687	2,045	10,732	42,928	1,028	6,661	11,760	49,589
1895	14,000	4,800	18,800	56,400	5,797	34,782	24,597	91,182
1896	9,796	7,300	17,096	68,384	2,119	10,477	19,215	78,861

The principal foreign sources of bauxite are in France and Ireland. In the former country they occur in the departments of Var, Bouches de Rhone, Alpes Maritimes, Hérault, and Ariège, Messrs. Augé & Co., of Montpellier being the largest producers. The French bauxite deposits, which have been worked since 1872, are of massive conglomeritic or oolitic structure, occurring in Tertiary and Jura-Triassic formations. The red bauxites of Var average 60% alumina, up to 3% silica and 18% ferric oxide; the white bauxites of Ville-Veyrac Hérault contain 60% alumina, up to 20% silica and 3% ferric oxide.

The principal Irish bauxite deposits are situated on the land of the Marquis of Downshire at Strain, near Ballyclare, and at Glenravel, Antrim. The former are exploited by Messrs. Geo. G. Blackwell & Co., of Liverpool, and the latter by the British Aluminum Company.

Recently large deposits of bauxite have been opened in Styria, and are now being worked. It is a denser, more earthy mineral than the French bauxite.

Professor Helmhacker of Prague describes a deposit of bauxite at Langsdorf, near Likh, in Upper Hessia, where basalt is covered with red clay in which occur nodules of bauxite, sometimes arranged in strata. An analysis by Lang of a light red variety showed SiO_2, 10.3%; Al_2O_3, 49.0%; Fe_2O_3, 12.9%; CaO, 0.6%; MgO, trace; Na_2O, 0.2%; CO_2, 0.3%; P_2O_5, 0.4%; H_2O by ignition, 25.9%; H_2O at 100° C., 0.9%—total, 100.1%.

The bauxite deposits of the world are described in previous volumes of THE MINERAL INDUSTRY, and in the following special articles:

F. Laur, "Bauxite," *Transactions of the American Institute of Mining Engineers*, Vol. XXIV., 1894, p. 234.

A. E. Hunt, "The Manufacture of Aluminum by Electrolysis at Niagara," *Proceedings of the Institution of Civil Engineers*, Vol. CXXIV., 1895–96, Part II.

Henry McCalley, "Bauxite," THE MINERAL INDUSTRY, Vol. II., 1893.

Chas. Willard Hayes, "Bauxite," *Mineral Resources of the United States;* Metallic Products; 1894–95, Vol. III.

THE MANUFACTURE OF ALUMINA AT LARNE.

The manufacture of alumina at Larne (near Belfast), Ireland, by the Bayer process is described by J. Sutherland.* The raw material is bauxite from County Antrim, averaging 56% Al_2O_3, 3% Fe_2O_3, 12% SiO_2, 3% TiO_2, and 26% H_2O. It is crushed to pass a ¼-in. sieve, and calcined in an Oxland & Hocking cylinder, an excessive temperature being avoided lest the alumina be rendered insoluble. The calcined mineral is treated with caustic soda solution of 1.45 sp. gr. in pressure keirs made of ⅝-in. steel plate, 11 ft. long and 5 ft. diameter, a 3-in. horizontal shaft with eight 16-in. by 9-in. paddles (agitators) passing through stuffing-boxes in the ends. The soda solution is first introduced and the ore (about 3 tons) is added subsequently, the paddles being in motion to insure thorough mixture. After closing the charge-door steam is turned on and the pressure raised gradually to 70 or 80 lbs., at which it is maintained two hours, when decomposition is complete. The discharge-cock being opened the mud and solution are raised to tanks at the top of the building by the pressure in the keir. Water is added to reduce the specific gravity of the liquid to 1.23, and the solution of sodium aluminate is separated by means of filter-presses, the residue being washed and thrown away. The solution is filtered again through wood-pulp contained in lead-lined vats 10 by 6 by 3 ft., the filter being placed 6 in. above the bottom. The filter consists of a ⅛-in. screen, on which is run a thin paste made by boiling 50 lbs. of wood-pulp with water. The pulp soon settles evenly over the sieve, forming the filter proper. The sodium aluminate solution, having

* *Inst. Mech. Eng.*, 1896; *Engineering*, 1896, 62, 291, 292; *Jour. Soc. Chem. Inst.*, Oct. 1896, 722; *Engineering and Mining Journal*, Oct. 3, 1896.

been purified, is run into tanks 13 ft. in diameter, 20 ft. deep, provided with agitators. The precipitation of aluminum hydrate is effected by an excess of the same substance. Within 36 hours' agitation 70% of the alumina is precipitated. The precipitate is then allowed to settle, and the supernatant liquor is drawn off to storage-tanks, after which the precipitate is removed and pressed, enough being left in the vat to precipitate the next lot of solution. The pressed cakes having been washed free from soda, air at 80 lbs. pressure is forced through them to drive out as much water as possible, and they are finally burned at 2000° F. A lower temperature is of course sufficient to dry them, but at this high heat the alumina becomes crystalline, in which state it is less prone to absorb moisture from the air. The weak liquor from the precipitating vat, having a sp. gr. of 1.2, is brought up to 1.45 sp. gr. in triple-effect evaporators and used to decompose calcined bauxite.

The Larne works are connected by railway with the mines at Glenravel, 35 miles distant. The main building is 240 ft. long and 80 ft. wide, constructed of brick. Steam is developed by two 25-ft. by 8 ft. Lancashire boilers, and the requisite power is furnished by a 14-in. by 20 inch non-condensing engine. The inventor of the process above described is Dr. K. H. Bayer.

BISMUTH.

By Albert Doerr.

Bismuth ores are found in many places in Saxony and the Saxon Erzgebirge, principally at Schneeberg, Altenberg, Annaberg, and Johanngeorgenstadt, and at Joachimsthal in Bohemia. Metallic bismuth is produced in considerable quantities at Schneeberg (blue color works), at Altenberg (tin works), and at Freiberg. Schneeberg is the largest producer, but the process there is a secret, and little is known concerning it except that it is entirely a dry one, the metallic bismuth being liquated out of its ores in special furnaces.

At Freiberg bismuth is produced as a by-product. The lead bullion from the blast furnaces contains many foreign constituents, among which are gold, silver, copper, arsenic, antimony, tin, zinc, iron, bismuth to the amount of 0.066%, cobalt, nickel, and sulphur. It is first liquated to free it from the greater portion of the copper, after which it goes to the refining furnace, where tin, arsenic, and antimony are removed in the order mentioned, and the bullion, which for the most part contains silver, gold, lead, and bismuth, is pattinsonized, by which means the larger portion of the gold and silver is concentrated into a rich base bullion, together with all the bismuth, while the rest of the lead forms a base bullion containing but little silver (0.10%) and no bismuth, and is treated by the Parkes process.

The rich gold-silver-bismuth-lead bullion is cupelled in large furnaces holding from 10 to 11 metric tons, until its percentage of silver has reached about 80, when most of the lead has been oxidized, and the remaining 20% is principally bismuth. The bullion is then ladled out of the large cupellation furnace and transferred to a smaller one having a capacity of about one metric ton, where the remaining lead and the bismuth are driven off, the greater portion of the mixed oxides being drawn off in the usual manner, and the rest absorbed by the cupel. The products are "blick-silber" or doré bullion, and a "bismuth litharge," consisting of the mixed oxides of bismuth and lead, principally bismuth oxide, which is crushed fine, together with part of the cupel, and transferred to earthenware pots (0.78 meter deep and 0.90 meter in diameter), in which it is treated with dilute hydrochloric acid. The contents of the crucibles are heated and frequently stirred, and in about 6 hours the Bi_2O_3 passes completely into solution, while the PbO is converted into $PbCl_2$, the greater portion of which remains insoluble. Water is then added until a very slight milky turbulence is apparent, whereupon

the solution is allowed to stand 7 hours, to permit that portion of the $PbCl_2$ precipitated by the addition of water to settle. The solution is then syphoned into wooden vats 1.88 meters deep and 1.57 meters wide, and sufficient water is added to precipitate entirely the bismuth as oxychloride, BiOCl, after the settling of which the solution is drawn off. The oxychloride contains a small quantity of lead chloride, to remove which it is twice redissolved with hydrochloric acid and precipitated by water. After the third precipitation the oxychloride is filtered, washed, dried, and fused in a cast-iron crucible with lime and charcoal, or with glass, soda, and charcoal. The crucible, which is 0.5 meter deep and 0.3 meter wide, is charged with from 15 to 20 kgms. of oxychloride, 16% lime, and 6% finely powdered charcoal. As soon as the charge is thoroughly fused the slag is dipped off and the metallic bismuth is cast into molds. It is fused again with the above-mentioned fluxes, to free it, as far as possible, from the lead which it still contains, and is then ready for the market, though 0.25 to 0.30% lead and 0.025% silver yet remain in it.

At Altenberg native bismuth and bismuthine (Bi_2S_3) occur with cassiterite. The ore is crushed, concentrated, and roasted in a reverberatory furnace to free it from sulphur and arsenic. The roasted ore while still hot is spread out in wooden troughs, about 10 ft. long, 2 ft. wide, and 2 ft. deep, and covered to a depth of from 8 to 10 in. with dilute hydrochloric acid (1 part acid to 1 part water). After standing for about 6 hours the solution is drawn off into other wooden troughs, partly filled with water, where it becomes so dilute that the bismuth is precipitated as oxychloride. At Altenberg the ores are almost entirely free from lead; consequently the precipitate contains no more than traces of this metal, but is contaminated by considerable iron, and in consequence is of a yellowish color instead of pure white. After the precipitate has settled, the water is drawn off, the oxychloride dissolved in hydrochloric acid, and reprecipitated by pouring the solution into a large volume of water. The precipitate still contains iron, however, and is dissolved a third time in hydrochloric acid and again precipitated. The oxychloride is then thoroughly washed, dried, and fused in a graphite crucible with lime and charcoal and varying quantities of slag resulting from previous fusions; the metal is then in a marketable condition.

Besides the Saxon works the chief producer of bismuth is the firm of Johnson, Matthey & Co., London, which reduces Australian and Bolivian ores, but no account of the process employed has been published. A description of the treatment of Bolivian ores has been written, however, as follows: The ores, containing 15 to 30% bismuth, and contaminated by sulphides of copper, iron, antimony, and lead, are roasted in reverberatory furnaces for 24 hours with repeated raking and admixture of charcoal. The roasted ore is mixed with charcoal (3%), lime, soda, and fluorspar, and smelted in a reverberatory furnace, yielding metallic bismuth (containing a little lead, antimony, and copper), copper matte (containing 5 to 8% bismuth), and a ferrous slag, which are separated in conical iron pots outside the furnace. The bismuth has to be refined; the matte is roasted and resmelted, giving up half its bismuth as metal, while the second matte (with 2 to 3% bismuth) is treated with HCl in the usual manner. It is supposed that the Johnson, Matthey & Co. process is of this nature.

BORAX AND BORACIC ACID.

By Walter Renton Ingalls.

The compounds of boron principally used in the arts are borax (biborate of sodium) and boracic (boric) acid. They are derived from various minerals, of which the most important commercially are borax (tinkal), sassolite, boracite (stassfurtite), colemanite (priceite, pandermite), and ulexite.

Sassolite (B_2O_3, 3 H_2O) is native boric acid, containing 56.4% boric anhydride; borax (Na_2O, 2 B_2O_3, 10 H_2O) is biborate of sodium, with 36.6% boric anhydride; ulexite is borate of sodium and calcium (Na_2O, 2 CaO, 5 B_2O_3, 16 H_2O) with 43% boric anhydride; colemanite is borate of calcium (2 CaO, 3 B_2O_3, 5 H_2O), with 50.9% boric anhydride; and boracite is a compound borate and chloride of magnesium (6 MgO, $MgCl_2$, 8 B_2O_3) with 62.5% boric anhydride. Any of these minerals may serve as source of either borax or boric acid, since borax is decomposed by mineral acids with the formation of boric acid, which is soluble in hot water and crystallizes out in cooling, while boric acid drives carbonic acid from carbonate of sodium with the formation of borax.

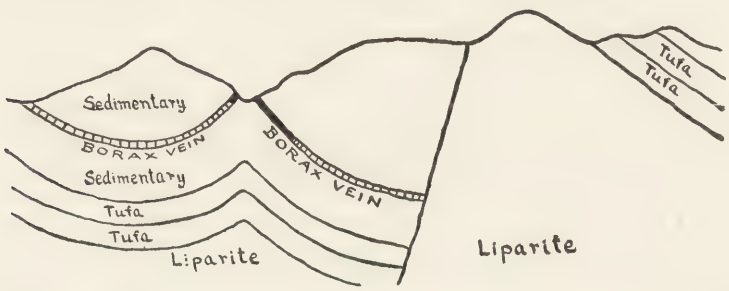
The borax and boric acid of commerce are obtained chiefly from California, Nevada, Italy, Germany, India, Chile, and Asia Minor, though the existence of the various boron minerals in many other parts of the world is well known, *vide* Vol. I. of The Mineral Industry. In Italy the total production is reported as boric acid, which is won almost entirely from the Tuscan lagoons of Monte Rotondo and Castelnuovo, in the water of which it occurs. The crude boric acid obtained there contains 74.84% B_2O_3, 2.4–1.2% $CaSO_4$, 14–8% $MgSo_4+Al_2(SO_4)_3$, 2.6–1.0% NH_4Cl, and 7–5.8% H_2O. Sassolite is found in the solid state at Sasso, whence it derives its name. For a further description of the industry in Italy see Vol. II., p. 75, and *British Consular Report,* No. 1250, 1893.

Boracite is produced only in Germany, where it occurs in the Stassfurth salt beds. The manufacture from this source is comparatively unimportant.

The production of borax in the United States comes wholly from Nevada and California, the minerals worked being borax, ulexite, and colemanite. Borax and ulexite are obtained from the marsh deposits, so-called; colemanite from a vein or bedded deposit 5 miles east of Calico, in San Bernardino County. At the latter point there are two parallel veins about 40 ft. apart, which have been opened at numerous points for a distance of nearly 5 miles east and west, and by one shaft to a depth of 500 ft. These veins are interbedded with sandstone and sandy clay, which are, of course, of sedimentary origin, the strata having been subsequently folded and the apex of the anticline scored away as shown in the accompanying sketch, which is drawn after one by W. H. Storms in the *Eleventh*

Annual Report of the State Mineralogist of California. The two veins, which were first discovered in 1882, are opened by inclined shafts, and so far show a width of 7 to 10 ft. The mineral is hauled in wagons to Daggett, 12 miles distant, and thence shipped by rail to the works of the Pacific Coast Borax Company (which owns the mines) at Alameda. The process of manufacturing borax employed there is a secret, though its general nature may be surmised from the well-known chemical reactions described further on.

Marsh deposits of borax are worked in California in Inyo County, where the Conn & Trudo Borax Works, situated in Salinas Valley, 18 miles north of Keeler, have a tract of several hundred acres covered with a crust of crystallized tincal 6 in. to 2 ft. thick, averaging 90% borax. The Salt Wells Borax Works, situated in Salt Wells Valley, 60 miles south of Keeler, partly in Kern and partly in San Bernardino County, have an area of about 100 acres, covered with a crust of sodium sulphate, 3 to 10 in. thick; adjacent tracts contain ulexite, assaying: Insoluble, 1.40%; water, 44.80%; sodium chloride, 16.28%; boracic acid, 36.83%. In Lake County borax was found in the bed of Borax Lake, but the deposit is now exhausted. The Nevada deposits have been fully described in previous volumes of this series.



Besides the Calico deposit colemanite occurs in the United States in Oregon, in Curry County, 5 miles north of Chetko, where it appears in a hard, compact form in layers between an overlying bed of slate (cavities and fissures in which it fills) and an underlying bed of blue steatite; it also occurs in masses, weighing as much as 200 lbs., imbedded in the steatite. This deposit has not been exploited.

In Asia Minor, however, an extensive bed of the mineral is worked. This lies beneath a thick stratum of gypsum, the colemanite (or pandermite, as it is called there) being found in more or less irregular masses, the aggregate supply of which is said to be large. The deposit is situated on the Chinar San, a small stream emptying into the Rhyndacus River, which flows into the Sea of Marmora near the port of Panderma. The crude mineral contains from 38 to 44% boracic acid, which by calcination, etc., is raised to 55% before shipment. The exportation amounts to from 14,000 to 15,000 metric tons per annum, some of it coming to the United States, but the more part going to England and Germany.

Borate of lime has also been imported into England from the west coast of Africa, but the character of the deposits has not been described and there are no statistics of the supply.

One of the most important of the world's sources of borax are the deposits of Asco-

tan and Maricunga, north of Copiapo in Atacama, Chile, where ulexite, known locally as "tiza," occurs in old lake beds in layers alternating with layers of salt and salty earth. A recent British consular report contains the following interesting particulars about the Maricunga borax district. The lake where the borax is found measures 32½ kilometers from north to south, and 20 from east to west, giving an area of 650 square kilometers. Uppermost is a layer of salt 20 cm. thick, mixed with clay and mud, and below this a layer 24 cms. thick, averaging 27% boric acid. Below the layer of borax is another of a mixture of borate of lime, soda, and clay, impregnated with iron. Below this again is another layer of borax 9 cms. in thickness. The quantity of borax in the explored part is calculated at 96,000,000 tons. A railway line passing through this district was recently surveyed, and its construction is now under consideration.

The total production of borax and boric salts and acid in the world in 1894 and 1895 was as follows, the figures being in metric tons:

Year.	United States. Borax.	Chile. Ulexite.	India. Borax.	Germany. Boracite.	Italy. Boric Acid.	Turkey. Pandermite.
1894.....	5,962	6,595	367	176	2,746	14,000–15,000
1895.....	6,126	4,355	400	150	2,633	

The production of borax in India is almost entirely from Bengal. The exportation of borax from Chile in 1891 was 815 metric tons, and of calcium borate 6463 tons. No statistics of borax proper are available for subsequent years. In 1887 the export of borax from Chile amounted to 3053 tons; from 1887 to 1891, however, it decreased, while that of "borate of lime" increased.

PRODUCTION AND IMPORTS OF BORAX IN THE UNITED STATES. (*a*)

Year.	Production.							Imports.					
	California.	Nevada.	Total.		Value.			Borax—Crude.		Borax. Refined.		Boracic Acid.	
	Pounds.	Pounds.	Pounds.	Met. Tons.	Total.	Per Pound.	Per M. Ton.	Pounds.	Value.	Lbs.	Value	Pounds.	Value.
1872..	280,000	*Nil.*	280,000	127	$89,600	32c.	$705	22,500	$80,000	35,542	$6,288	1,103,974	$191,575
1873..	1,030,000	970,000	2,000,000	907	496,000	24¾c.	547	*Nil.*	.....	9,284	2,152	1,222,006	255,186
1874..	1,829,771	2,170,138	3,999,909	1,814	567,987	14⅛c.	313	*Nil.*	.....	3,860	1,253	233,955	52,752
1875..	2,336,000	2,804,000	5,140,000	2,331	636,075	12⅜c.	273	588	78	5,153	1,224	41,742	6,280
1876..	2,873,909	2,307,001	5,180,910	2,350	563,424	10⅞c.	240	*Nil.*	.....	3,145	691	137,518	15,771
1877..	1,986,970	2,740,310	4,727,280	2,144	460,910	9¾c.	215	55	12	3,500	676	107,468	11,231
1878..	746,840	2,055,960	2,802,800	1,271	248,749	8⅞c.	198	286	61	3,492	574	22,839	651
1879..	727,146	827,840	1,554,986	705	139,949	9c.	198	*Nil.*	.....	3,472	490	36,000	916
1880..	1,219,948	2,640,800	3,860,748	1,751	472,942	12¼c.	270	22,122	742	15,278	2,011	243,723	18,473
1881..	1,380,205	2,665,200	4,045,405	1,835	556,243	13¾c.	303	*Nil.*	.....	4,136	866	187,055	15,771
1882..	1,465,732	2,770,559	4,236,291	1,922	582,490	13¾c.	303	*Nil.*	.....	15,710	3,774	536,335	71,343
1883..	1,800,000	3,800,000	5,600,000	2,540	826,000	14¾c.	325	*Nil.*	.....	5,611	1,359	4,334,432	580,171
1884..	2,037,940	5,675,363	7,713,303	3,498	752,047	9¾c.	215	142	34	7,332	1,692	44,512	4,494
1885..	1,885,300	5,586,104	7,771,404	3,389	616,391	8¼c.	182	*Nil.*	.....	240	41	48,517	4,035
1886..	2,569,630	6,524,542	9,094,172	4,125	608,857	6¾c.	149	4	1	4,625	770	430,655	26,238
1887..	2,029,380	5,571,735	7,601,115	3,493	437,064	5¾c.	127	33	4	3,731	439	376,184	19,885
1888..	2,809,088	4,004,355	6,813,443	3,091	476,941	7c.	154	455	38	4,705	600	487,777	26,394
1889..	1,939,650	5,333,482	7,273,132	3,299	545,485	7½c.	165	*Nil.*	.....	5,642	685	676,736	36,814
1890..	6,402,034	5,487,794	11,889,828	5,393	891,737	7½c.	165	29,608	800	2,302	275	867,802	43,967
1891..	8,533,337	3,296,663	11,830,000	5,366	887,250	7½c.	165	414,151	17,681	10,725	1,062	666,765	41,019
1892..	11,050,495	1,487,701	12,538,196	5,687	940,365	7½c.	165	40	6	3,970	426	701,625	39,418
1893..	7,999,562	1,199,438	9,199,000	4,173	689,925	7½c.	165	543,967	13,659	11,230	1,327	771,775	40,568
1894..	11,540,099	1,600,595	13,140,594	5,950	919,842	7c.	155	441,066	11,427	1,812	225	292,990	19,282
1895..	11,919,141	1,587,215	13,506,356	6,126	742,850	5½c.	121	4,234,261	105,604	612,730	26,429	925,158	42,056
1896..	13,609,614	1,648,400	15,258,014	6,649	762,900	5c.	110	4,307,100	104,951	11,376	796	555,769	21,899

(*a*) The figures of production are for calendar years; those of imports are for fiscal years ending June 30. Included with the production of borax in 1896 are 621,000 lbs. of boracic acid, worth 7½c. per lb. The manufacture of boracic acid in the United States is a comparatively new industry.

The borax industry in the United States began in 1864. It did not amount to much, however, until 1873, when the Nevada marshes and those of San Bernardino County, Cal., were first worked. In that year the price of borax in New York was 32c. per lb. A decline then set in and continued until 1878 when 8¾c. was reached. This was due in part to economies in production, and in part to the keen competition between the various producing companies. Finally a combination was formed among them whereby production was curtailed and the price of borax was arbitrarily increased 25%, but this led to a lively competition between the associated companies of California and a concern which was making borax from imported boracic acid and imports of the latter material increased heavily. In 1880, however, prices in London increased from £35 to £60, which was accompanied of course by a corresponding increase in New York, and the high prices being maintained until 1883 this was a prosperous period for the American producers, and their production increased largely. In 1883, also, the great deposit of colemanite at Calico, Cal., was discovered, furnishing a cheaper supply than the marshes, which had been up to that time the only source of the mineral.

According to the tariff of 1883 the duty on refined borax and boracic acid was 5c. per lb., on commercial boracic acid 4c. per lb., and on crude borax and borate of lime 3c. per lb., the crude material having previously been on the free list. With the increasing supply of borates prices began to decline again. In 1884 the price of borax in London fell from £60 to £40, and in New York 9¾c. was quoted. In 1885 the price had dropped to 8¼c. per lb., and in 1887 it fell to 5¾c. An attempt to check the decline was made by the organization of the Borax Board in California, but this did not accomplish much. In 1888, however, a stronger combination was formed and through it prices were raised a little. This agreement lasted until 1890, but on its expiration the Pacific Coast Borax Company was organized and absorbed nearly all the producers. The position of this company and the increase of the duty on crude boracic acid to 5c. by the tariff law of 1890 made it possible to maintain the price of borax at 7½c., production increased enormously, and large profits were realized in the business. In 1891 the Pacific Coast Borax Company declared a dividend of $180,000; in 1892 one of the same amount; and in 1893 one of $60,000, the falling off in the last year being due to the general disturbance in business and the restriction of production in consequence thereof.

The tariff of 1894 reduced the duty on refined borax and boracic acid from 5c. to 3c. per lb., and on borate of lime from 3c. to 1½c. The price in the United States did not at once decline correspondingly, however, and large quantities of English refined borax were imported. As a result of this new competition the price fell gradually to 6⅜c. Imported borax at that time could be laid down in New York at a cost of 6½c.* The cost of California borax in New York at the same time, calculating the difference between gross value of product in 1891–93 and dividends paid as cost of production, was about 6c. per lb., which was, however, probably higher than the actual cost.

The sharp competition finally brought the market down to 5¼c. per lb. for carload lots in December, 1894. In February, 1895, there was an advance to

*According to the *Oil, Paint and Drug Reporter*, March 8, 1897, from which much of these data of the fluctuations in the borax market is taken.

5½c., and in March to 6c.; but in the latter part of April 5¾c. was touched; in June 5½c., and in August 5¼c., which was the lowest price ever known in the New York market up to that time. In November, however, the price advanced to 6c.

Under the new tariff the New York refiners had a distinct advantage, and imported large quantities of borate of lime. On the other hand the California producers shipped large lots of crude mineral to Liverpool.

In April, 1896, a new company was organized, taking in the Pacific Coast Borax Company and Redwood's Chemical Works, Limited, of England, with a capitalization of $2,550,000 and $500,000 first mortgage bonds. The plan was to ship the California colemanite to England and refine it there. A large dividend was earned by the new company during the first six months, notwithstanding the low price prevailing. On November 9 this was reduced to 5c. in carloads, which is the lowest point borax has ever touched.

The Italian boracic acid is partly exported as such and is partly converted into borax for export, the proportion varying according to market conditions. Thus in 1894 boracic acid was quoted in England at so high a price that it could not be used profitably for the manufacture of borax there, and more was converted into borax at home; whereas in 1895, the price of the acid having fallen to £15 per ton in Liverpool, the manufacture of borax was resumed in England to the exclusion of the Italian salt. The export of boracic acid from Leghorn to England in 1894 was 1274 tons; in 1895, 1985 tons; while in 1896 trade was still more active. Italian producers are able apparently to meet the competition of Turkish, Chilian, and Californian borates.

It is apparent that the world's supply of borax is abundant, especially in the Calico district of California and in Chile, where there are immense deposits lying unworked which would be exploited if a rise in the value of the mineral should hold out sufficient inducement.

The method of treating calcium borate at the works of the Pacific Coast Borax Company at Alameda, Cal., is unknown, but mineral from the Stevens & Greer mines, on the south side of the Calico Range, has been worked by pulverization and carefully adding the powder to a boiling solution of sodium sulphate until effervescence ceases.* The precipitate having been allowed to settle, the solution is drawn off and the borax recovered by crystallization.

Boric acid was produced at the same place by treating the calcium borate with sulphuric acid, the boric acid formed thereby being dissolved in water, and separated by crystallization.†

In general, however, it is found that treatment with sulphuric acid does not give very satisfactory results, and chlorhydric acid is used instead. About two parts acid and four parts water to three parts mineral, all by weight, are required. The mixture is heated to boiling and kept at that temperature until digestion is complete, water lost by evaporation being replaced. The clear liquor is finally decanted, and on cooling the boric acid crystallizes out, the mother liquor retaining sodium or calcium chloride, together with a slight excess of hydrochloric acid. The boric acid crystals, according to Lunge, are drained, pressed, or whizzed, washed with cold water and whizzed again, whereby they are

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* *Twelfth Annual Report California State Mineralogist.* † *Ibid.*

obtained so highly refined that if used for the preparation of borax the latter is pure at the first crystallization. Borax is made from boric acid by boiling with a solution of sodium carbonate, or by heating in a muffle crude boric acid with half its weight of calcined soda, the melted mass being lixiviated in an iron pan and the clear liquor drawn off for crystallization. In France borax was originally made by fusing boric acid and calcined sodium sulphate in a similar manner. A new method of preparing borax by heating boracic acid and salt in an atmosphere of superheated steam was described in THE MINERAL INDUSTRY, Vol. II., but I am unaware that it has yet come into use.

The direct determination of boric acid is somewhat tedious, even by Gooch's method, which is the best. For commercial purposes the following indirect method is sufficient: Dissolve 3 gms. of sample in water; tint with methyl orange, and add from a burette normal solution of sulphuric acid until the color changes to pink. The result is calculated from the equation—100 c.c. normal solution of $H_2SO_4 = 7$ gms. $B_2O_3 = 10.1$ gms. anhydrous borax ($Na_2B_4O_7$). Determine water in borax by loss of weight on fusion; sulphur by acidifying an aqueous solution with HCl and precipitating with $BaCl_2$; chlorine by acidifying an aqueous solution with HNO_3 and precipitating with $AgNO_3$; alumina by direct precipitation with NH_4OH.

The value of commercial borax is determined by G. Heid in the following manner:* (1) Weigh up 5 gms. and determine chlorine by precipitation with silver nitrate. (2) Weigh up 1 gm., dissolve in water, add HCl and evaporate to dryness, obtaining a residue of sodium chlorine and boric acid. Take up in water and determine chlorine with silver nitrate. The difference between the results 1 and 2 represents chlorine equivalent to sodium existing as borax in the original sample, whence the percentage of borax may be calculated: $Cl_2 : Na_2B_4O_7 :: a : x$. This method is, of course, equally applicable to the determination of borax in crude tincal, and by modifying the formula will serve also for estimating borate of lime in crude colemanite or pandermite.

Borax is used chiefly as a flux in certain smelting processes, in welding steel, and in soldering metals; it is also used extensively in glazing earthernware, while its minor uses in the arts are numerous. The production of boric acid is for the more part consumed in the manufacture of borax. It is, however, an exceedingly important medicinal remedy, and it has been claimed that a mixture of boric acid and potassium nitrate is a better flux than borax.

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* *Zeits. für Angewandte Chemie*, 1896, p. 679.

BROMINE.

THE bromine industry experienced no material change in 1896, either in Germany or the United States, though the production in the latter increased considerably. In both countries it is won as a by-product, and consequently attracts no especial interest. The American product in 1896 was 550,285 lbs.; the German product has for several years been about 660,000 lbs. per annum; consequently the total amount of money involved in the industry is not large. Bromine and its derivatives are very important in the arts, but a small quantity of them meets the requirements, and no new uses are developing.

At one time several years ago there was a sharp war between the American and the German producers, but this was settled in October, 1892, by an agreement that each party should have its own field, *i.e.*, that there should be no imports and no exports. This agreement has been maintained, and the market has consequently been quiet.

A similar understanding exists among the American producers to limit their production to the demand of the market, and thus maintain the price. In 1896 this averaged 26c., which was also the price in 1895.

PRODUCTION OF BROMINE IN THE UNITED STATES.

Year.	Michigan.	Ohio.	Pennsylvania	West Virginia.	Total.	Metric Tons.	Value.	
							Total.	Per Pound.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.			
1892.....	43,864	135,683	92,978	104,142	376,667	171	$64,034	17c.
1893.....	42,565	113,575	111,403	80,852	348,399	158	87,100	25
1894.....	29,116	146,501	100,623	103,304	379,544	172	98,655	26
1895.....	30,280	152,360	104,647	107,567	394,854	179	102,662	26
1896.....	42,000	212,850	154,600	149,835	550,285	249	143,074	26

The expectation that there would be an increased demand for bromine in the extraction of gold from its ores has not yet been realized. The Sulman bromocyanogen process is now being exploited, and a small amount may be used in this; but there is no prospect that bromination will replace chlorination, wherein there would be more demand for the chemical.

The sources of bromine, methods of preparation, and general conditions affecting the industry are fully described in previous volumes of THE MINERAL INDUSTRY, especially in Vol. II.

CEMENT.

THE trade in Portland cement was good in the United States in 1896, although prices were low owing to the keen competition. For this reason also it is the opinion in the industry that the range will continue low, and preparations to meet it are being made by the more enterprising companies by improvements in methods of manufacture and enlargement of plants. Notwithstanding the large increase in production in 1896 there is prospect of a greater increase in 1897, the opinion having been expressed, indeed, that the output will amount to 1,500,000 bbls.

The prosperity in this branch of the cement industry is evidence that the American Portland cement is competing successfully with the foreign, of which the imports are decreasing, and also that it is displacing the natural or Rosendale cement. The latter is a development which is likely to gain force, since an artificial cement made under proper technical supervision is certain to be more uniform than a natural cement, however good, and although the Portland is more expensive its extra cost is fully compensated for many uses by its greater reliability.

There are 24 Portland cement works in the United States, of which 18 are in New York and New Jersey. Some of these are as well equipped as the best foreign works. Three kinds of kilns are used, viz., the intermittent or dome kiln, the continuous (Dietzsch or Shöfer), and the rotary, the last of which introduces a great economy in labor and is coming more and more into use, nearly 50% of the American cement now being burned in this manner.

Natural cement is produced in 15 States, but about one-half the entire product comes from New York, and of that nearly all is from the Rosendale district, where there were 15 works in 1896, operated by 12 companies. After New York, the Louisville district, partly in Kentucky and partly in Indiana, is the most important.

The industry in the Rosendale district of New York suffered severely in 1896, owing to the decrease in demand and continued large production, which naturally forced the price down. The average for the year was only 70c. per bbl. at Rondout against 75c. in 1895, and the complaint was general in the trade that none of the producers had made any money.

CEMENT PRODUCTION, IMPORTS, EXPORTS, AND CONSUMPTION IN THE UNITED STATES.

(In barrels of 300 lbs.)

Year.	Production.				Imports.		Exports.		Consumption.	
	Natural Hydraulic.	Portland.	Total Barrels.	Value.	Barrels.	Value.	Barrels	Value.	Barrels.	Value.
1892.....	8,211,181	547,440	8,758,621	$7,152,750	3,254,273	$3,378,331	107,894	$169,538	11,905,000	$10,361,543
1893.....	7,445,950	673,989	8,119,939	6,063,131	3,565,932	3,470,169	112,520	174,663	11,573,451	9,358,737
1894.....	7,895,259	738,196	8,633,455	5,478,051	3,517,476	3,396,729	12,966	15,072	12,137,955	8,859,708
1895.....	7,694,053	998,745	8,692,798	6,027,374	3,996,527	3,873,123	11,927	13,895	12,676,798	9,931,572
1896.....	7,454,611	1,239,185	8,693,796	6,063,528	3,556,902	3,393,199	54,639	86,757	12,195,959	9,369,700

PRODUCTION OF PORTLAND CEMENT IN THE UNITED STATES.

(In barrels of 400 lbs.)

States.	1895.			1896.		
	Barrels.	Value at Works.		Barrels	Value at Works.	
		Total.	Per Bbl.		Total.	Per Bbl.
California.....	12,212	$32,566	$2.67	8,985	$17,970	$2.00
New Jersey.....	135,000	236,250	1.75	227,100	397,425	1.76
New York.....	135,368	308,454	2.28	197,872	304,708	1.54
Ohio	117,706	225,235	1.91	163,182	320,364	1.96
Pennsylvania.....	272,048	457,584	1.68	396,739	583,632	1.47
South Dakota.....	61,725	130,000	2.11	23,776	47,552	2.00
Texas.....	15,000	40,000	2.67	15,000	38,500	2.56
Total barrels.....	749,059	$1,430,089	$1.91	1,032,654	$1,710,151	$1.65
Total metric tons.....	135,879	Per met. ton..	10.53	187,365	Per met. ton..	9.13

PRODUCTION OF NATURAL HYDRAULIC CEMENT IN THE UNITED STATES.

(In barrels of 300 lbs.)

States.	1895.			1896.		
	Barrels.	Value at Works.		Barrels.	Value at Works.	
		Total.	Per Bbl.		Total.	Per Bbl.
Illinois.....	216,010	$75,604	$0.35	246,755	$86,364	$0.34
Indiana and Kentucky.....	1,701,023	597,091	.35	1,434,337	504,049	.35
Kansas.....	188,000	121,340	.65	171,017	109,747	.64
Maryland.....	144,450	74,993	.52	188,050	95,846	.51
Minnesota.....	90,687	37,291	.41	53,098	26,591	.50
New York, Ulster County.....	3,230,051	2,422,538	.75	3,577,594	2,518,393	.70
New York, Onondaga County.....	259,843	141,120	.54	94,816	47,881	.50
New York, Schoharie County.....	31,980	23,804	.74	28,465	20,898	.73
New York, Erie County.....	970,427	618,577	.64	955,970	553,027	.57
Ohio	39,401	36,024	.91	32,319	24,735	.76
Pennsylvania.....	400,998	233,713	.58	312,041	185,397	.59
Virginia.....	1,600	1,440	.90	3,749	2,249	.60
Wisconsin.....	419,583	213,750	.51	356,400	178,200	.50
Total barrels.....	7,694,053	$4,597,285	$0.58	7,454,611	$4,353,377	$0.57
Total metric tons.....	1,047,006	Per met. ton..	4.39	1,014,424	Per met. ton..	4.29

IMPORTS OF CEMENT INTO THE UNITED STATES ACCORDING TO SOURCE.

Year.	United Kingdom.		Belgium.		France.		Germany.	
	Short Tons. (a)	Value.	Short Tons. (a)	Value.	Short Tons. (a)	Value.	Short Tons. (a)	Value.
1894.....	200,780	$1,255,575	107,426	$654 642	3,175	$20,084	209,361	$1,391,643
1895.....	161,376	1,014,316	141,775	865,687	4,567	27,842	259,984	1,737,442
1896.....	127,215	815,053	137,446	800,285	9,283	51,918	245,656	1,623,322

IMPORTS OF CEMENT INTO THE UNITED STATES ACCORDING TO SOURCE.—*Continued.*

Year.	British North America.		Other Countries.		Total.		Exports.	
	Short Tons. (a)	Value.	Short Tons. (a)	Value.	Short Tons. (a)	Value.	Short Tons. (a)	Value.
1892	.....	.....	.....	.....	488,131	$3,378,331	15,836	$137,167
1893	.....	.....	.....	.....	534,830	3,470,169	18,697	154,461
1894	1,239	$11,938	9,142	$62,847	531,123	3,396,729	23,545	180,881
1895	2,083	18,894	3,394	26,260	599,479	3,873,123	18,516	131,541
1896	1,879	15,989	2,490	21,641	533,535	3,393,199	13,146	103,315

(a) Estimated from number of barrels reported; at 1 bbl. = 400 lbs.

The German Cement Trade in 1896.

Nearly all the German cement works have joined in a syndicate the object of which is to prevent undue competition and maintain prices. In 1896 business was active and the production was greater than in the preceding year, and in spite of the competition and the low prices fair dividends seem to have been the result of the year's work. The total quantity exported by sea amounted to 199,677 tons, of the value of $1,845,600, being a very large increase in volume, but a decrease in value compared with 1895. Of the total quantity exported the United States took half, and the next largest customers were Brazil and Australia, while considerable quantities went to the different parts of Africa. The largest European customer was Norway.

There are 63 cement works in Germany, most of them in the Rhine district, but there are three or four in the neighborhood of Hamburg.

The Manufacture of Slag Cement.

In a description of the slag cement works at Vitry-le-Francois, Lorraine, in *Nouvelles Annales de la Construction*, it is noted that the manufacture of slag cement comprises two distinct operations, viz., the production of slag sand and its mixture with lime. The slag obtained from the Pont-a-Musson blast furnaces, containing alumina, 22%; silica, 32%; lime, 42%, and iron oxides, magnesia, etc., 4%, is well suited to this manufacture, being hot, calcic, and very regular. On issuing from the tap-hole it is granulated to a fine sand by water under pressure. The mixture of the sand with lime is effected in special pug-mills, consisting of steel-plate cylinders revolving around their horizontal axis, and each containing 1.6 ton of cast iron balls, turning out about a ton of cement per hour. Slag cement, according to the *Genie Civil*, appears capable of holding its own for submarine works. Mixed with lime, it appreciably improves the quality and hastens the setting, while it is also employed for mortar, concrete, pavement, and coatings exposed to the air, but care must be taken in the latter case to keep the work damp for several days. Tests show that the resistance to tensile strain of Vitry slag cement may attain 49 kgms. per square centimeter (697 lbs. per sq. in.) after a year's setting.

Cement Manufacture in Great Britain.*

By W. Harry Stanger and Bertram Blount.

The following article, which is divided into six main sections, deals chiefly, as far as manufacturing is concerned, with English practice. The general statements concerning the composition and nature of cements of course hold universally.

Section I.—Classes of Cements and Raw Materials.

Finished cement may be divided broadly into two classes: (1) those of the Portland type, in which the act of setting is one involving the resolution of existing compounds, and (2) those of the puzzuolanic class, the setting of which connotes the formation of new bodies by the combination of substances previously existing separately. Portland cement is commonly spoken of as "natural" and "artificial." The phrase is a bad one because "natural" cement does not exist; what is meant by it is simply that the raw materials for making Portland cement, viz., chalk and clay, may exist ready mixed in about the right proportion to make Portland cement when the cement rock is burned. Save from a manufacturing point of view there is no distinction between "natural" and "artificial" Portland cement except in regard to the fact that the natural cement is usually badly mixed and consequently of inferior quality. There is nothing to prevent the production of a high-class "artificial" Portland cement from "natural" rock if the latter be thoroughly mixed. The ground may be cleared at once by saying that "natural" cements, as far as they are reckoned to belong to the type "Portland," will not be considered separately because they may be regarded, for all purposes, as badly made "artificial" Portland cements.

(1) *True Portland Cements.*—The nature of the raw materials from which Portland cements are made is a matter of indifference if the product have the correct ultimate and proximate chemical composition. Its essential constituents are lime, silica, alumina and ferric oxide, and these may be brought together in the form of chalk, limestone, marl, gault, clay, river-mud and shale. In England, in the London district, chalk and clay are almost exclusively used. The composition of Medway mud, which is employed in most of the cement-works both on the Thames and Medway, is that of an impure kaolin mixed with finely divided sand, as may be gathered from the following analysis:

	Per Cent.		Per Cent.
Insoluble residue	(*a*)40.26	Magnesia (MgO)	1.90
Silica (SiO_2)	21.92	Sodium chloride (NaCl)	1.74
Alumina (Al_2O_3)	14.70	Combined water, organic matter, and loss	9.13
Ferric oxide (Fe_2O_3)	8.92		
Calcium carbonate ($CaCO_3$)	1.43		100.00

(*a*) Consisting of quartz (SiO_2), 28.42; and feldspar, 11.84, viz.: silica (SiO_2), 8.33; alumina (Al_2O_3), 2.36; lime (CaO), *nil*; soda (Na_2O), 1.15; total, 40.26.

* In this article by tons is meant long tons of 2240 lbs.

The chalk is nearly pure calcium carbonate, and the adjustment of the proportion of the two materials is tolerably simple. In the Midland districts, and in the North of England, limestones and marls are used to some extent, each deposit needing special knowledge of its average composition and of its liability to fluctuate in character for successful working. From whatever raw materials the mixture for making Portland cement may be prepared, it is necessary to commingle them most intimately (the methods of doing this will be described in Section II), and to heat the resulting slurry to a clinkering temperature. It is the subjection of the slurry to this temperature which transforms the material into cement; consequently if the temperature be not attained, or if any material be added to the vitrified mass after burning, the product resulting is not Portland cement. In practice these conditions are very nearly fulfilled; the risk of failure to fulfill them, and the consequences of this failure, will be dealt with later.

The ultimate chemical composition of good Portland cement may be gathered from the following analyses, which comprise: (1) two samples of representative English manufacture; (2) a sample of well-known German make; (3) an analysis which does not represent any particular cement, but may be considered typical of modern cement made from non-magnesian raw materials.

	1.		2.	3.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Silica (SiO_2)	21.44	21.66	20.10	22.0
Insoluble residue (sand and clay)	1.08	1.28	0.62	1.0
Alumina (Al_2O_3), Ferric oxide (Fe_2O_3)	13.22	11.56	7.62 3.50	7.5 3.5
Lime (CaO)	(*b*)61.22	(*d*)61.18	(*f*)63.60	62.0
Magnesia (MgO)	1.04	1.09	1.30	1.0
Sulphuric anhydride (SO_3)	(*c*)0.96	(*e*)1.63	(*g*)1.98	1.5
Carbonic anhydride (CO_2), Water (OH_2)	(*a*)0.54	0.76	0.30 0.48	0.5 0.5
Potash (K_2O), Soda (Na_2O) and loss	0.50	0.84	0.50	0.5
Total	100.00	100.00	100.00	100.0
Specific gravity	3.135	3.185	3.165	3.175

(*a*) Volatile at a red heat. (*b*) Value after deducting that in calcium sulphate, 60.55%. (*c*) Calculated as calcium sulphate, 1.63. (*d*) Value after deducting lime in calcium sulphate, 60.04%. (*e*) Calculated as calcium sulphate, 2.77%. (*f*) Value after deducting lime in calcium sulphate, 62.21%. (*g*) Calculated as calcium sulphate, 3.37%.

Judgment of the quality of a cement from the data afforded by its analysis will be dealt with in Section IV. At present it is sufficient to say that it is possible to have a cement, the composition of which falls within the limits given above, which is nevertheless of bad quality; discrimination in a case like this is only possible to the expert. But the *proximate* composition of a cement is not expressed by its analysis. Of the many chemists who have worked on this subject, H. Le Chatelier has published conclusions which are based upon the best data and show the greatest insight into the problems to be solved. Pending revision by himself or some equally competent observer, his views will be adopted throughout this article. Le Chatelier has shown that the chief constituent of good hard coherent clinker is tricalcium silicate ($3\,CaO, SiO_2$). In ordinary cement it is mixed with dicalcium silicate ($2\,CaO, SiO_2$), which has little or no cementitious value. The

alumina and ferric oxide of the clay constituents of the cement appear in the clinker as the compounds tricalcium aluminate (3 CaO, Al_2O_3) and tricalcium alumino-ferrite (3 CaO, Al_2O_3, Fe_2O_3). A small quantity of monocalcium silicate (CaO, SiO_2) (wollastonite) is also found, together with varying amounts of magnesia and alkalies. The object of the manufacturer is to produce the maximum quantity of tricalcium silicate (3 CaO, SiO_2) and a reasonable amount of tricalcium aluminate (3 CaO, Al_2O_3). The latter is the body which first hydrates on mixing the cement with water and causes the cement to set, and the former is the substance which hydrates more slowly and causes the ultimate hardening of the cement; to the hydration and hardening of this tricalcium silicate the strength of the cement is chiefly due. Commercial Portland cement generally contains a certain amount of silicates poorer in lime than tricalcium silicate which are not wholly worthless. They serve to combine with any unsaturated lime which may be present in the irregularly burnt clinker, acting in this respect like puzzuolana (see below). The better the manufacture and product the smaller the percentage of this alien material.

(2) *Roman Cement.*—Roman cement is now made in England in comparatively small quantity. It is still used in certain work where quick setting is a necessity, and, though not so strong as Portland cement, is a good and reliable material. It is cognate in some respects with "natural" cement inasmuch as it is prepared from a native mixture of calcareous and clayey materials, but the English variety is usually distinguished from "natural" cements by its higher content of ferric oxide (Fe_2O_3), its lower percentage of lime, and the presence in it of manganese. The last is not of cementitious value, but serves to differentiate the product. Roman cement is prepared by burning septaria nodules which are dredged up on the coasts of Essex and Kent and are burnt at a temperature below the clinkering point. The following is an analysis of Roman cement:

	Per Cent.		Per Cent.
Silica (SiO_2)	19.62	Carbonic anhydride (CO_2)	3.43
Insoluble residue (sand and clay)	5.86	Water (OH_2)	0.25
Alumina (Al_2O_3)	10.30	Potash (K_2O), Soda (Na_2O) and loss	1.46
Ferric oxide (Fe_2O_3)	7.44		
Manganese dioxide (MnO_2)	1.57	Total	100.00
Lime (CaO)	(*a*)44.54		
Magnesia (MgO)	2.92	Specific gravity	3.11
Sulphuric anhydride (SO_3)	(*b*)2.61		

(*a*) Value after deduction of that of CaO as sulphate, 42.71%. (*b*) Calculated as calcium sulphate, 4.44%.

The proximate contents of Roman cement have not been closely investigated, but it is probable that they consist, like those of Portland cement, of tricalcium silicate and tricalcium aluminate, with the latter in much larger proportion than that which occurs in Portland cement. The production of these cementitious bodies at a temperature below the clinkering point of Portland cement slurry is to be attributed to the greater ease with which the union of the acid and basic constituents takes place, owing to the larger proportion of substances (*e.g.*, ferric oxide) capable of acting as fluxes.

(3) *Hydraulic Lime.*—Hydraulic lime is best grouped as included in the Portland-cement class, to which it belongs chemically, although for structural purposes it behaves very differently. This classification is due to the fact that hydraulic lime undoubtedly contains tricalcium silicate as its essential cementitious constituent. This substance is produced by burning a limestone containing sufficient silica to unite with the greater portion of the lime. The union of the silica and the lime may be induced by one or both of two circumstances, viz., the fine state of division of the silica, or the presence of Fe_2O_3 and Al_2O_3 as fluxing constituents. In the famous Chaux du Theil, the original limestone contains little besides calcium carbonate and silica; the union of these two is rendered possible by the extremely fine states of division of the silica. The English hydraulic limes, on the other hand, contain an appreciable amount of ferric oxide and alumina, and approach Portland cement more nearly in composition. The following are analyses of these hydraulic limes:

Constituents.	Hydraulic Lime.		Constituents.	Hydraulic Lime.	
	Theil.	English.		Theil.	English.
	Per Cent.	Per Cent.		Per Cent.	Per Cent.
Silica (combined) (SiO_2)	21.7	14.17	Sulphuric anhydride (SO_3)	0.3	1.63
Insoluble residue (sand)	0.3	2.39	Carbonic anhydride (CO_2), Water (H_2O)	0.6	3.64 / 2.69
Alumina (Al_2O_3)	1.8	6.79	Alkalies and loss	...	1.38
Ferric oxide (Fe_2O_3)	0.6	2.34			
Lime (CaO)	74.0	63.43			
Magnesia (MgO)	0.7	1.54		100.00	100.00

They may in fact be regarded as crude Portland cement containing an excess of lime which has to be slaked before they can be safely used.

(4) *Cements of the Puzzuolanic Type.*—A typical puzzuolana consists of some form of siliceous material containing silica in a sufficiently active state to be capable of combining with basic substances, *e.g.*, lime. Such active silica is commonly, but not necessarily, hydrated. The simplest conceivable puzzuolana would be made from a mixture of precipitated silica, dried but not dehydrated, and calcium hydroxide. When these materials are gauged together with water they unite slowly and form a stable silicate of lime, capable of resisting the action of water, and thus constituting an hydraulic cement. In practice, pure silica is not used for making puzzuolanic cements, because the few forms of it which are found hydrated or in sufficiently active condition to be capable of setting with lime, *e.g.*, kieselguhr, are valuable for other purposes. It follows from this that the siliceous substances actually employed are not pure silica, although they contain a considerable quantity of silica available for the purpose of setting with lime. Seeing that in all cases the lime is the essential basic constituent of a puzzuolanic cement, the acid constituents may be used as a means for the classification of these cements. Of these, the best known is (1) puzzuolana itself. It is a volcanic tufa found largely in the neighborhood of Naples and Rome, and has the composition shown by the following analyses:

	Neapolitan.	Roman.
	Per Cent.	Per Cent.
Soluble silica	27.80	32.64
Insoluble silicious residue	35.38	25.94
Alumina and ferric oxide ($Al_2O_3+Fe_2O_3$)	19.80	22.74
Lime (CaO)	5.68	4.06
Magnesia (MgO)	0.35	1.37
Sulphuric anhydride (SO_3)	trace	trace
Water and carbonic anhydride (H_2O+CO_2)	4 27	8.92
Alkalies and loss	6.72	4.33
	100.00	100.00

Similar materials are (2) trass, which is also a volcanic tufa, and is found in the Rhine district and in certain West Indian islands. (3) Santorin earth, from the Greek island of Santorin, is a similar material. In short, many products found in the neighborbood of volcanic districts, and consisting essentially of silicates mixed with silica more or less hydrated, may be and are used for the production of puzzuolanic cements. (4) An artificial puzzuolana may be prepared from burnt clay. The crude kaolin of which the clay consists is dehydrated by burning, and a portion at least of the siliceous matter becomes capable of acting on lime. Whether a resolution of the kaolin takes place under these conditions or whether it acts *per se* is not accurately known. (5) A second artificial puzzuolanic cement is made from blast-furnace slag and lime. The slag may be regarded as an artificially produced igneous rock, and to be most effective as a puzzuolana it must fulfill several conditions. It must in the first place be basic, containing if possible as much as 50% of lime ;* and in the second place it must have been cooled by quenching, when molten, in water, which is done by causing the fused slag to flow into a trough of running water. The granulated slag prepared in this way is partly vesicular in structure, and is chemically different from the same slag allowed to cool spontaneously Thermochemical determinations have shown that it possesses a greater chemical energy than does the unquenched slag; in fact it differs from slowly cooled slag much as plastic sulphur prepared by quenching fused sulphur differs from ordinary yellow crystalline sulphur. Unlike plastic sulphur, however, it does not appear to tend to revert to its more stable condition, and thus when once prepared can be kept for cement-making indefinitely. The granulated slag contains as much as 50% of water as it comes from the trough; it has to be at least partly dried before being mixed with lime, and if it needs to be carried to any considerable distance it should be completely dried, to save freight. The composition of granulated blast-furnace slag made for cement-making in the North of England, in the Middlesbro' district, is shown by the following analysis:

	Per Cent.		Per Cent.
Insoluble residue	1.04	Sulphuric anhydride (SO_3)	0.45
Silica (SiO_2)	31.50	Sulphur (S)	2.21
Alumina (Al_2O_3)	18.56		100.30
Manganous oxide (MnO)	0.44	Deduct oxygen equivalent to sulphur	1.10
Lime (CaO)	42.22		99.20
Magnesia (MgO)	3.18		
Soda (Na_2O)	0.70		

* Basic blast-furnace slag must not be confused with basic slag from a steel-furnace, which is of a totally different composition and consists essentially of crude calcium phosphate.

In all these cases the actual cement is best made by grinding together the siliceous material and slaked lime as intimately as possible. In practice the mixture is often done roughly; and where the puzzuolana is abundant and on the spot there is no objection to this, since it involves only the use of more puzzuolana than would otherwise be needed, and the surplus serves as an aggregate for the mortar or concrete which is being prepared. When, however, heavy carriage has to be paid on the puzzuolana, intimacy of admixture is economical. It is characteristic of cements of this class that they set very slowly and are usually weaker than cements of the Portland class. Their strength is, however, considerable, and their slow setting is sometimes an advantage, as it allows a structure made with them to settle slightly after building without causing cracks in the cement. Not much puzzuolanic cement is made or used in England. A small quantity of granulated blast-furnace slag is prepared, but the bulk of it goes, not for preparing slag-cement, as the mixture of granulated slag and lime is termed, but for adulterating Portland cement. This practice, which will be discussed more fully in Section IV., is in vogue chiefly in the North of England. True slag-cement is, however, prepared in Germany on a considerable scale; the mixture of slag and slaked lime is in the proportion of about 3 : 1, and the product is ground extremely fine, ball mills being generally used.

SECTION II.—PRINCIPLES OF MANUFACTURE.

The preparation of cements of all classes should be an exact chemical process. In practice it is too often a totally empirical proceeding. Improvement in quality can only be effected by due regard being paid to the chemical principles which underlie the manufacture. In the manufacture of Portland cement the following are the chief points which should be studied by the maker, and are being studied now in this country by the most enterprising manufacturers: The raw material must be sufficiently homogeneous to admit of fixed proportions being maintained for hours and days together. With a material of fluctuating quality a good deal of bad cement may be made before the alteration of quality has been detected and the proportions of the "wash" corrected. Even when the daily fluctuations of quality are small, constant vigilance and checking of the composition of the slurry by the calcimeter (see below) are necessary to produce a sound product. The raw materials must be finely ground and intimately mixed. In England as a rule the mixing is done by the semi-wet process in a "wash"-mill, and in the best works the slurry is further ground between wet stones, whence it passes through sieves and is pumped to the drying-floors (for fuller description of plant see Section III.). The dried slurry should show no visible specks of chalk, but it often does. Such specks are mischievous, because they indicate the presence of free lime in the finished cement. The fuel used is of importance. In England coke is most commonly employed, but it is supplemented by anthracite and coke breeze, these latter being resorted to when the price of gas-coke is high. Gas fuel is used at only one works in England, and there in an experimental manner. No furnaces heated by petroleum exist, this being in contrast to American practice; the price of the fuel is of course a

governing factor in this respect. Solid fuel being the commonest in England, the ash of the coke, breeze, etc., remains with the finished cement; it is found to distribute itself fairly evenly through the clinker, and, being rich in silica and alumina, increases the content of acid oxides in the clinker and correspondingly diminishes the percentage of lime. Consequently slurry is usually prepared with a larger proportion of chalk than corresponds with the percentage of lime desired in the finished cement, the excess of chalk being required to neutralize the acid ash of the fuel. The commonest proportion of calcium carbonate, as determined by the calcimeter, is 75% in the dry slurry. This percentage may be varied a few per cent either way according to the composition of the clay and of the ash of the fuel. At present this adjustment is generally done empirically; only a few works are, as they should be, under the control of a competent chemist. In cement made in England the raw materials are usually fairly free from sulphur; the amount present corresponds with about 1.5% of SO_3 in the finished clinker. Most of this is derived from the fuel; and, generally speaking, the more heavily burnt a cement is, *i.e.*, the larger the proportion of fuel which has been used, the higher will be its content of sulphur. This is of importance, because the proportion of SO_3 in the finished cement influences the setting-time. The clinker obtained from an ordinary intermittent kiln (see Section III.) according to English practice is not homogeneous; it contains a certain amount of underburnt stuff and of "fallen" clinker, *i.e.*, clinker which has spontaneously disintegrated to dust from the presence in it of dicalcium silicate due to local excess of silica in the slurry. The clinker, as it is drawn from the kiln, is not generally quenched as is sometimes done in Germany; this quenching is generally believed to prevent "falling." For the best cement all except sound, coherent, well-burnt clinker is rejected, but in careless manufacture all the dusty clinker and much underburnt finds its way to the crushers. Careful picking of the clinker is best done on a traveling band and by laborers employed to pick competitively, the wages of one depending on the quantity of underburnt material which he succeeds in picking after his colleague, whose own wages are proportionately diminished. The picked clinker is usually ground to a fineness such that about 10% residue is left on a 76 by 76 mesh sieve. The grinding is often a good deal finer than this when the plant permits of it and the market demands it. A great increase in fineness has lately taken place in England, cement-makers being forced thereto both by the demands of engineers and by stress of German competition in neutral markets, notably in the United States.

The finished cement is stored and packed usually without special precaution for aerating or slaking any free lime which it may contain. Some large users, notably the English Admiralty, insist on the cement being turned over and aerated for days or weeks, but there is at present no recognized system of effecting this slaking in any way comparable with the method of maturing, in use in France, for the preparation of hydraulic lime of high quality.

The principles underlying the manufacture of Portland cement apply generally to the manufacture of other cements of the Portland class, such as hydraulic lime and Roman cement, save that, as these materials are usually burnt in their native state, no attempt to produce an intimate mixture of their acid and basic constit-

uents is made. On the whole, their preparation is rough and ready, and the systematic maturing of hydraulic lime, such as that adverted to above as practiced in France, appears to be unknown. The reason that it is possible to use hydraulic lime and Roman cement, which undoubtedly may contain free lime, is that, as they are not so heavily burnt as true Portland cement, the slaking of the surplus lime is more readily effected when they are used with water, and slow slaking after setting, causing flying in the finished work, is not so likely to occur. The object at which the chief English makers are now aiming is to conduct their manufacture on lines which accord with the theory of the constitution of Portland cement as laid down by Le Chatelier. This theory indicates that as near an approach as possible should be made to the ratio $\frac{CaO + MgO}{SiO_2 + Al_2O_3} \leqq 3$. In this equation the values for both basic and acid oxides are reckoned in equivalents, and not in percentages. Any method of stating the ratio of basic to acid constituents of cement in terms of percentages is valueless. An ordinary English cement containing about 62% of lime, 1% of magnesia, 22% of silica, and 8% of alumina will give the value $\frac{CaO + MgO}{SiO_2 + Al_2O_3} = 2.54$, thus falling considerably below the ideal value. The difference from the calculated value is a measure of the present imperfection of manufacture, and it is being constantly reduced as improvements in method are effected. A cement made in accordance with these principles and approaching the ideal will contain a high percentage of lime, *e.g.*, 64 to 66%, and will be entirely free from underburnt clinker and therefore of free lime. It will need heavier burning than an ordinary cement, but, as a commercial compensation, will be much stronger and will be capable of bearing dilution with a larger proportion of an aggregate in concrete-making. Such accurate adjustment of the acid and basic oxides as will then be necessary can scarcely be attained by the use of any fuel containing ash; this will restrict the choice of fuel to liquid and gaseous kinds. Cement corresponding in quality with the conditions laid down above has been prepared in our laboratory, and its quality is greatly superior to that of any known commercial cement. Another manufacturing point which is beginning to engage the attention of English makers is the substitution of some kiln-lining which shall not be acted on by the clinker, for the firebrick at present used. Hitherto no successful use of a basic lining, *e.g.*, magnesia bricks, can be recorded, but partial success has attended the use of bricks made from bauxite, which are nearly neutral. It is believed by many manufacturers that clinker can be overburnt; this is true of the ordinary product which does not contain sufficient lime to bring it approximately to the ratio given above, but it does not apply to clinker containing an adequate (though not excessive) quantity of lime, as is proved both from our own experiments and one due to Michaëlis, who found that a clinker with a large proportion of lime can be actually fused in the oxhydrogen blowpipe flame and yet yield cement of excellent quality.

The conditions necessary to be observed for economical and successful manufacture are somewhat onerous, and at present they are not completely fulfilled in England. Methods of checking the quality of the product will be dealt

with in Section IV. At the present time in England it is customary merely to check the composition of the slurry in a rough way; the chalk and clay are loaded into the wash-mill in barrow or truckloads, which are roughly weighed or even only counted, their average weight being assumed; in the better-managed works samples of slurry are taken at fairly frequent intervals, are dried, and the percentage of calcium carbonate which they contain determined by means of the calcimeter.* This method allows of fairly rapid though not immediate rectification of errors that may be detected, for part at least of the cement will have found its way to the drying floors before an alteration of the wash can be made. Further, the mere determination of calcium carbonate tells nothing of the ratio existing between the silica and alumina plus oxide of iron in the clay, but in practice this is not of great importance if the clay be of uniform quality, a matter which can be ascertained by periodical analyses. When the clay contains lime in other forms than calcium carbonate, indications of the calcimeter will have to be correspondingly corrected. The method, sometimes used by continental makers, of igniting the slurry at a low temperature and determining the lime by titration is not to be recommended unless much of the lime be present in forms other than carbonate; as, for example, when a limestone instead of a chalk is used, and it is burnt previous to grinding it with the clay. This does not often happen in English practice. The works' analyses of cement raw materials comprise determining the moisture as frequently as possible in both chalk and clay, watching every fluctuation of the latter in its content of silica, ascertaining the coarseness of the free sand present in the clay, estimation of the ash of the coke used as fuel, and analyzing the finished cement. Such a complete method of control is only in use in a few works; generally it is thought sufficient to make mechanical tests of each batch of cement. These tests are similar to, but rougher than, those described in Section IV. They are usually confined to the determination of the tensile strength of neat cement at seven days, but in the better-managed works may include a hot test for soundness and the determination of the specific gravity of the cement. It may be said with regard to the whole of the foregoing section that the English cement industry is only now emerging from the control of rule-of-thumb methods and is being slowly put upon a scientific basis.

Another matter which has to be regarded by the cement-maker is the time of setting of his product. This depends chiefly on the richness of the cement in tricalcium aluminate, but the hardness of burning of the clinker and the age of the finished cement are also factors of importance. It may happen that a particular batch of cement may be turned out with a shorter or longer setting-time than is required for the work for which it is intended, and this is often the cause of disputes between makers and buyers. English engineers specify setting-times varying from one-quarter hour to five hours. For the American market slow-setting cement is generally required, and is sometimes prepared by adding 0.5 to 1% of gypsum to the clinker as it goes to the crushers. A similar practice obtains in Germany, where it is recognized by the German Association of Cement-makers;

* The calcimeter is an instrument in which the volume of carbonic anhydride evolved on dissolving a slurry in hydrochloric acid is measured, and the percentage of carbonate of lime thus ascertained.

but in England no such recognition exists, and much difference of opinion obtains as to the legitimacy of the device. It is sufficient to say here that the addition of gypsum up to 2% has not been shown to prejudice the strength or soundness of the cement. Further, the retarding effect of gypsum diminishes on keeping the cement, and a sample may actually set quicker after storage than it did at first. Probably the best method of slowing the setting of a cement consists in aerating the freshly-ground cement in some more systematic manner than by mere turning over under sheds. At the present date this plan is only in experimental use in Great Britain.

SECTION III.—PLANT USED IN MANUFACTURE.

There is a notable difference between the plant commonly in use in Great Britain and that employed on the Continent. In the former, the raw material most commonly adopted as a source of lime is chalk, and in consequence the Continental plan of making rough bricks of powdered limestone and clay is not in vogue. Generally speaking it may be taken that the plant chiefly employed in English cement-works consists essentially of some form of mixing machine, supplemented or not by a wet grinding plant, followed by an arrangement for drying the slurry thus obtained, and then succeeded by kilns, crushers, and grinding mills for the finished clinker, arranged in the order named. In detail the plant in use varies largely in different works, but a few principles which are almost universal, or tendencies which are becoming of increasing importance, may be discerned.

(1) *Mixing of Raw Materials.*—The old plan of mixing the raw material with a large quantity of water in a wash-mill, separating the fine product from the coarse residue by subsidence and collecting the fine suspended material in large settling tanks, or backs, is now nearly obsolete. The semi-wet process (often erroneously termed the semi-dry, a term properly applied to the formation of rough bricks from slightly damped materials) is the method most commonly used nowadays. In this the chalk and the clay are filled into a wash-mill, broken up and intimately mixed, and the slurry obtained in this manner is then frequently pumped directly upon the drying floors. In the best works the slurry passes from the wash-mill through edge-runners or between stones, so as to crush those small particles which have escaped comminution in the wash-mill. It may be said that the fine grinding and intimate admixture of the raw materials while in the state of slurry is of fundamental importance, and that it is impossible to make good cement unless this condition be observed. Where limestone is the calcareous material in use, two processes are available; the one consists of grinding it and mixing it with the clay in a slightly damp state and molding it into bricks, while the other involves the calcination of the limestone and the mixing of the lime therefrom with clay in the same manner. The relative advantages of these depend upon the cost of fuel and of power; the latter depends jointly on the cost of fuel and on the interest on the capital sunk in the necessary prime mover. The fineness of the raw materials is assured in the best works by the use of wet sieves after the slurry has passed the wet stones, but this use of sieves is not very

common. Their employment will doubtless become universal when it is generally understood that slurry containing coarse particles of chalk cannot under any circumstances produce an unexceptionable cement.

(2) *Apparatus for Drying the Slurry.*—In the older works the slurry is still dried by independent fires or by the waste heat of coke ovens. The continued employment of this system is rather to be ascribed to the disinclination or inability of manufacturers to adopt a modern plant than to any inherent advantage presented by it. In modern practice the waste heat of the kilns is invariably utilized. It is evident, in this case, that the cement-maker is dependent for the coke which he needs on the existence of a gas-works within reasonable distance of his own factory; in England this is usually the case, and gas-coke, being a commodity difficult to dispose of in large quantities for other purposes than cement-making, can usually be bought fairly cheaply. Thus it will be seen that the conditions of manufacture are radically different from those obtaining in the United States, where coal-gas, properly so called, is not so largely used as it is in England. Where coke is made on the works, and the waste heat of the kilns is not utilized, the simplest form of kiln for burning the clinker suffices (see below).

When the waste heat of the kiln is utilized each kiln is provided with a wide exit-flue for its waste gases, which are caused to pass over or under the drying floors upon which the slurry is spread. These horizontal flues are about 65 to 70 ft. in length; if longer, it is found that the water evaporated from the part of the slurry nearer the kiln is apt to be condensed again at the end of the drying chamber. A group of such chambers is connected with a tall chimney-shaft, so as to get the necessary draft. The slurry from the wash-mill and wet stones is pumped to the drying floors, but the dry slurry is generally broken up by hand and conveyed in barrows to the kilns. There is no doubt that the labor item in cement manufacture is unduly large and will be eventually reduced. When dry, the slurry has a considerable coherence, and in drying breaks on the floors of the drying chambers into blocks convenient for loading into the kiln. A more gritty material than chalk will not yield without pressure blocks of the requisite texture, and friable blocks, when loaded into the kiln and burnt, are apt to fall to pieces before clinkering sets in, and impede the draft, yielding inferior cement. A system has been devised by Mr. Wilfrid Stokes for drying the slurry more systematically and with greater economy of heat than is practicable with ordinary drying floors. His method forms part of his complete system of cement manufacture, which is described briefly in a separate paragraph.

(3) *Burning the Slurry to Clinker.*—English practice is tolerably uniform in this stage of the process. It is true that a great number of bottle-kilns are still in use, but when they are pulled down they are generally replaced by chamber-kilns of one type or another. Both the bottle and the chamber-kiln are simply upright brick structures lined with firebrick and provided with a rough grate at the bottom; the former is open at the top, but the latter is connected by a side flue with the drying chambers mentioned above. In each case the slurry and fuel are loaded into the kiln in alternate layers, in precisely the same manner as obtains in a running limekiln; the operation of the kiln differs, however, in the important respect that it is intermittent instead of being continuous. The charge takes

about 2 to 3 days to burn off, and, including the loading, burning, cooling and drying, the whole cycle of operations generally takes about a week. This plant, which is the commonest in England, is thus of the simplest description. The adoption of continuous kilns of the Dietzsch or "stage" type is not usual in English practice. It is alleged that the clinker made in such kilns is inferior to that prepared in intermittent kilns, and also that the cost of labor and repairs is heavier. Stage-kilns are, however, used here and there, and are largely employed on the Continent; they are generally built in pairs, back to back. The Hoffmann or "ring" kiln is almost unknown in the cement trade of Great Britain. The reason is that very little cement is made from raw materials which need molding into bricks suitable for the loading of a ring-kiln.

In almost all cases the kilns are lined with firebrick, which is a most unsuitable material, being a substance composed of acid oxides and likely to be considerably fluxed by the basic clinker; consequently the lining needs constant repair. Some attempt is made to mitigate this evil by plastering the inside of the kiln with slurry, but, in any case, the corrosion is apt to be severe. No successful attempt to use a basic lining can be put on record in spite of the obviously rational character of such an attempt.

(4) *Grinding-plant.*—Until about ten years ago French burrstones were almost universally used in England; at the present time a great diversity of practice exists. In many works stones are being replaced by edge-runners, tube mills, ball mills, and various other contrivances of the same class in which the action is not purely shearing, but partly percussive. It is, however, difficult to ascertain how far these alterations may be reckoned improvements. It is claimed by many makers that the more modern plant has a greater output per horse-power hour than has the old method with stones. It is stated, on the other hand, by the users of stones that a finer product is obtained therewith, and that the cost of repairs is smaller. It is also maintained that the cement particles are more regular in character than when edge-runners, ball mills, etc., are used. This statement is inherently improbable and is, moreover, of no practical significance if true, because if a cement particle be sufficiently small to react fully with the water with which it is gauged and thus to suffer complete change of form and composition, its shape before the reaction is immaterial. Whatever process of grinding be adopted, there is a growing tendency to use sieves or separators to size the finished product. When large quantities of cement per horse-power hour are turned out by any system of grinding, there is a practical certainty that much material will be imperfectly ground, and the coarse particles must be separated and returned to the feed-hopper. The particles thus sent to be reground are mixed with the portion which is already finely ground, a practice which tends to yield a product somewhat less homogeneous than one which is ground together at one time. Moreover, when a given quantity of cement is ground until the whole of it is as fine as will pass modern standards, and the grinding has been done without the separation and restoration of any part of the cement, it is certain that a large fraction of the cement will have been ground to an exceedingly fine, "floury" condition in the effort to reduce the coarser particles. Thus a better product will be obtained than by more systematic working, but of course at a

higher cost for power. The fineness generally attained by English cement-makers is such that a cement will leave about 3% of residue on a 50 by 50 mesh sieve made of wires .007 in. in diameter. Such a cement will leave about 10% residue on a 76 by 76 mesh sieve. But it does not follow from this that two cements leaving identical residues on these sieves will necessarily contain the same quantity of very finely ground cement.

On determining the percentage of very fine particles, *e.g.*, those which pass through the finest sieve in ordinary use (160 by 160 meshes per sq. in.) it will be found that the percentage of "flour" will vary with the method of grinding. With a grinding plant working in a systematic and selective manner the tendency is to turn out a cement which consists largely of very fine grit and contains a smaller quantity of impalpable powder than does a cement ground in a less systematic (and more costly) way.

(5) *The Stokes Process.*—This process is the invention of Mr. Wilfrid Stokes, who has devoted much time and money to the perfection of the system, which, however, can scarcely yet be said to have passed beyond the experimental stage. The object aimed at is the production of a good, sound uniformly burnt clinker, free from the disturbing influence of the fuel-ash, and at a reduced cost both for labor and burning, due to the mechanical means employed, the

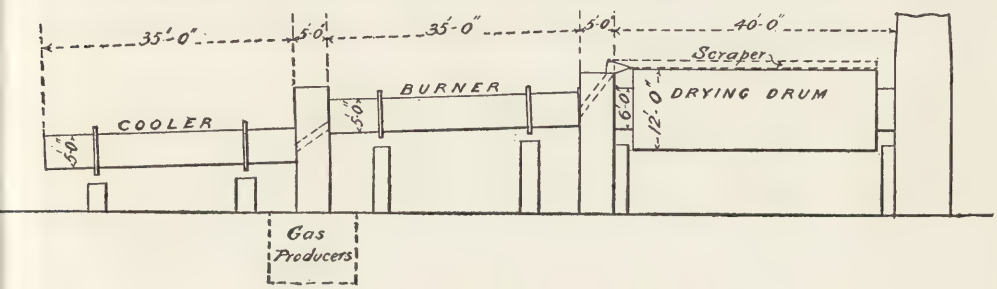


FIG. 1.

use of gaseous fuel and the continuity of the operations. The apparatus used consists of three main parts: a drying drum for the slurry heated by the waste products of combustion on their way to the chimney; a revolving burning cylinder; and a revolving cooling cylinder in which the burnt clinker is cooled by the passage over it of the air necessary for combustion in the burning cylinder. The slurry is prepared in the usual manner in the wash-mill and wet stones according to the semi-wet process. It is then fed upon the top surface of the revolving drying drum in such quantities as will be dry in half an hour, by which time the drum has nearly completed one revolution, when the dried slurry is carried by a traveling scraper-conveyor along the surface of the drum and delivered into a chute formed in the brick pier constructed between the ends of the drum and the burner. The drum is not lined, and is carried on rollers at each end. The burner is a revolving furnace carried on rollers and driven at a speed of one revolution per minute. It is set on an incline so that the material gradually finds its way from the upper to the lower end, from which it falls into a chute in the pier brickwork and thence into the cooler. In this pier the gas

from an ordinary gas-producer with forced steam-draft (so as to generate gas of high calorific value) is delivered into the lower end of the burner, where it meets with the highly heated air arriving through the cooler, thus producing a flame of the required temperature, easily under control. The burner is lined with a basic refractory material so as not to combine with the clinker. The slurry travels through the burner from end to end in about 25 minutes, and issues in small fragments in the original shape of the dried slurry removed from the drum. Samples of the clinker are taken from time to time so that the man in charge may judge whether the temperature be sufficient. The cooler is very similar to the burner, but is fitted inside the lining with a central group of metal gills to collect the radiant heat from the clinker and pass it on to the air by contact.

On issuing from the lower end of the cooler the clinker is comparatively cool, and may at once be ground, the smallness of the pieces obviating the use of a crusher, and reducing the cost of power for the mill. The apparatus is attended by one gas-producer man, who is also in charge of the burner, one man at the drying drum to regulate the slurry, etc., one man at the cooler to remove the clinker, and one boy wheeling coal for the producer. The power required to drive the plant is small. In practice it is found that no unburnt clinker and no dust due to the dicalcium silicate are produced, so that the cement is of the highest quality. The subjoined diagram serves to make the above description more easily understood.

SECTION IV.—QUALITY OF CEMENT OBTAINED.—METHODS OF TESTING AND ANALYSIS.—ADULTERATION.

Portland Cement.—Portland cement of all makes as it comes into the market is generally of similar appearance, and possesses properties which are characteristic of all samples as a class, but its quality nevertheless varies considerably in detail. The methods of testing which are in use throughout the world are designed to discriminate between cements varying thus in quality. The following are the chief criteria adopted:

(1) *Color.*—Good Portland cement made from the usual raw material is of a cold gray color with a faint tint of blue. The presence of more than the usual quantity of iron (3 to 4% calculated as ferric oxide) generally causes the production of a cement of a yellowish or reddish tint. This applies chiefly to cements which have not been particularly well burnt. A larger quantity of iron may be present when the product is hard burnt without causing this "foxiness." The color is relied upon by some engineers as a test of the quality of cement; but although it is safe to say that a cement departing largely from the usual standard of color is in some degree abnormal, the difference is not sufficiently regular to admit of any valuable deduction being drawn. No reference of the color to any color standard, as is possible by the use of a modern colorimeter, has been generally adopted.

(2) *Fineness.*—The fineness of ordinary cement made in England is such that about 10% is retained upon a sieve having 76 by 76 meshes per sq. in.

and made of wires .005 in. in diameter. In all determinations of fineness it is important to make sure of the sieve having the correct number of meshes, and being made of wire of the standard diameter, otherwise a great discrepancy in the results will naturally occur. The nominal values stamped on the sieves are often erroneous. A great deal of cement is made according to special requirements, so that a smaller percentage than 10% is left on a 76 by 76 sieve, and in some cases as little as 10% on 120 by 120 may be obtained; this is, however, the exception. The old standard of 10% residue on a 50 by 50 sieve is rapidly becoming obsolete, as stated in a previous section. The tendency of modern cement manufacture is to prepare a more finely-ground product. The best method of obtaining this fineness, which can only be done by the expenditure of a good deal of power in grinding, has been dealt with in Section III. In general it may be said that, up to a limit which is fixed by the increased cost of grinding cement exceedingly fine, it is advisable to turn out the most finely ground material that can be prepared, for two reasons: (*a*) that finely ground cement goes farther in mortar and concrete work than one which is coarsely ground, and (*b*) that there is less likelihood of a finely ground cement containing unslaked particles than there is of a coarsely ground cement, because the former is more readily aerated than the latter.

(3) *Weight per Bushel.*—This test was devised to discriminate between well-burnt and underburnt cement. Given an equality of fineness, a well-burnt cement will weigh more per unit volume than one which is underburnt. By unit volume is meant the volume apparently occupied by a given weight of cement delivered into a standard measure under standard conditions. The usual measure adopted in England is the imperial bushel; on the Continent the weight of a liter measure filled with cement is generally used. These tests are fast becoming obsolete, because a finely ground cement packs less closely than one which is coarsely ground and therefore seems to be specifically lighter. On this account the test is being rapidly replaced by determination of the specific gravity.

(4) *Specific Gravity.*—This is ascertained by determining the volume of a liquid which does not act on cement (such as dried oil of turpentine) displaced by a known weight of cement.

For rapid routine work in the determination of the sp. gr. of cement we have devised an extremely simple form of volumenometer, which is free from most of the defects of those generally made. It consists, as will be seen from Fig. 2, of a small flask very much flattened in shape, so as to insure its stability, and provided with a short slender neck with fine graduations, not ground into the lower part of the vessel, but blown in one piece therewith.

The lowest graduation mark on the neck is not 0 c.c. but 14 c.c., and the capacity of the flask to the first graduation mark is known and equals 64 c.c. The instrument is used by running into it 50 c c. of oil of turpentine from a pipette which has been accurately calibrated against the flask. As the oil of turpentine is introduced by means of a pipette, the neck of the apparatus is kept dry, and the known weight of cement (50 gms.) can be readily added through a small funnel without blocking the tube. The shortness of the neck conduces to the easy introduction of the cement. The plan of first placing a known weight of cement

in the flask and then running in 50 c.c. of liquid appears at first sight preferable to the method described above, but the complete displacement of the air entangled in the cement is less easily accomplished than when the cement is caused to shower down through the liquid. The temperature of the oil of turpentine must be known, but need not be fixed at any standard point, and after the operation the volumenometer (which is stoppered to prevent evaporation) is brought to the same temperature by immersion in the same vessel of water as that previously used for the stock-bottle of turpentine. As all ordinary hydraulic cements have a specific gravity such that the displacement of 50 gms. falls between 14 and 20 c.c., this range of graduation on the stem is ample; but it is obvious that with any material differing largely from sp. gr. 3, more or less than 50 gms. can be taken so

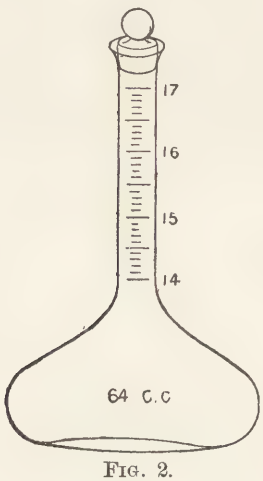


FIG. 2.

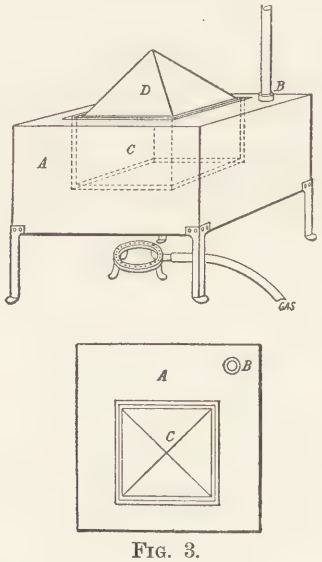


FIG. 3.

as to bring its displacement within the range. Seeing that, in all cases, the temperature before and after the displacement of the liquid by the cement must be identical, because of the high coefficient of expansion of oil of turpentine, it is evident that an apparatus of small total capacity, such as that just described, is convenient, as it can be immersed bodily in a beaker of water and brought to the proper temperature with ease and rapidity. The indications of the instrument are in no way dependent on alterations of the specific gravity of the liquid selected which may take place on keeping, so that a large supply of dry oil of turpentine (or petroleum) can be prepared at one time and used to the last drop without further examination.*

Good well-burnt cement, as it comes from the mill, has a specific gravity of not less than 3.15; it is generally higher than this, and may even reach 3.20. The specific gravity falls rapidly when the cement is aerated, and this must be considered in conjunction with the percentage of constituents absorbed from the air,

* Always provided that the oil of turpentine be not kept so long as to resinify appreciably, as in that case its viscosity increases and a decrease of the quantity delivered by the standardized pipette occurs.

viz., water and carbonic anhydride, as shown by the analysis of the cement. A cement containing more than the usual quantity of ferric oxide (3.5 %) will have a higher specific gravity than one with the ordinary percentage; and this must also be considered in judging the quality of a cement from its specific gravity.

(5) *Time of Setting.*—The time of setting of cement is commonly determined, on the continent and in England, by ascertaining the time which must elapse from the gauging of a pat until it resists the impress of a needle having a sectional area at the point of 1 sq. mm. and loaded with 300 gms. An alternative test much in use in England is with a point having a sectional area of $\frac{1}{16}$ in. square and loaded with $2\frac{1}{4}$ lbs. This is about equivalent to the continental standard already mentioned. The quantity of water used to gauge the cement should be determined in the following way: The cement is gauged to "normal consistency"—by which is meant the consistency produced by the addition to the cement of so much water as will enable a cylindrical rod 1 cm. in diameter to sink to a depth of 3.5 cm., until it is within 5 mm. of the bottom of the vessel containing the freshly mixed cement. The setting-time is then determined on cement gauged to this "normal consistency." As a matter of fact the determination of the quantity of water necessary to bring the cement to the correct consistency is more often left to the judgment of the gauger than ascertained by the method described. A larger quantity of water than is necessary to bring the cement to normal consistency makes it set more slowly, the reverse being true of a smaller quantity of water. Other things being equal, the higher the temperature of the water and of the air of the room in which the cement is gauged, the quicker it will set. The standard temperature should, however, be maintained as far as possible. The determination of the time of setting is chiefly of value as a test, not of the quality of a cement, but as a means of deciding whether it is fitted for particular work. Thus a cement which is very quick-setting, *e.g.*, sets in 5 to 10 minutes, may be difficult to get properly mixed for making mortar and concrete and into place before setting has begun; and, on the other hand, a very slow-setting cement is unsuitable for use in work which has to be exposed to the wash of the tide soon after it is in place.

(6) *Chemical Composition.*—The chemical composition of the cement commonly made in England has been stated in Section I. Divergencies from this standard are comparatively small when the manufacture is well conducted. Cements made from a sandy clay will generally contain a higher proportion of silica and, above all, of insoluble matter which is in itself inert and moreover indicates a defect in the raw materials or mode of manufacture, and a consequent poorness of quality in the product. The addition of imperfectly burnt clinker, which should be picked out, will also increase this insoluble residue. Clinker burnt in the laboratory and well-burnt pieces from the kiln in actual manufacture will contain considerably less than 1% of insoluble residue. Ordinary English cement as put on the market commonly contains 1% of insoluble residue, and may contain as much as 2 or 3%, in which case it is of inferior quality. The silica generally ranges from 20 to 24%, and, as a rule, is nearer the former limit. The alumina and oxide of iron vary from 10 to 13%, and of this quantity about two-thirds is generally alumina. The lime in modern English cement rarely falls

below 60%, and is sometimes as high as 64%; the average is about 62%. Magnesia in cements made on the Thames and Medway rarely exceeds 1%, although there is no reason to suppose that any quantity smaller than 3% is actively harmful. The sulphur, calculated as sulphuric anhydride, ranges from 1 to 1.8% in England (corresponding to 1.70 to 3.06% of calcium sulphate) unless gypsum has been added, when slightly higher figures will be obtained. Water and carbonic anhydride in fresh cement are present to the extent of 0.5 to 1%, but in an old cement may be as much as 3%; about one-third of this quantity is usually carbonic anhydride, the balance being water. Alkalies are present in only small amount, a good deal of soda being driven off at the temperature of clinkering. Judgment of the quality of a cement from its analysis is only possible to the specialist, and to him best in conjunction with adequate mechanical tests (see below).

(7) *Soundness.*—Owing to the fact mentioned in the first section, that the proximate composition of cement cannot be determined by its usual ultimate analysis, it is necessary to ascertain not only that the cement is made of acid and basic oxides in the correct proportions, but also that these are properly combined.

The specific gravity is a help in this case. When duly considered it indicates whether the cement has been well burnt, but it may be usefully supplemented by special tests for soundness. Those most in use in England consist in exposing thin pats of cement under cold water for 7 or 28 days. If these become buckled or cracked, it is certain that the cement is unsound. As an unsound cement may not show its bad quality when tested thus, especially at the shorter date, hot tests have been devised. The rationale of these hot tests is simply that a given chemical reaction, such as the hydration and setting of cement, can generally be expedited by rise of temperature. As practically applied, one of the best *qualitative* tests, devised by the late Mr. Faija, consists in exposing thin pats of cement to a temperature of 110 to 120° F. in moist air and in warm water. An unsound cement will crack and buckle when thus treated. The test much in vogue on the Continent of heating spheres of cement over a burner is exceedingly rough, and moreover wrong in principle; it is not used in England. Le Chatelier has devised a better test, which consists in measuring by a multiplying lever the expansion of cylindrical test-pieces kept under boiling water. The best test, however, for the absolute soundness of a Portland cement is that formulated by Deval. This, which is based upon a large number of observations, consists in comparing the tensile strength of briquettes (3 sand to 1 cement) kept in cold water for 7 and 28 days and in water at 80° C. (176° F.) for 2 and 7 days respectively. Deval found, and the authors have confirmed his observation, that perfectly sound cement thus treated has approximately the same strength at 2 days hot as at 7 days cold, and at 7 days hot as at 28 days cold. A cement which is not perfectly sound has a lower strength when tested hot than when tested cold, and one which is thoroughly unsound goes to pieces. Under these standard conditions the carrying out of this test necessitates the use of a bath of water capable of being kept at a temperature of 80° C., with only small fluctuations, for at least 7 days. The authors have devised such a bath, which is shown in Fig. 3 and described below.

A is a rectangular copper water-bath completely closed save for a small hole *B*, to which a condenser can be attached. This condenser may be a straight metal

tube, jacketed with water like a Liebig condenser, or, if space permit, a piece of lead or "compo" pipe, 40 ft. long or so, passing out of the room containing the bath and going up the outside wall to a neighboring chimney. An air-condenser of this kind is cheap, convenient, and, if long enough, quite effective even in hot weather. The tube must be vertical, or have at least a sharp rise, and be free from any kink or obstruction, as otherwise the return flow of the condensed water will be impeded. The top of the water-bath *A* is not flat, but has a rectangular depression or well into which the hot bath *C*, to be maintained at the required temperature, fits, clearing the walls of the well, and supported by a flange on the top of the bath *A*. A pyramidal cover *D* is provided for the bath *C*, and, if fitting well, suffices to condense such water-vapor as may be given off at 80° C. The space between the inner and outer bath is either empty or filled partially or entirely with sand or shot for a reason about to be stated. The principle underlying the working of the apparatus is simple enough. The outer bath, which, as it is provided with a good condenser, can be kept boiling vigorously, constitutes a source of heat at a constant temperature, 100° C. The inner bath is separated from the outer by a constant distance, which offers a constant obstruction to the transmission of heat from the outer bath. The temperature of the inner bath is therefore always less than that of the outer bath by a constant number of degrees. The size of the separating space must be determined empirically. It is best made sufficiently great to insure the temperature of the inner bath being somewhat lower (when the space is empty) than that ultimately to be attained. The obstruction offered by the separating space is then gradually diminished by introducing into it successive small portions of any convenient substance in grain or powder, *e.g.*, shot or sand. By a few trials a point is reached when the obstruction to the transmission of heat from the outer to the inner bath is so adjusted that when the former is at 100° C. the latter is at 80° C. It is obvious that fluctuations of the gas supply, provided that it be always large enough to keep the outer bath boiling, have no effect on the temperature of the outer bath, and therefore are equally without effect on that of the inner bath. The only change of external condition likely to affect the temperaure of the inner bath to an appreciable extent is that of the temperature of the room in which the apparatus is placed. When the air-temperature rises, loss of heat from the pyramidal cover, which is the only portion exposed to the atmosphere, naturally diminishes, while the supply of heat from the steadily boiling bath is maintained at its former rate; consequently the temperature of the inner bath rises slightly. The converse is true on a fall in the temperature of the air taking place. Experience has shown that the fluctuations due to this cause are not large in an ordinary laboratory atmosphere, of which the temperature does not vary greatly; they might, however, become considerable if the bath were used in an unprotected shed or other slight building. The apparatus in the form shown is quite independent of any particular method of heating It can be used with a paraffin lamp as well as with a gas-burner, the sole condition being that the source of heat shall always be sufficient to keep the water boiling briskly in the outer bath.

At present but few English cements will withstand this test, a fact to be attrib-

uted to want of perfection in the mixing of the slurry and the picking of the clinker. Since its introduction, however, more attention has been paid to the production of perfectly sound cement, and at the present day certain makers are capable of turning out a product quite able to fulfill the necessary conditions. A fundamental distinction between the Deval test and all others of a similar character is twofold: (*a*) it rests on a *quantitative* basis, and (*b*) it has been formulated after the accumulation of adequate experimental data. Another quantitative test for soundness, which, however, requires much time for its completion, is the measurement of the expansion of a bar of cement in the apparatus designed by Bauschinger. In this, a bar 100 mm. in length is fitted at each end with a metal plate, and the distance between these plates is determined by gripping the bar between center-points, one of which is provided with a micrometer-screw. A normal sound cement tested in this way (the bar being kept under water) will increase in length not more than 0.05 mm., and when tested in air will show a similar contraction. An unsound cement will expand largely in water. The tests should if possible be prolonged for at least 28 days, and thus, except for lengthy investigations, the method is not generally applicable for the detection of a latent unsoundness of the cement.

(8) *Strength in Tension and Compression.*—The tensile strength of cement is generally determined in England on neat briquettes of 1 in. section. The usual process of testing is too well known to need description here; the following points may, however, be noted. Class of testing machine: Multiplying-lever machines the load of which is put on by the running of shot into a bucket, the flow of shot being automatically cut off on the breaking of the briquette, are much used in English cement-works. Single-lever machines consisting of a plain steelyard with a jockey weight, the movement of which is set at a known speed, are, however, to be preferred. The rate of putting on the load influences the result obtained, the higher the speed the greater being the apparent strength of the briquette. A standard rate of 100 lbs. in 15 seconds is largely adopted in England. Sand tests are being much more freely used than they were a few years ago. They are to be preferred to neat tests because the latter do not give full credit to a finely ground cement. The usual mixture employed is 3 of sand to 1 of cement by weight, and the quantity of water used in gauging is $8\frac{1}{2}$ to 10%. The sand used is not a government standard product like that adopted in Germany, but what is practically a standard exists in the shape of sand from a particular bed at Leighton Buzzard in Bedfordshire. This sand, which has a very uniform quality, is washed free from dust and sized by passage through a 30 by 30 mesh sieve and retention on a 20 by 20 mesh sieve. (The use of quartz-sand and of mixtures of 1 part of cement to 2 of sand, such as are sometimes required in the United States, is not common in England.) This standard sand gives lower results than do many building sands, for two reasons: (*a*) its size is too nearly uniform, and (*b*) it contains but little material capable of acting as a puzzuolana with a portion of the lime of the cement. The action here referred to may be briefly explained. When Portland cement sets, the fundamental reaction is represented by the equation $2\ (3\ CaO,\ SiO_2) + 9\ H_2O = 2\ (CaO, SiO_2).5\ H_2O + 4\ Ca(OH)_2$. Thus a considerable quantity of lime is set free as calcium

hydroxide, and is capable of combining with any active siliceous material which may be presented to it. Quartz, which constitutes the bulk of ordinary sand, is not capable of thus combining, but associated with the quartz are silicates or silica of a more active quality. It is these which in uniting with the surplus lime of the cement cause the obtainment of a better result with building sand than with standard sand which is nearly pure quartz. The ordinary requirements for the tensile strength of cement in England are that when tested neat it shall have a tensile strength of 350 or 400 lbs. at 7 days, and one of 500 to 550 lbs. at 28 days, and when tested with sand (3 to 1) that it shall have a tensile strength of 90 to 120 lbs. at 7 days, and 180 to 200 lbs. at 28 days. Good cements will easily pass these tests. The compressive strength of cement is not very frequently used as a means of judging it in commerce. It may be stated as having an average at 28 days of 2700 tons per sq. ft. when tested neat and 1080 tons 3 to 1, *i.e.*, 400 and 180 lbs. per sq. in. respectively. The endeavor to establish a ratio between the tensile and compressive strengths of cement has proved impracticable. With cement which is to be used for sea-work the employment of sea-water for gauging is sometimes specified, and the briquettes are kept under sea-water. No great difference in the results, as compared with those of fresh water, is observed if the cement be sound and the briquettes gauged so as to be non-porous. The mode of preparation of test-pieces of cement greatly influences the results obtained. In general, the smaller the quantity of water used and the more thoroughly it is mixed with the cement and the harder the cement is rammed into the molds, the higher will be the results. To obviate the variations unavoidable in hand-gauging several methods of mechanical gauging have been adopted, but they are not very largely used in England. The Böhme hammer apparatus, which delivers a fixed number of blows on the briquette molds filled with cement, thus consolidating it to a uniform extent, is but rarely employed, although it is freely used in Germany. Much, therefore, depends upon the skill and experience of the cement-tester.

(9) *Adulteration.*—English cements are not usually adulterated whether they be of good or bad quality. There is, however, a certain amount of adulteration practiced, notably in the North of England, where blast-furnace slag is ground up together with the clinker, and the cement thus mixed sold as genuine Portland cement. It is contended by those who adopt this practice that the slag acts as puzzuolana towards the lime liberated in the setting of the cement, and this is probably correct; but whether the result be good or bad it is incontestable that the mixed cement, prepared in the manner described, is an adulterated product and not genuine Portland cement. It should be sold under its own name and not as true Portland cement. Within the last three years another material has been used to a minor extent for adulterating cement, namely, "Kentish rag." Kentish rag is limestone containing a certain amount of siliceous matter which cannot be reckoned as puzzuolanic material as slag may be; ragstone must be considered merely a makeweight and a dilutent of the cement. Like most adulterants it is added to the clinker just before it goes to the crushers, and is used in the proportion of about 15% of the weight of the finished cement. English cement-makers have viewed the use of rag as a whole with great disfavor, and

have discountenanced it in every possible way, and at the present time no important firm is adopting the illegitimate practice of selling as Portland cement a mixture of Portland cement and Kentish rag. Both these and any other adulteration that is likely to be practiced can be readily detected by the expert analyst. Simpler methods of detection, such as those based on the determination of the specific gravity of the cement (which is lowered by the addition of either ragstone or slag) or examination under the microscope, are only useful in roughly sorting suspicious samples. The detection with certainty of adulterated cement, and the determination of the quantity of the adulterant, requires careful analysis as aforesaid. In general it may be stated that the English cement trade is creditably free from the stigma of adulteration.

Other Cements of the Portland-cement Class.—The systematic testing of the less important cements of the Portland-cement class is not very common. In general Roman cement and hydraulic lime, when tested at all, are submitted to tests similar to those already laid down for Portland cement, with such modification as the nature of each material requires. In the same way the analysis of these products will detect adulteration, though this is rarely practiced. It is stated that a small amount of factitious hydraulic lime is made by mixing ordinary "fat" lime with so-called "natural" cement. This product is of course illegitimate, and can be detected on analysis; there is no reason to suppose it to be prevalent.

Cements of the Puzzuolanic Class.—Owing to the small use of cements of this class which obtains in England, the testing of them is not generally practiced. The methods of testing and analysis are generally similar to those already described for Portland cement. One point of special interest is the determination of the fineness of grinding of the mixture; this is particularly important in the case of slag-cement, which requires to be exceedingly finely ground in order that the slag and lime may interact; the freshness of the lime should also be ascertained. It is necessary that the lime should be as free as possible from water and carbonic anhydride, as otherwise the action upon it of the siliceous constituent of the puzzuolanic cement is sluggish and imperfect. Lime which has become in any degree air-slaked is quite unsuitable for the manufacture of puzzuolanic cement. Owing to the slow setting and hardening of cement of the puzzuolanic class, mechanical tests, to be of much value, should extend for a long period, 28 days being usually the shortest time in which valuable results can be obtained.

SECTION V.—THE USE OF PORTLAND CEMENT AS A STRUCTURAL MATERIAL.

It is not our intention in this section to speak of cement as a structural material from an engineering point of view, but to indicate wherein the quality of cement influences its utility when employed for making mortar, concrete and composite structures of cement and iron. Not much cement is used neat, except occasionally for grouting. In general the tendency of neat cement to contract slightly in air and to expand slightly in water militates against its use without the addition of some inert aggregate. For most purposes, therefore, the use of a mortar is

not only cheaper, but better. The chief causes of the failure of cement-mortar are as follows: (1) The cement may be too slow-setting to become fairly hard before the mortar is subjected to some disruptive strain. (2) The cement may be too quick-setting to allow of the mortar being got fairly into place before setting has begun. (3) The mortar may have been mixed with such an excess of water as will leave, when the surplus dries out, a loose and porous material. (4) The mortar may be exposed to drying influences, such as sun and wind, which will remove from it that water which is necessary for its setting, it being remembered that setting is primarily a phenomenon of hydration. (5) The cement may be intrinsically unsound, and may blow and crack at a shorter or longer time after it has been got into place. It will be seen that there are more external than internal causes likely to bring about the failure of cement-mortar, but nevertheless failures undoubtedly due to the bad quality of the cement are by no means infrequent. They arise from the neglect of those principles of manufacture which have been already described and are now becoming generally admitted and observed in the cement trade of Great Britain.

Cement-mortar exposed to the action of water is but little affected if it be impervious. Imperviousness becomes of even more importance with concrete, which is dealt with below. In sea-water cement-mortar is slightly more acted on than in fresh water, because there is a well-marked reaction between the magnesium salts of the sea-water and the calcium compounds in cement. Imperviousness is therefore even more necessary than it is with cement-mortar used in fresh water. With regard to concrete it may be said that the chief points which influence its strength and permanence and thus have to be considered in judging whether the cement has shown itself to be of good quality, and whether any failure is to be attributed to its handling or some inherent defect, are as follows: The aggregate should be various in size, preferably of slightly rough surface and of fairly high compressive strength; the average size of its pieces should be planned so as to yield with the sand which is used in conjunction with it a mass of which the calculated average interstices are reduced to a minimum. Any failure in this respect may be disastrous in sea-work for the reason alluded to above. A block of concrete exposed to sea-water is only affected from the surface inward if it be impermeable, but when it is sufficiently porous to allow sea-water to enter and flow out again and to be replaced by fresh quantities of sea-water, the destruction of the block is only a matter of time. This may be fairly insisted on, as it is not generally realized that any cement, however good, is eventually destroyed by free contact with sea-water, and that the permanence of concrete structures depends largely upon the degree to which they prevent the entrance and reflow of water with which they are in contact. The sulphates of calcium and magnesium which are present in sea-water also exercise a specific action on Portland cement. The calcium sulphate, whether existing in the sea-water as such, or formed from it by the action of the lime compounds present in the cement on the magnesium sulphate of the sea-water, combines with the calcium aluminate of the cement to form a salt to which Candlot has assigned the formula $Al_2O_3, 3\ CaO + 2.5\ CaSO_4 + 80\ H_2O$. This body, occupying a larger volume than that of the aluminate *per se*, causes disintegration of the cement. The compressive strength of con-

crete depends largely upon its richness in cement; that for 6 to 1 material at six months' date may be taken as about 91 tons per sq. ft. It is probable that certain aggregates, notably broken brick, exercise a puzzuolanic action on the lime set free in the setting of the cement. This tendency has been adverted to in the section on testing cements. The testing of concrete can only be carried out on large blocks at long dates; pieces of about a cubic foot in content are necessary to avoid the inevitable irregularities which affect too largely the strength of a small test-piece. As such condition can be rarely complied with, it is important not to rely on testing the finished concrete, but to make certain at the first that the cement is good and that the aggregate is well chosen.

The system of building construction in use on the Continent which consists in the formation of such articles as large pipes, light bridges and fireproof floors from an interlacing network of iron rods embedded in cement-mortar does not obtain in England. Some ten years ago experiments in this direction were tried, but were not an unqualified success, separation of the cement and iron sometimes occurring. This was probably due in great measure to the cements of that date being less regular in quality than those procurable now; and it is not unlikely that the gradual adoption of this system of construction may take place. A great deal of cement is used in England for the production of artificial stone, which is commonly prepared from a hard aggregate, such as granite chippings, and good Portland cement. In one process the surplus lime available after the setting of the cement is rendered useful by placing the finished blocks of stone in a solution of silicate of soda whereby calcium silicates and caustic soda are formed; the latter can then be washed out. Stone of excellent quality may be thus prepared. Modification of the principle of diluting cement with a fine aggregate, such as sand, to form a mortar is found in so-called "sand-cement," which is made to some extent in Belgium and in the United States, but is not manufactured in Great Britain. This material is obtained by grinding the sand and the cement together, the quantity of sand used being often greater than that of cement. The intimacy of admixture prepared in this manner allows the "sand-cement" to be diluted with a fresh quantity of unground sand to make good mortar. Economy of cement is thus obtained, but against this must be set the extra cost of grinding the sand. The practice of selling this "sand-cement" as Portland cement is by no means unknown; it is a gross fraud. One other point in the use of cement, as affecting the quality of the finished work, is the behavior of cement-mortar when exposed to low temperature. It is found that cement or cement-mortar exposed to a temperature near or below the freezing-point of water does not set, or sets very slowly; when, however, it does set it shows no sensible depreciation of quality. The failure to set is simply due to the partial inhibition of chemical action which occurs at low temperature—a general phenomenon. Various nostrums are in vogue for preventing this retardation of setting, the use of carbonate of soda being particularly favored; but where work must be executed in spite of frost, it is probably best to heat the portions immediately in construction by means of steam, proceeding with the thawed work in the usual way.

SECTION VI.—NUMERICAL DATA AND STATISTICS.

The total output of Portland cement in the United Kingdom is about 1,300,000 tons per annum, and of this quantity about 90% is made on the Thames and the Medway. The total amount of cements other than Portland is relatively inconsiderable. As regards general cost of manufacture the figures given recently by Mr. Gilbert R. Redgrave may be taken as fairly representative. The average cost of the chalk delivered is from 1s. 2d. to 1s. 6d. per ton, and that of clay ranges from 1s. 8d. to 2s. per ton. Where limestone has to be quarried, or where the carriage is by rail instead of by water, the cost is necessarily higher and will vary considerably at each works. The average cost of the coke may be taken at from 8s. 6d. to 12s. 6d. per ton, and that of coal for steam-raising at from 15s. to 18s. per ton, but in both items the question of freight is an important one. The labor required can be obtained for a wage of about 18s. to 24s. per week; the wages paid to artisans as distinct from laborers are about 29s. to 32s. per week. Reckoned on each ton of finished cement the labor item is as high as 6s. or 7s.—about one-third of the total manufacturing cost of the material. It is therefore evident that the introduction of labor-saving appliances, *e.g.*, in the form of continuous kilns mechanically loaded and discharged, would have a great influence on the cement trade. The proportion of fuel necessary for the burning of the cement varies considerably with the type of kiln employed. In England, with cheap gas-coke, economy of fuel, though important, is not of so much significance as to lead to the adoption of fuel-saving kilns such as the Dietzsch, which consumes about 20% of fuel reckoned on the weight of the finished clinker. The quantity of coke needed in chamber-kilns is 30 to 35% of the weight of the clinker, and this amount is often exceeded. As stated above, the cost of the coke may be taken as from 8s. 6d. to 12s. 6d. per ton, so that the cost per ton of cement for fuel alone, assuming that the waste heat of the kiln is sufficient to dry off the complete charge for that kiln, is from 3s. to 4s. The cost of casks is about 6s. to 7s. per ton of finished cement. Bags are, however, largely used for the home trade.

The quantity of cement exported last year according to the Board of Trade returns was 353,863 tons. Thus it will be seen that the home consumption is extremely large. The present selling price of Portland cement is quoted at 24s. per ton f. o. b. London in bulk, which is considerably lower than that which obtained a few years ago; nevertheless a modern cement-works run with high technical skill and ample capital is generally a remunerative undertaking.

THE CHEMICAL INDUSTRY.

It is only a few years since there has been a chemical industry in the United States, yet it is already beginning to assume vast proportions, and before long the domestic demand for the heavy chemicals ought to be fully met by American producers. The raw materials, the capital, and the technical skill are at hand.

Alum and sulphate of aluminum are made by nine companies. The statistics of their production are given under a separate caption.

Bichromates of potassium and sodium and chromic acid are manufactured extensively at Baltimore, and on a smaller scale at Philadelphia. The processes were fully described in The Mineral Industry, Vol. IV.

Borax, refined and fused, has for a long time been produced in the United States. The manufacture of boracic acid has been undertaken recently. This industry is treated in this volume under a separate caption.

Bromine is discussed under a separate caption in this volume. The entire home demand for bromides is supplied by domestic manufacture.

Calcium carbide is manufactured by the Acetylene Light, Heat & Power Company, at Niagara Falls.

Potassium cyanide, which is now of great importance in gold-winning, is manufactured on a large scale by the Roessler & Hasslacher Company, at Perth Amboy, N. J. The manufacture of cyanides was described in The Mineral Industry, Vol. IV.

Soda Ash, Caustic Soda, etc.—The great seats of the soda industry are Syracuse, N. Y., Saltville, Va., and Detroit, Mich. At Syracuse are the vast works of the Solvay Process Company, of which statistical data are given in the accompanying table. At these works, of course, the ammonia process is employed. At Saltville, Va., are the finely equipped works of the Mathieson Alkali Company, employing a modification of the Solvay process, and at Detroit, Mich., and vicinity, the Michigan Alkali Works, those of Church & Co., and the new works of the Solvay Company. Soda ash, sal soda, and caustic soda are also made at Natrona, Pa., by the Pennsylvania Salt Company, which was, indeed, the pioneer in this branch of industry in the United States.

At Niagara Falls the Mathieson Alkali Company is erecting a large plant (now nearly completed) for the manufacture of caustic soda by the Castner-Kellner process, and Walton Ferguson is establishing works for the production of potassium chlorate (electrolytically) at the same place.

Sodium peroxide is now being manufactured by the Niagara Electro-Chemical Company, at Niagara Falls.

Sulphuric acid is now manufactured extensively in the vicinities of Boston, New York, Philadelphia, Baltimore, Cleveland, and Chicago. There are also many isolated factories in the South, and even so far west as Clifton, Ariz., and San Francisco, Cal. Sulphuric acid is made from the waste gases from blende-roasting at two points, namely, La Salle and Peru, Ill., where the feat of utilizing the fumes from mechanical roasting furnaces has been successfully accomplished, though some difficult obstacles had to be overcome. The roast-gases at La Salle, Ill., contain only 4.75% by volume of sulphurous anhydride, and we fancy that nowhere else are so weak fumes used for sulphuric acid manufacture. The works at Peru, Ill., were newly constructed last year. Another new works was built at Argentine, Kan., to supply acid for the Hunt & Douglas process in use at the smeltery there.

Most of the sulphuric acid produced in the United States is made by burning brimstone and pyrites, the residues of the latter being utilized in various ways. Comparatively little is made as a by-product; that which is produced at Argentine, Kan., and Clifton, Ariz., is employed in the metallurgical processes of the respective works. The only by-product sulphuric acid on the market at present is that of the two zinc-works of Illinois. In 1896 this amounted to 23,300 tons, reduced to 66°, although the whole was sold as chamber-acid.

Superphosphate is manufactured at numerous works from domestic phosphate rock and sulphuric acid. The manufacture of this fertilizer in the United States ought to increase immensely, since with the introduction of more scientific methods in agriculture care will be taken to replace the elements of fertility which are extracted from the soil. At present the American fertilizer manufacturer, unlike the European, is not obliged to compete against the cheap and efficient Thomas slag.

Vitriols.—Green vitriol, or copperas, is made as a by-product by the wire-drawers. The production in 1896 amounted to 11,170 short tons. Blue vitriol, or bluestone (copper sulphate), is made as a by-product on a large scale by the gold and silver refiners. In 1896 the production was 48,732,840 lbs.

MATERIALS CONSUMED BY AND PRODUCTS OF THE SOLVAY PROCESS WORKS, SYRACUSE, N. Y.
(In metric tons.)

	1892.		1893.		1894.		1895.		1896.	
	Tons.	Value Per Ton	Tons.	Value Per Ton	Tons.	Value Per Ton	Tons.	Value Per Ton	Tons.	Value Per Ton
Mat rials used:										
Salt as brine	150,000	$1.00	160,000	$1.00	197,000	$1.00	215.000	$1.00	215.000	$1.00
Limestone	209,840	1.00	199,070	1.00	240,000	1.00	270,000	1.00	265.000	1.00
Coal	173,180	2.25	162.485	2.25	170,000	2.25	200,000	2.25	190,000	2.40
Coke	18,785	3.50	17,220	3.50	20,000	3.50	25.000	3.50	25,000	3.60
Ammonia as sulphate	1,670	70.00	1,600	65.00	1,790	64.00	1,800	65.00	2,400	60.00
Sulphuric acid	.....	.....	.....	.....	700	.....	700	7.00	1,800	8.00
Bauxite	.....	.....	545	.....	160	.....	.....	.....	.....	.....
Products:										
Ammonium sulphate	75	65.00	130	65.00	135	65.00	135	65.00	200	45.00
Tar	276	8.00	420	8.00	415	8.00	420	8.00	820	4.00
Soda ash, 58%	82,000	33.60	85,000	32.00	104,600	23.50	120,000	23.00	122,000	20.00
Caustic soda	23,800	69.00	22.700	66.00	30,000	53.00	36,000	41.00	26,000	40.00
Soda bicarbonate	8,400	39.00	8.949	37.00	9,900	36.00	9,900	36.00	16,000	35.00
Soda crystals	.....	.....	.....	.....	430	40.00	430	40.00	1,400	35.00
Sulphate of soda	.....	.....	100	.....	330	11.00	350	11.00	.....	.....
Crown filler ($CaSO_4$)	.....	.....	.....	.....	700	25.00	700	25.00	1,400	20.00
Hydroxide of alumina	.....	.....	70	66.00	100	66.00	.....	.....	.....	.....

The growth of the chemical industry in the United States may be said to have begun in earnest with the establishment of the Solvay Process Company at Syracuse, N. Y., in 1884. This company has built up a great industry so successfully that its works have now grown to very large dimensions, and it has recently established a new plant near Detroit, which adds very largely to its productive capacity. The original location near Syracuse was made chiefly on account of the proximity of abundant supplies of salt, the raw material of the industry. The company owns a number of wells, the brine from which is piped to the works. Limestone is supplied from the company's quarries at Split Rock.

At the commencement the company acquired the title to the American patents of Ernest and Alfred Solvay, the originators of the business, who provided for independent organizations, one for each of the great countries whose consumption of alkali warranted a home supply. These national organizations are entirely independent, and each is under exclusive home control, but each has the privilege of sharing in the improvements of all the others, both by constant interchange of technical and factory reports and by personal visits of their staff experts. The business world can hardly offer a parallel to this fraternity, and 30 years' experience has shown it to be as practicable as it is desirable.

The ammonia used by the company is obtained as a by-product in the manufacture of coke. The company has for a considerable time had coke ovens of the Semet-Solvay type in operation at its works, and recently established a large block of these ovens at Dunbar, in the Western Pennsylvania coal region.

An interesting product furnished by the company, the demand for which has been caused by the great progress made in artificial cooling, is chloride of calcium in a liquid form, suitable for use as a circulating medium in refrigerating machines. In order to be suitable for this purpose the calcium chloride must be very free from salt, as the presence of this impurity causes the pipes to rust badly and reduces the efficiency of the circulating medium, since the presence of salt raises the freezing point of the solution. Calcium chloride as prepared at Solvay is contaminated by no salt whatever, and has a strength of about 40° B., containing about 39% actual calcium chloride. The impurities present consist of a very small amount of calcium sulphate, and a small amount of magnesium chloride. The sp. gr. of this solution is about 1.384, and the freezing point is about 6.7° C.

The range of the Solvay Company's products as at present made is shown by the following list: 1. Soda ash: including 58% for easy dissolving; dense 58%, for melting purposes; 48% for general commercial use; special 48%, for glassmakers; 36% for special chemical operation. 2. Caustic soda: including high test, 76%, nearly chemically pure NaOH; 74%, 70%, and 60%, the three customary commercial grades; special 70% and 60%, made softer than the ordinary grades. 3. Soda crystals: including monohydrate crystals, containing 49.8% actual alkali, and snowflake crystals, with 40.9% actual alkali. 4. Bicarbonate of soda: including pure bicarbonate, 99% $NaCO_3$, for baking soda, and anchor dust, an inferior grade for making carbonic acid. 5. Crown filler, a pure hydrated sulphate of lime for surfacing papers. 6. Chloride of calcium, for use in refrigerating machines. 7. Muriatic acid.

The New York Chemical Market in 1896.

Heavy Chemicals.—The heavy chemical market during 1896 was a rather uneventful one. This was due mainly to the harmony which exists among the various producers, and to the dullness in trade that affected every line of business during the greater part of the time. This dullness influenced the course of prices rather than the amounts consumed, which in most cases exceeded the imports during 1895. In two notable instances, however, the imports decreased very much, owing to the supply manufactured here at home supplanting the foreign article, which formerly had no opposition from domestic manufactures.

Caustic soda was one of these exceptions, the falling off during the year 1896 from 1895 being nearly 12,000 tons. In January the market opened with quotations of 2.25@2.37½c., fell slightly by the middle of the month, but regained the first figure by the middle of March. Toward the latter part of April the quotation for smaller shipments reached 2.45c. At the beginning of May the lowest values for the year were obtained, 2⅛@2$\frac{3}{16}$c., but three weeks later prices had recovered to 2⅛@2¼c., which has been held ever since. These figures are for 72, 74 and 76% strengths.

Caustic potash had a quiet time. Quotations at the beginning of January were 80@90c., which were maintained for just one month, when they increased to 85@95c. These prices in turn held until April, when 80@95c. was obtained. Before the end of the month 90c.@$1.15 was asked, which early in May was shaded first to 87½c.@$1.15, and then dropped to 77½@87½c. By the end of the month the figures had changed to 80@95c., the same that they had been just two months previously. Toward the end of July the market stiffened somewhat and 80c.@$1 had to be paid. No further change was noted until the middle of September, and then the increase was very slight, 82½c.@$1 being asked. The last change of the year occurred during the second week in November when the price fell to 70@90c.

Sal soda again showed a large decrease in the amount of imports, the difference between 1896 and 1895 in sal soda and ash being no less than 58,000 tons. The difference is, of course, made up by the production at home. Quotations at the beginning of the year were 65@70c., which in April had fallen to 60@65c. From this time on they varied slightly, and before the middle of June advanced to 67½@70c., which was followed immediately by a further increase to 70@71½c. There was no further change until near the middle of November, when prices fell to 62½@67½c.

The heavy increase in importations of bleaching powder in 1895 over 1894 (about 20%) was not continued in 1896; but, on the contrary, the first 10 months of the year showed a decrease of more than 6,000,000 lbs. from the same period in 1895. This deficiency occurred toward the close of the year, when bleaching powder was scarce for quite some time. October alone showed a decrease of nearly 2,500,000 lbs. from the imports during the same month in 1895. January opened with quotations at $1.85@$2; they fell to $1.65@$1.90 by the middle of the month, and recovered to $1.85@$1.90 by February 1. Before the middle of March a decline occurred to $1.70@$1.80,and in another month there was a fall to $1.62½@$1.75. Toward the end of April prices increased

slightly, but May opened with the lowest figures reached during 1896, namely $1.50@$1.55. However, they recovered almost immediately to $1.62½@$1.75, yet declined to $1.50@$1.75 by the end of May. Early in June $1.87½@$1.95 was asked; two weeks later this became $1.75@$1.85, which, with only slight changes, held to the end of the year.

Nitrate of Soda.—There was an almost continual variation in the price of nitrate of soda throughout 1896, although the fluctuations were within narrow limits. The year opened with a quotation of 1.72½@1.75c., which fell to 1.70c. by February. With slight changes occasionally, this figure was practically held until the middle of May, when 1.75c. was reached, and 1.80c. by the end of that month. This price remained invariable throughout June (the only month of the year in which there was no change), but fell early in July. By the end of July it more than recovered, 1.80@1.82½c. being realized, but for a short time only. During August the price fell slightly, and early in September reached 1.75c. After this it increased to 1.80c., but again fell to 1.75c. in the early days of October. From this time forward, however, there was a gradual rise, first to 1.80c. and then to 1.82½@1.85c. by the end of October. Soon after the middle of November the quotation was 1.85@1.87½c., and before the middle of December reached the highest mark of the year, namely, 1.92½@1.95c.

The following table shows the imports of chemicals for the five years ending with 1896:

IMPORTS OF CHEMICALS INTO THE UNITED STATES. (IN POUNDS.)

Year.	Nitrate of Soda.		Bicarbonate of Soda.		Caustic Soda.		Sal Soda and Ash.		Other Soda Salts.	
1892.....	213,456,320	$2,933,174	1,466,595	$25,874	54,384,120	$1,309,500	361,648,637	$4,698,379	40,954,822	$234,853
1893.....	257,626,880	3,673,537	1,380,426	23,136	52,116,492	1,171,878	348,972,506	3,982,772	29,850,109	205,523
1894.....	219,824,646	3,189,084	4,479,524	63,625	46,554,322	919,197	321,300,874	2,665,838	17,667,314	132,385
1895.....	236,959,000	3,778,360	8,862,028	123,425	72,019,114	1,211,000	307,026,104	2,321,612	9,943,915	155,006
1896.....	258,728,960	3,566,744	4,343,786	60,878	47,827,247	859,809	191,780,537	1,443,107	5,623,071	100,917

Year.	Nitrate of Potash.		Muriate of Potash.		Chlorate of Potash.		Chloride of Lime.	
1892.....	13,012,087	$382,771	70,227,971	$1,098,267	3,568,751	$365,326	100,888,561	$1,962,084
1893.....	13,374,016	369,274	74,663,116	1,192,516	3,881,791	481,839	98,618,148	1,843,410
1894.....	9,375,950	249,842	101,597,074	1,540,081	4,599,969	573,154	96,256,251	1,697,038
1895.....	11,419,090	505,207	81,732,736	1,296,164	4,917,636	456,342	104,317,968	1,628,877
1896.....	20,085,327	470,899	86,525,983	1,392,594	4,659,893	366,360	99,471,335	1,441,600

Acids.—During the year 1896 the trade in acids was without special feature, particularly during the first six months. Business during this time was fair and steady, but consisted mainly of deliveries on yearly contracts. There was considerable competition among the dealers for the jobbing trade doing, to secure which prices were more or less shaded. In the spring months business fell off somewhat, but the early summer saw a revival. In June the rise in brimstone was felt in the acid market and dealers endeavored to get an advance of 5c. per 100 lbs. of sulphuric acid on all new business. Although the price of brimstone continued to increase for some months, and was at the close $6 to $8 per ton higher than at the opening of the year, the quotations on acids have not shown any considerable increase. The effect was mainly to make prices very firm and to materially strengthen the market on current business. On contracts for 1897

it has had the effect of increasing quotations 10c. to 15c. over the 1896 contract prices, but even at this figure producers were not very anxious to enter into agreements early in the fall.

Prices opened in January as follows: Acetic acid, $1.40@$1.70; muriatic acid, 18°, 75@80c.; 20°, 80@90c.; nitric acid, 36°, $3.50@$4; 40°, $4@$4.50; 42°, $4.25@$4.75; oxalic acid, $7.10@$7.60; sulphuric acid, 66°, 75@80c.; chamber acid, $6.50@$7.25. These figures were held until the latter part of April, when a general decline took place, and prices became as follows: Acetic, $1.25@$1.30; muriatic, 18°, 70@80c.; 20°, 75@80c.; nitric, unchanged; oxalic, $7.25@$7.50; sulphuric, unchanged; chamber acid $6@$6.50. In the beginning of May a further change occurred, in which nitric acid took a tumble, from which it recovered somewhat, however, almost immediately. Prices were: Acetic, $1.25@$1.60; muriatic, 18°, 65@75c.; 20°, 75@85c.; nitric, 36°, $3.25@$4.25; 40°, $4@4.25; 42°, $4.50@$5.25; sulphuric, 66°, 75@80c. On July 1 the prices were these: Acetic, $1.25@$1.40; muriatic, 18°, 75c; 20°, 75@85c; nitric, 36°, $3.25@$4.35; 40°, $4@$4.50; 42°, $4.50@$5.50; oxalic, $7.25@$7.50; sulphuric, 66°, 75@95c.; chamber acid, $6@$6.50. From that date these figures remained practically unchanged.

Brimstone.—During the first five months of 1896 the brimstone market was without feature, unless an almost unvarying price during that time can be called a feature. Throughout January the ruling spot quotation for best unmixed seconds was $16.50, which on February 1 fell to $16 per ton, and remained at that figure until the latter part of April, when $16.25 was realized. May opened with the lowest prices of the year, namely, $15.50@$15.75, which held, however, only one week, at the end of which $16 was asked per ton for spot or near-by. May also saw the highest point reached so far in the year, the quotation at the end of the month having risen to $17. During all this time thirds were obtainable at 25c. to 50c. less per ton.

The higher price in the latter part of May was caused by the report that steps were being taken to organize an association of the brimstone mine owners in Sicily for the purpose of regulating prices. The May price continued in June with a firm market, until in the middle of the month word was received that the projected association was to become a reality, when a jump of $1 per ton occurred, making the quotation for spot and to arrive $18. Before the end of June more definite information was received about the brimstone syndicate. Among its members appeared the names of Alexander M. Chance, of Birmingham, England; Baron Robert Oppenheim, Baron Eugene Oppenheim, Alfred Gordon Solomon, Charles Wightman and Ignazio Florio, of Palermo, Sicily. The agreement entered into was to be in force five years and stipulated that the mine owners shall not compete with the trust in any way; it regulated the output at 340,000 tons per annum, to be sold to the syndicate at about $14.20 for best unmixed seconds and $13.85 for best unmixed thirds, delivered free on board steamers at the loading places in Sicily. At the same time application was made by the syndicate (Societa Anglo-Siciliana) to the Italian government to abolish the export tax on sulphur, amounting to $2.12 per ton. With the receipt of this information the price of brimstone took another advance, and spot quotations became $19@$20 for seconds and $18.50@$18.75 for thirds.

July opened with a drop in prices, $18.50 per ton being asked for seconds for July shipments, and $18 for thirds. These prices ruled during the first half of the month, after which they increased to $19.25 for seconds and $18.50 for thirds, and still later to $20 per ton for best unmixed seconds.

By the middle of August brimstone had taken quite a jump. Best unmixed seconds sold on spot for $23 per ton, and the best figure that could be obtained at that time for August arrival was $21, and thirds 60c. less per ton. For October shipment the price named was $19, which, as developments in that month proved, must have been made by dealers here "reckoning without their host" (the Societa Anglo-Siciliana) as at all times during October the spot price did not fall below $23 per ton. Before the end of August prices for spot fell to $21, but revived by the close of the month to $23@$24.

During September the price hovered around $23, sometimes a little above and sometimes a trifle below, depending upon the arrivals.

Conditions in October were much like those in September, prices remaining the same, with the market rather stronger. Lack of importations toward the end of the month, due to the increase in freight rates, raised the price of best unmixed seconds to $26 per ton for spot delivery.

Throughout November the market continued strong, though some spot sales of best unmixed seconds were made at a figure several dollars below the closing price of October.

So long as the Sicilian sulphur trust exists—and there appears to be no reason why it should not outlive its five years' compact—the market must continue to maintain its strength, for there are no prospects at this time of a dangerous competitor coming into the field to oppose it. Louisiana deposits have yielded comparatively little; Texas lands have yet to be developed; while the extensive pyrites deposits often mentioned as existing in the South Atlantic seaboard States are beyond reach of railroads as yet.

Imports of sulphur or brimstone in 1896 were 145,318 tons, valued at $2,085,076. This compares with 125,959 tons, valued at $1,593,048, in 1895.

Fertilizer Manufacture and Consumption.—With the exception of 1894 the year 1896 was the most active since 1886. The manufacture of fertilizers in the different sections of the United States amounted, it is estimated, to 1,500,000 tons in 1896, which compares with 1,325,000 tons in 1895, and 1,600,000 tons in 1894. Consumption during 1896 showed an increase, reaching 1,355,000 tons, against 1,128,000 tons in the previous year and 1,575,000 tons in 1894. Commercial manures are used in the Southern States principally for the growing of cotton, while in the other States these fertilizers are used in general farming. In the Western States, where 125,000 tons of commercial manures were used in 1896, there is noted increased activity in the wheat sections.

The Northern and Northeastern States were not as active in their consumption of fertilizers. In the latter case the consumption reached 100,000 tons, the same as in 1895 and a decrease of 20,000 tons from 1894. In the Northern States the consumption was 220,000 tons in 1896, a decrease of 35,000 tons from 1895 and 145,000 tons from 1894.

The Liverpool Chemical Market in 1896.

By Joseph P. Brunner & Co.

The chemical trade during 1896 was not at all satisfactory, the increased competition abroad, coupled with the business depression in the United States, having had a marked effect on the exports of several lines of heavy chemicals. Toward the close, however, there were signs of an improvement, and in several cases prices rallied from the bottom.

Attempts were made to fix up a working arrangement between English and foreign manufacturers on some lines, in order that prices might be placed on somewhat of a remunerative basis, and although nothing definite was reported, it is not unlikely that some such agreement will be arrived at.

The Castner-Kellner Company's works for the production of caustic soda and bleaching powder by electrolysis are now approaching completion, and their products are expected to be on the market during 1897. The development of the concern is awaited with interest by the chemical trade generally.

Soda Ash.—As ammonia soda has so overshadowed ash made by the Leblanc process it is only necessary to review the course of the former product. The agreement made between the leading English soda-ash manufacturers in the autumn of 1895, as a result of which the price of ammonia soda was advanced 10s. a ton, pointed to better times for the producers.

Although prices were well maintained for the home trade and also for most export markets, extremely low figures prevailed in America, owing to the increased production in that quarter, and also to the severe trade depression.

The result was a dull dragging business all through the year, while the exports from Great Britain to the United States decreased materially. In several cases, during the early months of the year, contracts entered into by American buyers were resold on this market and diverted to other channels, while in other cases deliveries on contracts had to be cancelled. An expansion in trade may be anticipated, but the increased production in America is likely to continue a factor for English makers to contend with.

Caustic Soda.—For the first six months a firm tone prevailed, the market then easing off and showing a steady decline up to October. After that, with an improved demand, spot prices hardened, and at the close quotations were only slightly below the figures current at the beginning of the year.

During September and the early part of October manufacturers were competing for 1897 business and very low prices were quoted for forward contracts. This did not last long, however, a sharp reaction taking place and prices recovered 10s. to 12s. 6d. per ton, which advance was maintained later.

Bleaching Powder was in a more or less lifeless condition all through, gradually but steadily declining, closing about 10s. down from the opening.

Chlorate of Potash.—During the early months makers tried to infuse a little backbone into the article by advancing prices, but buyers failed to respond, and beyond one or two spasmodic spurts, the demand was unsatisfactory, buyers having been rather overweighted with contracts in 1895 for delivery over 1896. The highest price touched was 4⅛d. during the early months, dwindling away to 3¾d. in October. There was an improvement later, however, and quotations were run up to 4¼d., while a further advance was talked of.

The English Chemical Industry.

The past year was especially noteworthy in the English chemical industry for the number of new processes introduced. Among these were the Loewig caustic soda process, as adapted by Messrs. Brunner, Mond & Co.; the new chlorate of soda process, as worked by Messrs. Bowman, Thompson & Co.; the nitric acid chlorine process, as worked by Messrs. Sadler & Co.; and the Castner-Kellner electrolytic process for making both bleach and caustic soda.

The sulphuric acid makers have enjoyed good business. It is only a few years since it was hard to dispose of surplus acid at any price, but now there is an actual shortage of supply, and consequently a strong prospect that several new plants will be erected. It has now been fully demonstrated that sulphuric acid of 149° Tw. can be made for about 16s. per ton.

At the annual meeting of the United Alkali Company the report stated that the year ending December 31 showed a net profit of £288,070. The president, in the course of his annual statement, said that the affairs of the company at present looked promising, a great improvement having taken place in 1896.

At the annual meeting of the Castner-Kellner Alkali Company, Limited, it was reported that the installation for 1000 horse power had been practically completed, and in a short time the products would be ready for the market. The buildings which had been erected covered an area of upward of 2¼ acres. The total expenditure on the works had been £70,000, exclusive of the cost of the land, for which £5500 was paid. In addition to this expenditure the directors determined last autumn to incur a further outlay on a second installation of 1000 horse power in consequence of the information supplied to them from one of their allied companies that the Castner-Kellner process was capable of realizing even greater results in the production of caustic potash than of caustic soda. In order to test this statement Mr. Castner was requested to produce caustic potash by means of the plan he used at Oldbury for the production of caustic soda. The results were so satisfactory that the directors felt justified in proceeding at once with the erection of a second installation. The Solvay Werke, of Germany, have been pushing forward their works, which were arranged on exactly the same plan as those of the Castner-Kellner Company, with the utmost speed, and it was expected that they would follow the latter company closely in commencing operations.

In forecasting the future of the English chemical industry the *Chemical Trade Journal* said the following: "Although the year is entered upon with a lower scale of prices than has existed during the past year, manufacturers will have some compensation, as fuel has also receded in price, and contracts have been made covering the whole of 1897 at 10d. to 15d. less than was paid during 1896; at the same time other raw materials remain at nearly the same level.

"On the whole the year does not augur badly, but it will be a trying one for those who have antiquated plant and second-rate supervision. The margin for profit depends entirely upon getting the maximum yield from the raw materials employed, and in reducing the general expenses of the establishment to a figure that can be squeezed no more."

Progress and Profits in the German Chemical Industry in 1895.

By George Lunge.

The materials for the following notes are chiefly taken from the statements made to the Society for the Promotion of the Interests of the German Chemical Industry, by its secretary, Mr. Wenzel, at the last annual meeting. I supplement these from the official custom-house reports, and add a few comments of my own. The whole should be read in connection with my paper in The Mineral Industry for 1895, Vol. IV.

The situation of the German chemical industry during the year 1895 was on the whole satisfactory, but there are a few dark spots on the bright picture of prosperity. The number of factories has since 1894 increased from 5758 to 5947 (3.28%), that of fully employed workmen from 110,348 to 114,581 (3.84%), the amount of wages from 98,500,000 to 103,500,000 marks (4.91%). The annual wages per head in 1893 averaged 879 marks; in 1894, 885 marks; in 1895, 894 marks, showing a satisfactory increase in this direction. The quantity of raw materials used in the chemical industry and of manufactured products carried on the German railway lines increased from 7,063,000 to 7,240,000 tons; the quantity received from or sent to foreign countries increased by 141,044 tons, and thus reached 2,493,522 tons. The analysis of the last figure shows an importation of 1,326,160 tons of raw materials and 239,821 tons of manufactured products, of the aggregate value of 279,800,000 marks, and an exportation of 387,229 tons of raw materials and 540,322 tons of manufactured goods, valued at 339,200,000 marks, leaving a balance of 59,400,000 in favor of importation; that is, 26,300,000 more than the previous year.

The prosperity of the chemical industry can be measured only as far as the joint-stock companies are concerned, for they are obliged to publish their balance sheets. This, of course, does not yield a complete picture of the state of affairs, but it is quite sufficient for forming a general judgment of the state of the industry, since the bulk of most chemical industries in Germany is in the hands of joint-stock companies, and since we can in any case compare their dividends with those which they have earned in former years.

The 95 joint-stock chemical companies in Germany, with a paid-up capital of 248,000,000 marks, in 1895 paid an average dividend of 12.71%, against 13.44% in 1894, which had been the highest on record. The division of profits among the various companies was also less satisfactory than in the preceding year, for 22.1% of the companies paid no dividend at all (against 14.29% in 1894), and 20% of the companies paid only up to 5% (against 13.19% in 1894). We shall now consider the different classes of chemical industries seriatim.

The heavy chemicals (acids and alkalies) paid an average dividend of 8.64%, against 9.23% in 1894. This decrease was almost entirely caused by a deficiency of profits in the different branches of the potash industry. The exportation both of kainit and of chloride of potassium (muriate of potash) was less than in 1894. Chlorate of potash suffered severely under the competition of the article made by means of electrolysis in countries with cheap water power (Switzerland and Sweden). Saltpeter is consumed less and less, since hardly any black powder is now employed in Germany for military purposes, and much less than formerly for

mining operations. The sales of carbonate of potash are greatly diminished by the large quantity of electrolytically produced caustic-potash liquor. Prussiate of potash was influenced by the fact that much cyanide of potassium is now made directly from ammonia.

The situation of the soda market is in a very different and much more auspicious state. The factories were fully employed all the year round, and the excess of exportations over importations was 31,000 tons of soda ash and 4625 tons of caustic. This amounts to a total equivalent of about 36,700 tons, calculated as 100% sodium carbonate, being a slight decrease against 1894, which decrease seems to have been fully compensated by an increase of the home consumption. Sulphuric acid was in greater demand than ever, and sometimes even scarce in the market, partly owing to the increasing consumption for sulphate of ammonia, made as a by-product from coke ovens. The exportation of sulphuric acid rose by 2800 tons, in spite of the fact that Russia and Japan are now manufacturing large quantities of that acid from their own raw materials. Hydrochloric acid was also exported in larger quantities, the price remaining stationary.

A great change has taken place with bleaching powder. Formerly Germany was practically dependent upon Great Britain for this article, as in the last-named country hydrochloric acid had very little value, while the German factories could dispose of that acid better than for manufacturing bleaching powder. As the manufacture of heavy chemicals increased more and more in Germany, that country gradually emancipated itself from foreign supplies of chlorine products; since 1892 the exportation has exceeded the importation, and in 1894 there were already 3558 tons of bleaching powder exported, against 1380 imported. In 1895 we find the respective figures of 3747 against 1228; and in 1896 the balance has still further turned in favor of an excess of exportation, so that it will probably reach more than 6000 tons. This startling change is entirely due to the electrolytic manufacture of chlorine, in which Germany has been hitherto far ahead of all other countries. This fact is very suggestive indeed for the United States, with their immense stores of water power and coal, and a corresponding chance for electrolytic processes.

The joint-stock companies manufacturing pharmaceutical, photographic, scientific, and technico-chemical preparations in 1895 netted only 9.13% dividends, against 11.23% in 1894. This deficiency was caused by a decline in prices, due to increased competition. We should not forget, though, that in this specialty there are in Germany many private firms of the highest standing (Merk, Kahlbaum, Trommsdorf, De Haën, Schuchardt, and others), which causes the dividends of the joint-stock companies to be a much less sure index of the general prosperity in this kind of industry than in the other chemical branches. A most remarkable addition to the pharmaceutical preparations is the constantly increasing production of biologically produced therapeutic agents (serums of various kinds), for which Germany has as yet almost a monopoly.

For the industry of alcoholic preparations the year 1895 was decidedly a prosperous one. Some very inconvenient fiscal restrictions have been removed; the imports have increased by 10%, the exports by 100%. But the constant downward tendency of prices has cut down the profits. Similar conditions are found in the trade of acetic, tartaric, and oxalic acids. Better conditions were ruling in the

stearine manufacture, with its by-products, olein and glycerine, the latter having doubled its price. Germany does not export any stearine candles, as she cannot compete with countries more favorably situated with regard to raw materials and wages, but the home consumption has considerably increased. For soaps and perfumeries the year was also a good one ; the exports increased much more than the imports.

As usual, the industry of coal-tar dyes stands out brilliantly among chemical industries. The dividends increased from 23.13% in 1894 to 23.59% in 1895; but we must not overlook that the factories derive much of their profits, not from dyes, but from synthetic medical preparations. The value of coal-tar colors, including alizarin, exported in 1895 from Germany was nearly 75,000,000 marks, against an importation of 3,600,000 marks.

The immediate products of the distillation of coal-tar benzol, naphthalene, pyridine bases, anthracene, pitch, etc., were quite remunerative, in spite of the large addition to the supply of the raw material as a by-product from coke ovens.

Explosives netted an average dividend of 18.40% in 1895 (17.37% in 1894). The Transvaal was the principal foreign customer; elsewhere the American competition is more and more felt. Some smaller nations still consume considerable quantities of black powder. Lucifer matches averaged 7.30% dividend, an increase of 1.24% against 1894, which must have proceeded from special causes, as otherwise this industry has to carry on a very hard struggle, principally owing to the Japanese competition in Eastern countries.

Almost alone among the chemical industries, that of artificial fertilizers was in a decidedly bad situation, the dividends having shrunk from 6.69% in 1894 to 3.75% in 1895, and this in spite of an increased consumption, all in consequence of the extraordinary decline of prices, brought about by excessive home and foreign competition.

Concerning the profits made by the German chemical industry in the year 1896 the writer is not yet in a position to make any statement. It seems as if this year had been running almost exactly in the same grooves as its predecessor. I have not heard any complaints about bad trade; no commercial treaties have expired, nor have any important new treaties been concluded, except that with Japan. No stirring inventions have ripened in this present year. Altogether it seems to me as if the former balance of profits could not have been much changed either one way or another. It is, however, impossible to state this with certainty, as no figures on this subject will be available for some time to come.

An important matter, which presents special interest to Americans, remains in suspense. This is the question whether the duties on crude and refined petroleum are to be regulated in such manner that it becomes possible to refine crude petroleum in Germany, which is not the case at present. This measure is vividly advocated by the manufacturers of heavy chemicals, who would have to furnish the acids and alkalies for the refining operations; but it is strongly opposed by the manufacturers of paraffine oil, etc., from brown-coal tar, a very important industry flourishing especially in Prussian Saxony. The chemical manufacturers try to reassure the paraffine-oil men by telling them to take the refining of petroleum into their own hands, simply substituting petroleum for their present raw material. But the paraffine-oil men refuse to be comforted in this way; they say

that they would not merely have to change almost the whole of their plant, but the refining of petroleum would be most certainly carried out at the seaports, not, as at present, in the center of Germany, where they are located; and it can hardly be denied that this is correct. Up to the present it looks as if the old state of affairs was not to be disturbed. The above-mentioned proposal was to a great extent prompted by a desire to escape the monopoly of the Standard Oil Company; but the discussion carried on in this matter has abundantly proved that neither the Russian nor the Galician petroleum could be expected to keep out the American product. If at any time the projected reform of the import duties were carried out, it would simply mean that instead of refined kerosene, a corresponding quantity of crude petroleum would be imported, and for that matter both the Russian and the Austrian product are foreign products as well as the American. Nobody certainly objects to the American product as such, but merely to the prospect of being at the mercy of that gigantic trust.

The question of raising the flash-point of petroleum, which has caused so much feeling against the American petroleum in England, and is now forming the subject of a parliamentary inquiry, has not awakened in Germany more than a discussion in technical journals, and is not at all likely to be touched by legislation in that country.

The Chemical Industry of Italy.

The consumption of chemicals in Italy is increasing with the requirements for manufacturing purposes, and at present about one-half of these requirements is supplied by Italian manufacturers. There are now in the kingdom nearly 300 works, employing nearly 5000 persons, and making an annual output of about $6,000,000. The most important branch of the industry, as regards value of output, is that of superphosphates and other fertilizers, of which upwards of 86,000 metric tons are produced annually. Next in importance are the manufactures of sulphuric acid and various sulphates. The exports of chemicals from Italy are insignificant.

The Chemical Industry of Russia.

Russia now supplies itself with nitric and hydrochloric acids, of which the importation fell from 19,000 tons in 1885 to 313 tons in 1894. At the fifteenth exhibition of Russian industries at Nizni-Novgorod last year 15 sulphuric acid works were represented. All the acid now used in Russia is of domestic production. Equally rapid progress has been made in the alkali industry, the more so since there were no established interests with antiquated methods to be got rid of and modern processes could be installed at once. The firm Liubimov-Solvay has two works in Perm and Ekaterinoslav which in 1894 produced 45,406 tons of soda crystals out of the 64,450 consumed in the empire. Soda-ash and caustic soda works have also been established, the latter chiefly in the neighborhood of Baku, where the product is used in the refining of petroleum derivatives and

where the abundance of raw material enables the maker to turn it out cheaply by the Le Blanc process. The manufacture of chromates has been undertaken on an extensive scale, while many minor industries, such as the preparation of salts of lead, zinc, copper, tin, and strontium, have been inaugurated. Two refineries of boracic acid have been built, and benzine-distilling has become a huge business.

Imported sulphur still serves as the chief material for many of the smaller acid-works, the Russian sulphur mines meeting with great difficulties. Pyrites is produced on the Asta River in the Government of Novgorod, and in Tula some is obtained as a by-product from the coal mines. The Caucasian pyrites, which was formerly used by Siemens Bros. as copper ore, have been employed in acid-making at Baku since 1894. In the Urals large amounts of pyrites are mined for the acid-works, the chief consumers being Uschkow & Co. These ores are mixed so as to average 43% to 45% S; the cinder contains 3% to 3.5% Cu. The Ural pyrites is free from arsenic and burns well, the cinder assaying only 3% to 5% sulphur. The Kolatinsk pyrites, however, contains selenium and tellurium, and makes an acid of red color. The yearly consumption of domestic pyrites for acid-making is about 24,570 metric tons, in addition to which about 32,760 metric tons are imported. The price of Spanish pyrites, with S = 45% to 50%, in St. Petersburg in 1896 was $4.88 per metric ton, and at Baku $6.10 per metric ton; Caucasian pyrites sold for 12 cents per unit of sulphur, and $1.10 per unit of copper. The ore of the Urals sells at $3.41 to $3.80 per metric ton at the mine; freight brings the cost up to $7.80 to $8.68 per metric ton at Uschkow's works at Jelabugga.

The largest acid-works in Russia are those of Uschkow and Tentelew, and at the Schastaschian Powder-works. The Uschkow works have 24,100 cu. m. chamber-space, and use annually 16,380 metric tons of Ural pyrites, which is the only raw material employed. The residues are smelted for copper, which is marketed chiefly in the form of blue vitriol and oxide. The annual product is about 245 metric tons of copper and about 164 metric tons of vitriol. The Tentelew works, which have 25,500 c. b. m. chamber-space, use only Spanish pyrites, and produce about 24,570 metric tons of chamber-acid per annum.

The larger Russian acid-works are fitted with all modern requirements. On account of the considerable difference between summer and winter temperatures it is necessary to allow a chamber-space of 1.50 to 1.75 c. b. m. per kgm. of sulphur to secure a good yield of acid with a small consumption of niter. For roasting the fine ore Maletra and Schaffner kilns are used.

The total production of monohydrated H_2SO_4 in Russia is about 98,280 metric tons, and only about 491 metric tons are imported. The cost of production in the east of Russia is estimated by P. P. Fedotijew at $5.37 per metric ton of 52° B., while the selling price at Moscow is $13.18 to $14.64 per metric ton. In order to prevent overproduction the manufacturers formed a combination in 1890, but it lasted only until 1892, when prices dropped heavily. Lately, however, in consequence of the great advance in the petroleum industry the price has risen again. The cheapening of this product, which is so indispensable for the development of the chemical industry, depends chiefly upon the reduction of freight rates, and secondly upon an increase in the production of pyrites.

Progress in Electro-Chemistry and Electro-Metallurgy in 1896.

By W. Borchers.

I. Generating electricity by chemical processes.

"Electricity directly from coal" did not lose its fascinating power as a watchword in electro-chemical discussions from the first of January to the end of December, 1896, though in some quarters it seemed to exert the influence of a war-whoop.

The first welcome target for the furious critic who had just slaughtered the Borchers cell—which by the way is still alive—was a harmless preliminary note on some observations by Dr. A. Coehn. He did not pretend at all to have invented "the" coal battery. Indeed, the conclusions from his first experiments simply amounted to the following not very surprising facts:

1. It is possible to dissolve carbon electrolytically.
2. From such a solution carbon may be precipitated as kation.
3. It is possible to construct an element, the soluble electrode of which consists of carbon.

The word "coal" seems to have been mistaken for carbon by eager critics. Coal in any of its various forms is a recognized chemical compound, or a mixture of compounds, of carbon with hydrogen, oxygen, and small quantities of other elements. Taking into account the constituents just named, Dr. Coehn has not said anything very startling. It was known years ago that carbonaceous electrodes in electrolytic cells, no matter whether the latter were generating or consuming electric energy, slowly but surely dissolved. Yet we know just as well that these by-processes in our electrolytic cells are very far from having been explained satisfactorily, though prominent chemists like Matteuci, Von Babo, Brandt, Bartoli, Papasogli, and Millot have been investigating the processes and products of such solutions during the last 40 years. If Dr. Coehn now considers the present time and the present facilities of electrical and chemical equipments favorable enough to determine exactly the conditions most suitable for utilizing a process that has so far been considered an unavoidable nuisance, there is no reason to discourage him by intolerant criticism, especially as none of his critics has been able to discover a fault in his preliminary experiments, which I briefly reported in The Mineral Industry, Vol. IV., p. 802.

Tommasi claims to have done 12 years ago the same thing which Dr. Coehn pretends to have achieved to-day, yet queerly enough his battery, even as he describes it under his present impression, differs entirely from that of Coehn.

Concerning a very interesting method proposed by C. J. Reed for the transformation of the chemical energy of coal into electrical energy, I need not enter into details, as the text of the original publication on this subject is easily accessible by referring to the *Journal of the Franklin Institute*, to the *Electrical World*, to the *Electrician*, to *Industries and Iron*, and to the *Zeitschrift für Elektrochemie*, the latter giving a German translation of the paper. To summarize Mr. Reed's idea, he proposes to burn carbon first in an atmosphere of sulphur vapor to produce carbon disulphide, then to decompose this product by means of steam into carbon dioxide and sulphureted hydrogen, using the latter as reducing agent in a gas battery, the oxydizing agent of which is to consist of

sulphurous anhydride. This battery, giving an electromotive force of 0.36 volt, returned about 35% of the energy of the chemicals expended.

Mr. Reed offered the following rules for the reactions to take place in a direct or indirect coal-gas battery:

"Such a set of reactions would be represented by the following equations, in which T represents one combining equivalent of a transforming substance, C an equivalent of carbon, and O an equivalent of atmospheric oxygen:

"1. $T+O=TO$.

"2. $C+T=CT$.

"3. $2\,TO+CT=CO_2+3\,T$.

"In equation 1, TO represents the secondary reagent, produced by the action of T upon atmospheric oxygen, and CT, in equation 2, represents a secondary agent, produced by the action of T upon carbon. Equation 3 represents the reaction that takes place in the discharge of the battery, CO_2 and the original transforming substance T being the products of discharge."

In 1894 the same author wrote the following on the Borchers gas battery: "The writer (Reed) tried about a year ago the same experiments, described by Borchers, using carbon tubes for both electrodes instead of copper. Over 100 different solutions, including cuprous chloride, were tried as electrolytes. I was unable to find that any two gases would combine spontaneously under these conditions unless they were capable of combining spontaneously at low temperatures when brought in contact with each other in the gaseous state. Electrolytic gases formed at the electrodes by an electric current will recombine. But I was unable to get positive evidence of combination of even oxygen and hydrogen obtained from other sources and free from ozone and other impurities."

Let me now review the principle of the Borchers gas battery, choosing the general way of explanation as in the first-cited words of Mr. Reed:

T may represent one equivalent of the combining substance "cuprous chloride," CO an equivalent of carbon monoxide, and O an equivalent of atmospheric oxygen. According to my first explanations, the cuprous chloride was oxidized by atmospheric oxygen to cupric chloride, which means in the general equation used by Mr. Reed:

1. $T+O=TO$.

At the same time the carbon monoxide at the other pole of the battery was to combine with cuprous chloride:

2. $CO+T=COT$.

And finally the reaction:

3. $TO+COT=CO_2+2\,T$.

The formation of carbon dioxide and the recovery of the original transforming substance is here also the final result.

Everybody will admit that my equations mean essentially the same as those of Mr. Reed, who in 1896 considers the manner of working rejected by him as hopeless in 1894 as the only direction in which we are likely to attain any great improvement over thermo-dynamic methods.

In the meanwhile the faults of the apparatus first proposed by me have been discovered to be structural and not involving the principle, which has not been abandoned. Further publications on this subject will not be made until the

experiments have exhaustively settled the question of the practical applicability of the process.

On the Jacques battery, which follows next, I need not report from this shore; American literature on this subject will be abundant enough.* Concerning C. J. Reed's interesting so-called thermotropic process, I may also refer to American periodicals, especially to the *Journal of the Franklin Institute.*

The Bucherer battery, which has lately been patented, contains molten alkaline carbonates as electrolytes, into which carbon monoxide gas is made to diffuse through iron or nickel tubes, while oxygen is supplied through platinum tubes. The inventor has as yet not given any data that would permit a judgment on the efficiency of this certainly interesting apparatus and manner of working.

II. STORING ELECTRICITY BY CHEMICAL PROCESSES.

The greater part of the second day of the annual meeting of the German Electro-Chemical Society was spent in discussing the theory of the lead accumulator and the value of the so called "masse-plates," electrodes, which consist chiefly of active material, stuck together by some organic substance such as glycerine.

As to the theory, no word was said on the Darrieus persulphuric acid theory. All those who participated in this discussion, as for instance Professor Elbs of Giessen, Mr. Liebenow of the Akkumulatorenfabrik Hagen, Dr. Sieg of Cologne, Director Pollak of Frankfort, Dr. Correns of Berlin (most of them representatives of large accumulator works), only referred to possibilities, standing on the ground of the theory of free ions.

The "masse-plates" were almost unanimously condemned. It was stated that the organic cements intended to hold the lead or peroxide particles together were almost completely destroyed during the process of formation, so that the advantages claimed for these electrodes seem to be most doubtful indeed.

The largest accumulator factory of Germany, the works at Hagen, has adopted a modified Planté process of forming the electrodes for such cells, which have to be charged in a short time by means of currents of high density. In this way they have succeeded in charging street-car cells while running on the line through suburbs and open roads where overhead conductors are admitted, while the cars use this charge in streets where overhead lines are forbidden. This system is working to the greatest satisfaction in Hanover, one of the handsomest cities of North Germany.

The alkaline cupric oxide-zinc accumulator has apparently proved a complete failure. It has been abandoned wherever it has been tried.

III. ELECTRIC HEATING.

At present the chemistry of high temperatures is enjoying an attention which means more than a transient flirtation on the part of practical electro-chemists. The electric furnace enables us to turn out products which a year or two ago could not be found in the price lists of chemical works. It is especially the new and rapidly growing industry of carbides which is pushing the development of

*See paper by T. W. Sprague in this volume.

the electric furnace in a most astonishing way. For this reason it may be worth while to review the past and the present of this kind of furnaces somewhat closer in order to see what has been done and what may still be expected.

The Different Modes of Electric Heating.—There is a marked difference between one kind of heating, consisting in sending currents of supernormal density through metallic or electrolytic conductors, and the other kind by means of the electric arc. In the following text I may be permitted to call the first mode "resistance heating," the second "arc heating." The conditions of working in these two processes are directly contrary to one another. The distance of the terminals of the circuit being equal, the resistance heating requires currents of low tension but high intensity, while the arc heating requires currents of high tension and low intensity.

Resistance Heating.—We may subdivide this manner of heating into two classes·

(*a*) *The substance to be heated is itself the resistance of the circuit.* Apparently this kind of heating was tried for metallurgical purposes first of all. Certainly an experiment described in the *London Philosophical Transactions* of 1815, Vol. CV., p. 570, by Pepys, may be called electric cementation. Pepys bent a wire of pure soft iron, so as to form an angle in the middle, in which part he divided it longitudinally by a fine saw. In the opening so formed he placed diamond powder, securing it in its place by two finer wires, laid above and below it, and kept from shifting by another small wire bound firmly and closely around them. All the wires were pure soft iron, and the part containing the diamond powder was enveloped by thin leaves of talc. Thus arranged the apparatus was placed in the electric circuit, where it soon became red-hot, and was kept so for six minutes. On opening the cooled wire it was found that the whole of the diamond had disappeared, and that all that part which had been in contact with it was converted into steel. A portion of the latter being heated red and plunged into water, became so hard as to resist the file and to scratch glass. There is no doubt consequently that electric cementation was performed more than 80 years ago. (See. Fig. 1.)

It was a very long time, however, before anything was again heard of furnaces of this description. In 1884 Messrs. Cowles Brothers, of Cleveland, applied for patents on furnaces in which a mixture of refractory oxides and conducting carbonaceous material like coke was placed as resistance between two carbon terminals of a circuit of great intensity.

The furnace, as proposed in the first of the Cowles patents, and as shown in Fig. 2, was to reduce zinc oxide. For this purpose, however, no advantages could be obtained over the old method of distillation.

Another construction has met with better success, viz., the furnace for making aluminum alloys, which is shown in Fig. 3, taken from one of the patent specifications. Fig. 4 shows a longitudinal vertical section through the whole furnace as applied in practice, at the moment of the starting. Just then the two bundles of carbon rods, which are introduced through the side-walls of the furnace, are moved toward one another until two rods, projecting from the bundles, touch. The two projecting rods will be heated first, but as the conductivity of the charge in the furnace will soon increase, the carbons will have to be withdrawn far

enough to keep up a sufficient resistance for the current, so that very soon after starting the charge of the furnace will be the resistance, in which the electrical energy is directly transformed into heat. And the production of heat was the only duty the electric current was condemned to in the Cowles furnace, the remaining work being done by the chemical energy of the electrically heated carbon.

Héroult was the first experimenter to make the electric current do two different kinds of work at the same time by using an electrolytic conductor. Molten oxide of aluminum was heated to the fusing point and electrolyzed by one current. At first he did not succeed in producing pure aluminum, because the cathode of his smelting pot consisted of a carbon lining in an iron box, and aluminum has a tendency to combine with carbon, while a metal thus contaminated by carbon or aluminum carbide could never again be purified. For this reason Messrs. Cowles alloyed the nascent aluminum with copper or iron, and for the same reason Mr. Héroult resorted to a molten copper cathode in the bottom of his crucible, to prevent the formation of the annoying carbides. When a few years after the start the Neuhausen aluminum works commenced to turn out pure metal, they certainly must have done away with the conducting carbon lining, though nothing has been stated publicly as to the present construction of the furnaces in use at those works. (Fig. 5.)

Borchers' experiments in this line have led to the construction of a crucible made of sheet iron, with a fireclay bottom, through which a hollow metal cathode is introduced. The latter may be cooled so as to prevent the cathode metal from being dissolved in the molten aluminum collecting at the bottom. The interior walls of this furnace are to be lined with aluminum oxide, fluoride, or other suitable compound that is kept cool enough to prevent its melting. (Fig. 6.)

Urbanitzki's latest furnace also contains a cooled metallic bottom, serving at the same time as cathode. The anodes, carbon plates, are arranged in a circle around a central feeding tube. This furnace may just as well be used for arc heating. (Fig. 7.)

In Taussig's furnace the metal or ore to be melted was to fill an U-shaped hearth at both ends of which the terminals of a circuit of high intensity were introduced. This arrangement was to replace the open hearth in the iron foundry and in Martin steel works. Being inclosed, together with a collecting hearth for accumulating molten metal for large castings, inside of an iron box to which the molds could be directly attached, the melting could be carried out in any desirable atmosphere: oxidizing, reducing, or neutral. (Figs. 8 and 9.)

Before going into details of the construction and management of this kind of furnace it may be better to look at the most characteristic examples of another class of furnaces using the resistance heating method.

(*b*) *The substance to be heated is directly or indirectly in contact with a heated resistance of an electric circuit.* The Pepys appliance of 1815, as described under (*a*), may just as well be counted in this class of furnaces, if we look at it from a different point of view, viz., taking the diamond powder as the object of an investigation regarding its behavior while heated electrically by means of a current going through an iron resistance.

In 1849 a second experimental furnace of this kind was constructed by Depretz, who sent a current through a tube of carbon containing the substance to be heated. (Fig. 10.)



FIG. 1.



FIG. 2.

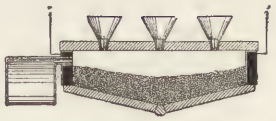


FIG. 3.

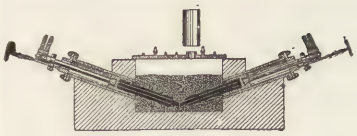


FIG. 4.

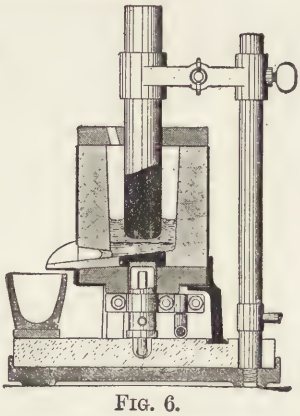


FIG. 6.

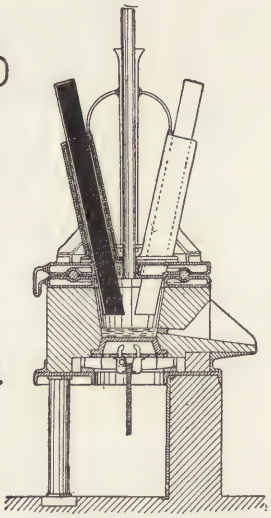


FIG. 7.

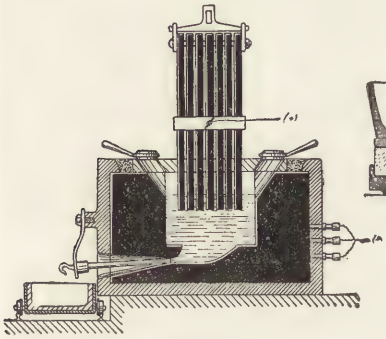


FIG. 5.

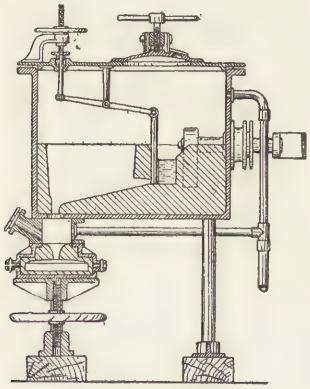


FIG. 8.

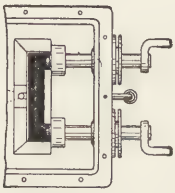


FIG. 9.



FIG. 10.

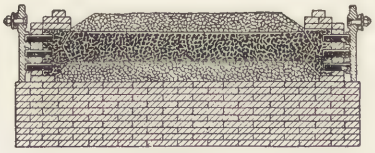


FIG. 11.

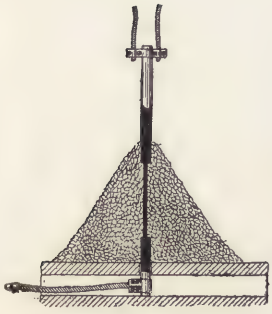


FIG. 12.

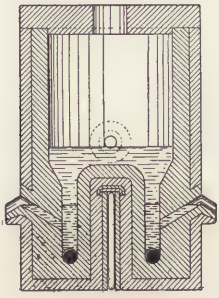


FIG. 13.



FIG. 14.

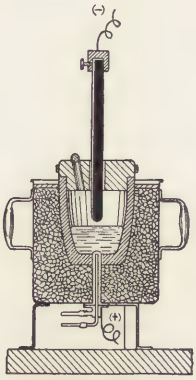


FIG. 15.

At the start of the latest form of the Cowles furnace, as shown in Fig. 4, the charge, which later in the smelting process acted itself as heating resistance, was heated by means of the carbons projecting from the rest.

An experimental furnace, described in Vol. IV. of THE MINERAL INDUSTRY, p. 812, in which the principle suggested by Pepys and Depretz was applied from the start to the end of the smelting process, has become familiar to electrometallurgists, since the author of this article published the results of his experiments on the reducibility of all oxides previously considered unreducible by carbon.* This very publication not only stated that all oxides of metals could thus be reduced by electrically heated carbon, but that other refractory oxides, like the dioxide of silicium, *i.e.*, quartz, would not resist. It was furthermore stated that some of the reduced products could not be prevented from combining with carbon. Finally another important feature for smelting processes of this kind was mentioned, namely the addition of some easily fusible or easily reducible substance to aid as a flux or as a reducing agent. That these observations were right was clearly proved by their general application in the Acheson carborundum process and furnace (Fig. 11), in some of the recent carbide furnaces, of which for its remarkable simplicity that of King and Wyatt may be mentioned (Fig. 12), and in several other less important constructions.

A liquid resistance has been used by De Laval. In his well-known furnace a molten oxide of the metal to be refined supplies the resistance between the terminals, which when the process is going on chiefly consist of the metal to be recovered collecting in the lower ends of the A-shaped smelting hearth. (Fig. 13.)

From these specimens, selected from the large number of furnaces of this principle, it will be seen that with the exception of a few like those of Héroult and De Laval using liquid resistances, carbon in various forms is the material par excellence. In the first place, therefore, it is important to know what current density has to be applied to heat a given amount of carbonaceous material to a desired temperature. The composition of such materials, however, its physical properties, and hence the conducting qualities of the same, will naturally show a great variety. For this reason we will have to be content with the facts known about the purest qualities of electrically-conducting artificial carbon bodies, and draw inferences as to the inferior, but nevertheless useful, kinds like coke, etc.

The following table contains some results obtained in heating rods of pure carbon by currents of different density:

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
	Diameter.	Length.	Weight Per 10 Millimeters Length.	Potent Difference Per 10 Millimeters Length.	Current Density Per Square Millimeter Cross Section.	Loss of Energy Per Square Millimeter Section and 10 Millimeters Length.	Per Gram of Technically Pure Carbon in 10 Millimeters Length.	
							Intensity of Current.	Loss of Energy.
	Mm.	Mm.	Grams.	Volts.	Ampères.	Watts.	Ampères.	Watts.
(*a*)	4	20	0.21	2.50	10.00	25.000	595	1,487
(*b*)	4	30	0.21	2.30	8.00	18.400	476	1,095
(*c*)	4	50	0.21	2.20	5.00	11.000	297	655
(*d*)	4	100	0.21	2.00	3.00	6.000	178	357
(*e*)	6	60	0.52	1.80	2.14	3.850	115	207
(*f*)	6	200	0.52	0.85	1.43	1.210	77	65
(*g*)	10	200	1.26	0.70	0.64	0.448	40	28
(*h*)	10	300	1.26	0.47	0.57	0.268	34	16

* Borchers: *Electrometallurgy*, First Edition, 1891.

In speaking of current density it has been a custom to refer the intensity of the current employed to the area of cross section of a conductor or to the surface of an electrode. But, as will have been noted from columns VII. and VIII. of the preceeding table, I have been referring the intensity of the current to the unit of weight of the resistance to be heated contained in a unit of length. No doubt, for the practice of electrothermic processes this method will prove the more reliable in comparing two processes or of choosing the correct density for a desired temperature and for a given intensity of current. No two kinds of coke, for instance, will be found alike in their chemical composition and in their physical properties; and even if these qualities could be kept in a certain kind of carbonaceous material approximately regular, two workingmen of different strength would certainly stamp different weights into a given space, supposing that the resistance had to be built into the charge of the furnace, say, for instance, according to the rules for charging the Acheson furnace. If the manager of an electric smelting works knows the weight of carbon contained as resistance in the charge of his furnace, he will more surely hit the correct intensity of current to send through the resistance than when he merely knows the cross section of the latter.

At current densities of about 900 ampères per gram of carbon in a resisttance of 10 mm. length, carbon commences to volatilize. At this density, therefore, a temperature approaching 3500°C. must have been reached. The temperature obtained by using a density of about 600 ampères per gram carbon in 10 mm. length has proved to be sufficient to decompose by the carbon thus heated the most refractory oxide hitherto known. A current density of 300 ampères per gram carbon in 10 mm. length will produce the temperature necessary to reduce lime to calcium carbide. Current densities of about 30 ampères per gram of carbon in 10 mm. length will only heat the carbon to a barely visible red.

As will be seen from the preceding table, these results have been obtained by means of comparatively small carbon bodies, and there is no doubt that a furnace built on a large scale would show a very much better gain of heat.

Electrolytes which have to be kept fused by the heat developed by very dense currents, as for instance in the Héroult process, require a much lower density than when carbon or metallic conductors are used. In the Héroult apparatus, for instance, a square meter of cross section of the electrolyte will receive about 25,000 to 30,000 ampères.

A most decided advantage of electric heating over the old methods for producing high temperatures consists in the possibility of generating the heat at the spot where it is needed, and so fast that disturbing reactions, which have a chance to occur during a gradual warming process, are entirely avoided. This point is furthermore of the greatest importance in the choice of the building material of the furnace. Thus if I were to need a furnace for the decomposition of a chemical compound, which when fused will corrode any refractory material we have, I may build an electric furnace out of some metal like iron, line it with the very compound to decompose and keep the metal walls cool enough to prevent the lining from melting, while in the interior of such a furnace a temperature may be kept up which will evaporate even carbon, the most refractory substance known. And we may safely say that with the electric furnace all restric-

tions limiting the construction of high-temperature furnaces are entirely done away with. Electric heating has cleared away the last obstacle against manufacturing aluminum on a large scale; it has given us the means of making products which theretofore were known as chemical curiosities; and I dare say that there is no task the chemistry of high temperatures may devise which to-day we are unable to solve.

Arc Heating.—Though the light spending qualities of the voltaic arc were recognized soon after the discovery of the Volta battery, and especially when Davy commenced his classical researches in 1808, after a battery of 2000 elements had been put at his disposal, the utilization of the high temperatures produced in an arc between carbon electrodes was not thought of, even for experimental purposes, until the middle of this century.

As in the previous remarks on appliances for resistance heating, we distinguish two different classes of furnaces:

(*a*) *The substance to be heated constituting one or both electrodes of the arc.* To investigate the behavior of carbon at the temperature of the electric arc, Depretz made an arc between a small retort of sugar carbon and a pointed carbon rod introduced into the retort, thus showing that sugar carbon was transformed into graphite. The result of this experiment and consequently the principle of the first arc furnace were published in 1849 in Vol. XXIX. of *Comptes Rendus.* Of course at the age of the galvanic battery there was no chance to employ this precious source of heat for manufacturing purposes, and hence it is not surprising that the year 1878 brought the first electric arc furnace fit for work on a large scale. And in justice to the inventor, C. W. Siemens, it must be admitted that the models of that time embrace about all important points of construction modern inventors are so eager to claim as ideas of their own.

Referring to Fig. 14, and quoting from the specifications of the English patent No. 4208 of 1878 and No. 2110 of 1879, the fused metal in the bottom of an iron crucible lined with non-conducting material formed one of the terminals of a circuit to which the metal was connected by means of a screw-pin projecting through the bottom of the crucible into the molten mass. The other terminal, which was cooled by the passage of water through it, to prevent its burning away, was suspended through a hole in the crucible cover in such a manner that the distance between the terminals could be regulated.

Another construction contained the cooled electrode in the bottom of the crucible, while the upper pole consisted of carbon; see Fig. 15.

A most important point for the successful operation of these crucibles seems to have been overlooked by all those who have reported on the Siemens apparatus, though it was clearly stated in the specifications above referred to: the application of a wire solenoid surrounding the crucible in order to prevent the arc from jumping to the walls of the crucible and to deflect the arc upon the metal to be fused.

The carbide industry has developed this kind of furnace to a very high degree of perfection, without giving up the simplicity of smelting arrangement. As the furnaces used by the Willson Aluminum Company for making calcium carbide, and those devised by Allen and Thwaite, by Tenner, by Bullier, and by Clarke, concerning all of which much has been written in technical journals, are already well

known to American readers, I will only mention one other. This is a furnace of Rathenau, shown in Fig. 16, which is intended to avoid the whirling up of the charge by ascending gases from the smelting zone, to protect the upper carbon electrode by the charge packed around it, and to gradually heat the charge which is slowly descending in the funnel surrounding the upper electrode.

A furnace several times patented by Heibling for smelting chrome and similar ores is impossible to work on the large scale for which it has been devised. (Fig. 17.)

(*b*) *Furnaces in which the substance to be heated is introduced into a space heated by an independent arc.* At a time when the possibility of an application



FIG. 16.

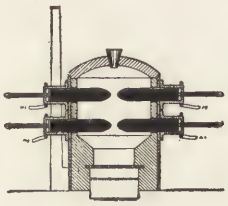


FIG. 17.

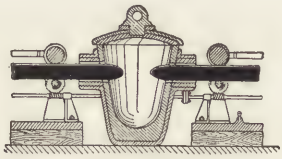


FIG. 18.

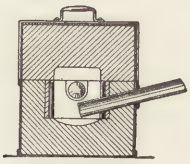


FIG. 22.

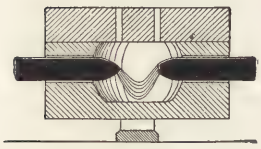


FIG. 19.

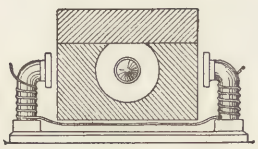


FIG. 20.

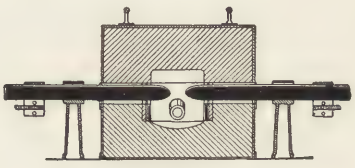


FIG. 21.

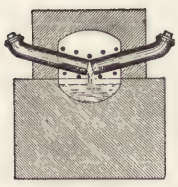


FIG. 23.

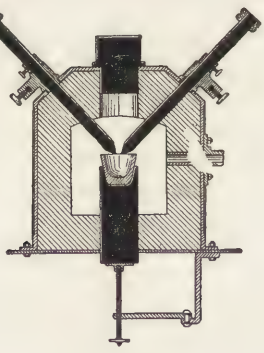


FIG. 24.

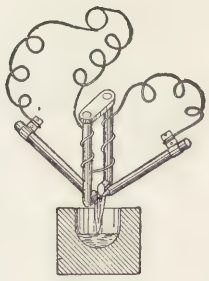


FIG. 25.

of the invention could certainly not be expected, a furnace was constructed by Pichou, who most likely applied for a patent on it in England through a person named Johnson, probably a patent agent. According to the specification, No. 700 of 1853, the ore or metal to be melted was to be dropped between a series of electric arcs, produced inside a furnace on the hearth of which the molten masses were to collect to be kept warm there until ready for use.

Now follows a Siemens' crucible, which differs from the two furnaces mentioned under (*a*), inasmuch as it was intended for smelting non-conducting sub-

stances, and for this reason required an independent arc. Fig. 18 does not need any further explanation; but it may be well to sum up the rules which C. W. Siemens set down, especially for the construction of arc furnaces and their operation. We see that almost 20 years ago he had constructed furnaces in which either the substance to be melted was forming one pole of the arc, or it was heated by an independent arc. He applied cooled electrodes, which have proved to be very useful in many cases. Finally he influenced the direction of the arc by means of electro-magnets.

After these considerations we must certainly admit that most of the modern furnaces ascribed to well-known names are built strictly on the ground of the principles settled by Siemens 20 years ago. Though each of the following furnaces may have its advantages for a special kind of work, a glance at the sketches will show us that they are mere modifications of the old Siemens form, which is still doing very well for a great variety of smelting operations.

Among the furnaces of this kind the first to be noted was that shown in Figs. 19 and 20, patented by Rogerson, Statter, and Stevenson in 1887. The arc was deflected by a horseshoe electro-magnet. The walls of the hearth were made of magnesia, lime, or other fireproof material.

In 1893–94 Moissan and Chaplet constructed similar furnaces, into the smelting hearths of which the charges had to be introduced through carbon tubes. The original Moissan furnace is illustrated in Figs. 21 and 22, the Chaplet furnace in Fig. 23.

If it is desirable to observe what is going on in the smelting hearth, the furnace of Ducretet and Lejeune is a very commendable appliance. (Fig. 25.)

For experiments on a very small scale, say for assaying purposes, where the blowpipe does not reach the desired temperature, the Zerener arc blower, in which all parts are arranged in such a manner as to be used like a soldering iron, will be found as convenient a tool for laboratory and for soldering use as the blowpipe. The essential parts of this apparatus are shown in Fig. 25.

The formation and maintenance of the electric arc not only depend upon the electromotive force and the intensity of the current but to a very large extent on the nature of the electrodes. Certainly no arguments will be required to confirm the statement that carbon ranks foremost as a material for the manufacture of arc terminals. Yet in electro-metallurgical practice we will have to reconcile ourselves to the necessity of using a less refractory material at one terminal, at least in some cases, and it ought not to be forgotten that there are some metals which do not allow the formation of an arc at all, for instance zinc, bismuth, and antimony; while there are others evaporating at too low temperatures to allow their being treated in the zone of the arc except to be volatilized. But even in the case of the first-named metals the formation of an arc is not an absolute impossibility. At all events those metals need only form one electrode, and if the other, especially the positive one, is made of carbon, or of some other refractory substance, an arc may be very well maintained.

Concerning the electromotive force and intensity of a current to produce an arc, the length of the latter and the loss of energy in the same, it will require a great deal of work to study all possibilities that may occur in the electro-metal-

lurgical furnace; the following table, however, will give a few data upon which an approximate idea of the relation between the above factors may be based:

E. M. F.	Intensity.	Cross Section of Carbons.		Length of the Arc.	Loss of Energy.
		Total.	Per Ampère.		
Volts.	Ampères.	Sq. Mm.	Sq. Mm.	Millimeters.	Watts.
39.0	5.0	38	7.6	3.0	195
46.0	8.6	95	11.0	3.0	395
49.0	12.3	165	13.0	3.5	602
49.0	11.1	165	14.0	4.1	543
47.0	13.1	111	8.0	5.1	615
48.0	18.1	203	11.0	5.5	868
48.5	10.0	203	10.0	7.0	970
49.0	21.8	203	9.0	7.5	1,068

The temperature of the arc amounts to about 3500° at the positive crater; it is somewhat lower at the negative electrode, where it is estimated to be 2700°, while in the center most likely a temperature between 3500 and 4000° will be reached, supposing carbons to form both electrodes. It is obvious that the temperatures of arcs between metal or other less refractory substances will be limited by their boiling points.

When using the electric arc for smelting chemical compounds it must be borne in mind that electrolysis is unavoidable, in the substance to be melted as well as between the poles. Hence metals like platinum, iron, chromium, and others will take up carbon when heated in arc with one or more carbon electrodes.

The materials for constructing electric arc furnaces are as numerous as those for constructing resistance furnaces. Indeed everything said on this point with reference to the resistance furnace will apply equally well to the arc furnace.

In the latest constructions the principle of heating by the arc has generally been preferred, but it must be admitted that though the arc offers us the best facilities to produce the highest temperatures in the shortest time, its heating sphere is very limited, so that the distribution of electrically generated heat cannot be perfected nearly as well as by solid or liquid resistances; and no doubt the dimensions of an electric furnace are very much less restricted, when using resistance heating, than when using arc heating.

ELECTRO-METALLURGY.

It almost looks as if electro-metallurgy had lost its interest since the carbides appeared on the field. Except the usual number of patents there is hardly anything new to report from Europe.

Zinc and nickel have received the most attention. As to the former it is still generally admitted that rich zinc ores are best made use of by the old metallurgical processes, but it will certainly pay to utilize such ores and residues, which cannot be worked metallurgically. The Elektrochemische Zinkwerke at Duisburg now seems to be in good running order and also seems to have overcome the greatest difficulties in electro-depositing zinc from aqueous solutions obtained by leaching the roasted blende pyrites from the Siegen mines. They have succeeded in extracting the zinc from these products so completely as to leave only about 0.5% Zn in the residues.

The metal itself is very dense, being turned out in plates of about 10 mm. thickness. At present the monthly output of the works amounts to about 90 tons. For 1897 a considerable extension of the works has been resolved; from which fact we may infer that the financial success is sure. On the cost of production nothing has been published yet, and the process of working is also kept secret. From the appearance of the plates it seems that a very lively circulation of the electrolyte has been kept up.

Very little has been heard of the other works concerning which I reported last year. A third electrolytic zinc works is being erected in Duisburg by the firm of Matthes & Weber.

Besides the American nickel producers the German firm of Gustav Menne & Co., at Siegen, is furnishing electrolytic nickel. The metal is considerably harder than the American product and shows a small percentage of lead.

Competent German metallurgists do not consider the electrolytic method dangerous to the present smelting process for ores free from copper, since they readily obtain a metal with 99.2 to 99.5% Ni, a percentage which has not yet been beaten by electro-chemical refiners. Ores, of course, like those of Canadian mines which cannot be worked successfully by smelting processes, will naturally give fair results by electrolytic refining processes.

The new plants for recovering copper from ores by the well-known Siemens & Halske process have been started; but practical results demonstrating that the old difficulties have really been overcome have not yet been published.

There seems to be some demand for pure chromium, the manufacture of which has been taken up by the Elektrochemischen Werke, at Bitterfeld, Saxony. The same works are manufacturing, besides their regular products, magnesium and sodium metals.

INORGANIC CHEMICALS.

Carbide of calcium proves to be an inexhaustible source for the manufacture of old and new products. Commencing with the oldest and most interesting of its offspring, acetylene, the hope to make this gas at the carbide works, and to liquify it for transportation and storage, has come to grief, since with the pressure in the condensing process the danger of explosion is increasing. It will be safer, therefore, to store and ship the carbide and make the acetylene at the place of consumption.

Dr. Caro has obtained a patent on the manufacture of cyanides from carbides, especially from the carbide of barium. The process has one drawback, however: the cyanide is finally obtained as an aqueous solution, and evaporating such solutions means decomposing them. The process consists in heating the carbide in a current of nitrogen and steam, and then leaching the product thus obtained.

The chemical industry is using very large quantities of carbides as powerful reducing agents. For deoxidizing and for carbonizing Bessemer, Thomas, and Martin steel, calcium carbide has not met with success. At present calcium carbide is scarce in the market, though the European works are running at full speed to supply the demand.

CHROME ORE.

THE total production in the United States was derived from California as heretofore. This, however, is far from enough to meet domestic requirements, and a large quantity is imported annually, coming from Turkey, Asia Minor, and Canada. The chief supplies of chrome ore for the world are Turkey (Macedonia and Asia Minor) and Russia. The occurrence and mining of chrome ore were fully described in THE MINERAL INDUSTRY, Vol. II.; the manufacture of chromium salts in Vol. IV.

PRODUCTION, IMPORTS, AND CONSUMPTION OF CHROME ORE IN THE UNITED STATES.

(In tons of 2240 lbs.)

Year.	Production.			Imports.			Consumption.	
	Quantity.	Value Per Ton.	Value.	Quantity.	Value Per Ton.	Value.	Quantity.	Value.
1892.....	1,650	$10.00	$16,500	4,930	$11.27	$55,589	6,580	$72,089
1893.....	1,629	9.84	16,000	6,354	9.23	58,629	7,983	74,629
1894.....	3,285	12.14	39,980	3,886	9.87	38,364	7,171	78,344
1895.....	1,553	10.81	16,795	5,230	15.84	82,845	6,783	99,640
1896.....	702	11.07	7,775	8,869	21.13	187,400	9,571	195,175

California.—The deposits in Alameda County were idle in 1896, owing to low price of product, and distance from market. The mines of the Tyson Mining Company in San Bernardino County were also idle for the same reason.

The most important deposits of chromite in California are in the Santa Lucia Mountains in the northwestern part of San Luis Obispo County, where it occurs in bunches and seams penetrating zones of serpentine, but the ore is too low grade and scattered to be worked profitably at present prices. The San Luis Chrome Company, of San Luis Obispo, purchases such ore as is offered, crushes it, dresses it on vanners, and ships the products to Baltimore, Md. The total production of chrome ore in this county, coming from 23 different claims, has amounted to 17,860 tons. From Tehama County, the Tehama County Chrome Deposit Company, of Red Bluff, has shipped all told about 4500 tons of ore averaging 47% Cr_2O_3. Other deposits of chrome ore occur in Shasta, Sierra, and Sonoma counties, but they were not exploited in 1896.

The production of chrome ore in California since 1894 is given in the following table, where tons are of 2000 lbs. and values at point of production:

Year.	San Luis Obispo.		Shasta.		Tehama.		Total.	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
1894	a800	$10,500	1,200	$16,800	1,680	$12,680	3,680	$39,980
1895	700	6,650	90	1,120	950	9,025	1,740	16,795
1896							786	7,775

(a) Concentrates.

Asia Minor.—The chrome ores of Asia Minor, which are noted for their purity, occur near the towns of Alexandrette and Antioch, Province of Aleppo; in the coast district south of Smyrna; and in the Province of Broussa, the last being by far the most important.

The Broussa ore occurs in irregular pockets in serpentine, which are worked open-cast, the mining being done in a very unscientific manner. The cost of mining is, however, very low, wages being only 20c. per day in summer, and 16c. in winter, working-time being sunrise to sunset. The ore undergoes a rough sorting at the mine, to bring it up to a grade of 50% Cr_2O_3, after which it is shipped by camel to Kutahia, and thence by rail to Gemlek, the seaport.

Macedonia.—According to a report by the French consul at Salonica the mining of chrome ore in Turkey is increasing, due chiefly to improvements in transportation facilities, which have enabled producers to reach the seaports at a cost permitting competition in foreign markets. The chief mines are in the Monastir and Kossovo regions of Macedonia. The best grades of ore assay: Cr_2O_3, 55%; FeO, 13%; Al_2O_3, 10%; MgO, 15%; SiO_2, 7%. The exportation is principally to Germany, England, and Austria, the average price being 107.5f. per ton, f.o.b. in country of destination. The exports amount to 10,000 or 12,000 tons per annum. The business is mostly transacted at Salonica.*

Newfoundland.—Mr. George W. Maynard contributes the following data relating to the Newfoundland deposits of chrome ore: "Extensive outcrops of what were supposed to be magnetic iron ore were known to exist near Bluff Head, Port au Port Bay, and the west coast of Newfoundland, as early as 1893, but not until the summer of 1894 was the ore recognized as chromite, this having been determined by Mr. Obalski, the government mining engineer for the Province of Quebec. In 1895 I made the first extended investigation of the property, but this was necessarily confined to the location of outcrops and stripping at many points for the purpose of sampling the ore. The outcome of this preliminary work was the formation of The Halifax Chrome Company, organized under the laws of Nova Scotia. The exploration of the deposits was begun in the summer of 1896 and has resulted in the uncovering of large bodies of ore. The shipment of ore was necessarily limited because the road to the shore and other preparatory work was not completed before the inclement weather set in.

"During the current season (1897) vigorous work will be carried on in the opening of the mines, the construction of a tramway, the building of dressing works, and providing shipping facilities. It is the purpose of the company to market dressed ore only, after the completion of the plant, although pending this some lump ore may be shipped. This, however, will depend upon the

*According to the *Second General Report of the Mineral Industry of the United Kingdom* the exports from Salonica were 5000 tons in 1894 and 4000 in 1895

demand. The capabilities of the property cannot be absolutely determined until the end of the season of 1897.

"In 1896 about 200 tons were shipped, of which 145 tons were sold in Pittsburg, Pa., for the manufacture of bricks for basic steel furnaces, and 40 tons were sent to the steel works at New Glasgow, Nova Scotia, for refractory material. The remainder was sent to the United States for dressing and other tests.

"The ore shipped, although not closely sorted, ranged from 48 to 49.9% chromic oxide. The complete analysis of a lump gave: Chromium oxide, 49.23%; iron protoxide, 17.21%; magnesia, 18.66%; alumina, 7.50%; silica, 6.51%; lime, vanadium, copper, and manganese, traces—total, 99.11%.

"The ore occurs in serpentine, the country rock being diorite. The ore is easily mined and breaks in masses of many hundred pounds. Granular serpentine is scattered through the mass but is readily separated in the dressing. Concentrates containing 57% chromic oxide have already been obtained."

Quebec.—According to Mr. J. Obalski the total shipments of chrome ore from this province in 1896 were 2037.5 tons of 2240 lbs., of which 750 went to Philadelphia, 1232 to Pittsburg and vicinity, and 55.5 to other points. The total value of the product at the mines was $26,000. The ore sent to Philadelphia was of high grade, and the remainder mostly of low grade, the value at the mine being respectively $18 and $10 per ton. The total production of the province has been as follows: Previous to 1894, 50 tons, 1894, 915 tons, 1895, 2837 tons, 1896, 2037.5 tons. total, 5839.5 tons; on hand ready for shipment, 1100 tons; total production, 6939.5 tons. The stock on hand at the end of 1895 was about 1200 tons.

Russia.—In Russia the chrome ore from the Ourals, which was formerly exported, is now used at home by a new method of Wahlberg's, an improvement on that employed at the Gossleth works at Hrastnig in Styria. The chromite, graded to contain 45 to 47% Cr_2O_3, is glowed with limestone (or preferably chalk) crude potassium carbonate, and crude potassium sulphate in reverberatory furnaces, whereby K_2CrO_4 and $CaCrO_4$ are formed, the latter being converted into K_2CrO_4 by action of the K_2SO_4 during the subsequent leaching. In this way 15 to 19 units of the CrO_3 of the ore are extracted, about 7 units remaining in the residue. As the solution cools K_2SO_4 first crystallizes out, which is removed as far as possible. Sulphuric acid is then added until the solution reaches 35 to 37° B. sp. gr. The addition of acid is then continued until the solution becomes clear, and K_2CrO_4 is finally crystallized out, being purified by a subsequent recrystallization. The process is somewhat unhealthful to the operatives, and requires proper sanitary precautions.*

USES OF CHROME ORE.

The use of chrome ore for furnace lining is growing rapidly, and from the attention now being paid to it bids fair to become very important. At present the mineral is employed in reverberatory copper-smelting furnaces, and in open-hearth steel furnaces, in both with very satisfactory results. Concerning the former Herbert Lang writes:† "The Selby Smelting and Lead Company has used it

* *Berg- und Hütten. Zeitung*, 1897, pp. 1, 6. † *Engineering and Mining Journal*, Jan. 23, 1897, p. 89.

quite extensively in the matting reverberatories at Vallejo Junction, Cal. The hearth is composed of large fragments of chrome ore, with the interstices well filled with smaller pieces, and the material is built up around the sides to the slag-line. The protection is such that fettling is almost done away with; in fact they have to fettle but once a week or so. The chrome ore is acted upon slightly if at all." Alfred von der Ropp, superintendent of the works, states* that the life of the furnace is more than doubled by the use of chromite. George A. Packard, commenting upon Lang's statement, wrote:† "At South Strafford, Vt., in 1890, ore (chalcopyrite mixed with much pyrrhotite) was heap roasted and then smelted in a water-jacket furnace to a matte, carrying about 40% copper the separation of slag and matte taking place in a forehearth tamped with brasque. Considerable trouble was experienced on account of the rapid corrosion and deepening of the channel, cut in brasque, through which the matte and slag flowed from the furnace to the separating basin, and as an experiment this channel was lined with chrome ore. The result, as far as the non-corrosion of the chrome ore was concerned, was satisfactory, but as the channel was formed of a number of pieces of ore, we were obliged, after fitting the pieces as carefully as possible, to tamp brasque around them. This was sooner or later cut out, allowing the slag and matte to undermine and displace the pieces of ore."

The use of chromite for lining open-hearth steel furnaces was referred to in THE MINERAL INDUSTRY, Vol. II., p. 163. In the *Journal of the British Iron and Steel Institute,* Vol. II., 1893, p. 470, it is stated: "Hitherto the chrome ore used in the lining of open-hearth furnaces has been employed in lump form. Now the ore mined near Schweidnitz in Silesia is converted into a kind of cement. To ordinary fire-resisting cement is added a mixture consisting of 32 to 42 parts of chromic oxide, 18 to 22 parts alumina, and 18 to 22 magnesia, mixed with 10% of its weight of aluminum acetate."

This was manifestly an unnecessarily expensive lining, as subsequent experience demonstrated. In Vol. II., 1895, p. 506, *Journal of the British Iron and Steel Institute,* reference is made to an article in *L'Echo des Mines,* Vol. XXI., p. 584, as follows:

"According to P. Speier chrome-ore linings for reverberatory furnaces have been successfully adopted in French, German, and Russian steel works. The bottom and walls of the furnace are lined with chrome ore in large blocks, united by a cement formed of two parts of chrome ore finely ground, and one part of lime as free from silica as possible. The introduction of chromium from the lining into the bath of molten steel takes place to a very limited extent. The iron chromate is decomposed only under the influence exerted by the reagents and oxidizing alkaline substances. Heat alone is sufficient to decompose chromate of iron, which may float in a bath of molten steel covered with basic slag without dissolving. One of the principal conditions of success in the employment of the chrome-ore lining consists in carefully picking the pieces of ore used, which should be of uniform composition, and should contain less than 5% silica."

Referring to the same subject, Messrs. George G. Blackwell, Sons & Co. stated that chrome ore had been used by steel manufacturers in Lancashire and York-

* *Mining and Scientific Press,* Sept. 26, 1896, p. 257.
† *Engineering and Mining Journal,* Feb. 13, 1897.

shire for about 15 years. They also remark that it is not necessary to make the chrome ore into brick, a better method being to crush it to pea-size, mix with tar, and line the furnace as if with mortar. The tar burns away, leaving a solid lining of chrome ore, which has been found to give satisfactory results.

The consumption of chrome ore in open-hearth basic furnaces in the United States amounts to about 1800 or 2000 long tons annually. The Pittsburg supply has been for the most part from the Province of Quebec. The requirement is for an ore of not less than 45% chromic oxide, but according to George W. Maynard, the shipments have rarely averaged as high as this. The ore is used in lump form for patching the sides of the basic-lined furnaces, and is found far more convenient and durable than a slurry of dolomite.

There are two manufacturers of chrome brick in Pittsburg. The standard bricks measure 9 by $4\frac{1}{8}$ by $2\frac{1}{8}$ in. and weigh 9 lbs.; they are very hard and durable. They are used as a neutral parting between the basic hearth and silica roof.

The irregular supply of chrome ore and its high price has undoubtedly had much to do with its relatively limited employment as a refractory, and the opening of the large deposits on the west coast of Newfoundland will, it is believed, result in the employment of chrome ore to a much larger extent for basic linings and also for the manufacture of chrome steel.

Silichromite is a new compound, manufactured in France. It is an extremely hard, crystallized substance, but easy to pulverize, and is used for molding and polishing purposes. It is prepared by treating chrome ore, sand, and coal in an electric furnace, wherein a molten mass, in which chromium silicate predominates, is obtained.

According to G. de Chalmot,* chromium silicide, Si_2Cr, was obtained by heating together in an electric furnace silica, charcoal, and chromium sesquioxide. It crystallizes in long, gray needles having a metallic luster; it is not attacked by cold hydrochloric acid or aqua regia, but dissolves in hydrofluoric acid. A sufficient quantity of pure crystals was not obtained for analysis, but the composition was determined in two specimens, to which some free silicon adhered. The sp. gr. is 4.393.

The analysis of chrome ores has been discussed by Elwyn Waller, *Journal Soc. Chem. Ind.*, 1896, pp. 436–7, and E. H. Saniter, *ibid.*, 1896, p. 155; also by A. G. McKenna in the *Proceedings of the Engineers' Society of Western Pennsylvania*, Vol. XIII., No. 3, p. 180.

* *American Chemical Journal*, Vol. XIX., pp. 69–70.

Chrome Ore Mining in Quebec.

By Matthew Penhale.

The occurrence of chrome ore in the Black Lake district of the Province of Quebec was described in The Mineral Industry, Vol. IV., and the history of the discovery was also told there. Mining during 1896 was somewhat limited, as the pockets first found have in many cases been worked out. These pockets are found in serpentine rock of the same kind as that in which the asbestos deposits of Quebec occur; but the chrome ore and asbestos are never found together. The working of the chrome ore has been altogether by open pits or small quarries. No deep mining has yet been undertaken, and so far it has not been needed, since the outcrops followed have led only to bunches or pockets of ore, and nothing approaching a vein in form has been found.

For the most part the ore taken out has been by farmers and others, very few skilled miners having engaged in the work. Moreover, miners and prospectors have been discouraged by the action of the Coleraine Mining Company, which owns all the property known as Block A, Coleraine township, where the chrome ores have been discovered. This company, which is composed of Montreal capitalists, has done no work, and permits mining on its land only on payment of a royalty of 50%. It has no agent near the property and permits are not easy to obtain. Under these circumstances mining has been discouraged, no new discoveries have been made, and the production has not increased. If a reasonable royalty were charged and proper arrangements made there is no doubt that a considerable increase in output could be secured, as the outcrops indicate the existence of many pockets which have not yet been touched.

The district where the deposits are found is rough, rocky, and barren. It is not provided with roads, and the ore mined is piled up to be carried away in winter, when the snow on the ground makes hauling comparatively easy. The land is worthless for any purpose except mining.

The Anglo-Canadian Asbestos Company kept a small force at work taking out chrome ore on its property. There the chrome and asbestos deposits, in the serpentine, approach each other more nearly than at any other point. The asbestos mine is not now worked.

The ore from the workings is all shipped over the Quebec Central Railroad. The shipments from the Black Lake district in 1896 were as follows, in tons:

Philadelphia	750	Milwaukee	18
Pittsburg	1,216	Other places	18
New York	20	Total	2,022

The total compares with 2837 tons in 1895. The price of the ore varied from $12 to $26 per ton, according to its quality.

The largest deposit yet found is the Leonard Mine at Lake St. Francis, but the ore is of low grade, and brings a low price; so the mining has not been profitable.

It may be said, however, that there is a prospect of improvement in the Black Lake district in 1897, as several outcrops have been found outside of the Coleraine Mining Company's property. Several locations have been made with a view to opening the mines in the spring, and in a few places men have been at work during the winter.

CLAY.

At the present time there are no reliable statistics of the total production of clay in the United States. The industry is so diverse and extensive that it is in any case difficult to cover it. A good deal of clay is dug and converted into brick, pottery, and used otherwise in the arts by well-organized concerns from which reports are obtainable; but there are many firms which keep very imperfect accounts and probably do not know how much they produce; while a good deal of clay is dug incidentally by farmers and persons engaged in other occupations, who are quite unable to make returns of their production.

For statistical purposes clays may be divided satisfactorily into three classes, namely: 1, Kaolin or china clay; 2, refractory clay; 3, common clay. The occurrence, methods of digging, properties, composition, and applications of the various kinds of clay in the United States have been so fully described in former volumes of The Mineral Industry (especially in Vols. II. and IV.) that it is unnecessary to make further reference to them here. In the classification previously made the first group includes kaolin or china clay used in pottery-making. The second comprises all the argillaceous material used for withstanding high temperatures; this is chiefly made into firebrick and tiles of special form for furnace construction, but a good deal is used for the fabrication of zinc retorts and smelting crucibles, and as mortar for laying firebrick. Under the caption "Common clay" is included all which is employed for making ordinary building-brick, paving-brick, tiling, and drain-pipe, besides many minor uses; this class represents the greatest part of the clay production of the United States. The production of clays of all kinds in the United States is given in the following table:

PRODUCTION OF CLAY IN THE UNITED STATES. (IN SHORT TONS.)

Year.	China Clay.		Refractory Clay.		Common Clay	Total Value.
	Tons.	Value.	Tons.	Value.	Value.	
1893	30,183	$205,667	3,214,989	$4,822,483	$60,000,000	$65,028,150
1894	24,552	185,169	3,375,738	4,050,885	59,800,000	64,036,054
1895	30,910	258,431	3,750,000	4,500,000	60,100,000	64,858,431
1896	29,418	217,551	4,000,000	4,800,000	65,000,000	70,017,551

KAOLIN OR CHINA CLAY.

Kaolin or china clay is decomposed feldspar. It occurs in masses, or pockets, resulting from the decay of feldspathic granite in place, or it may form beds of sedimentary origin, though these are usually classed as "ball" clays. The residual kaolin is the purer, but even this has to be washed to free it from impurities, chiefly quartz and mica, before it is available for use in pottery. In the trade the term kaolin is applied only to the raw material, the clean product being referred to as china clay.

The chief sources of kaolin in the United States are Brandywine Summit, Pa., the neighborhood of Hockessin, Del., and from a point four miles west of Dillsboro, N. C.; recently also a good deal of china clay has been dug in Florida. China clay is produced in Colorado, the output there being chiefly consumed as a filler in paper manufacture. Deposits of kaolin exist at Blandford, Mass., at Glen Allen, Mo., and in Lawrence County, Ind. The clay from the last locality, which is known as indianaite, is one of the purest kaolins found in the United States, but little or none of it has yet been used in pottery making.

The crude kaolin is generally cleaned by washing in a long trough with riffles in its bottom, the sand and mica settling out in the trough, while the water together with the fine clay passes through a screen into tanks. After settling the mud is pumped into presses, where it is freed from water, dried, and is then ready for shipment. The yield of china clay from crude kaolin varies from 25 to 45%, averaging about 33⅓%. American china clay is worth from $11 to $12 per ton delivered at Trenton. Considerable English clay is used by American potteries, it being cheaper than the American product.

The composition of china clay from Brandywine Summit, Pa., is as follows: Silica, 46.278%; alumina, 36.25%; ferric oxide, 1.644%; lime, 0.192%; magnesia, 0.321%; alkalies, 2.536%; water, 13.535%—total, 100.748%. Both the Pennsylvania and Delaware china clays have a yellowish tint, but become white in burning. The color which china clay takes in burning governs largely its value, but its percentage of shrinkage is also an important factor. The presence of iron in the clay is likely to result in the production of brown specks, or a yellow tint on burning.

Distribution and Character of Missouri Clays.

By Charles Rollin Keyes.

The clay industry is one of the most important in Missouri, and the most extensive of the kind in the whole Mississippi valley. In the State, with its 67,380 sq. miles, there is not a single county out of a total of 114 that does not contain clay deposits of greater or less value. The State is unusually well supplied with clays suitable for the manufacture of nearly all grades of clay products, from common building brick to a high class of chinaware. The loess and residual clays are used largely, producing an excellent grade of building material. The shales of the coal measures furnish good material for sewer pipe and drain tile, firebrick and refractory products, stoneware and pottery; also for common, ornamental, building, and paving brick. Vast quantities of clay, unexcelled for burning into railroad ballast, exist, and are largely used by the principal railroads of the northern part of the State. It makes an ideal roadbed—smooth, elastic, and free from dust and weeds, easy for the rolling stock, thus contributing a large saving. There are also extensive deposits of kaolin and china clays now being developed, and some of these clays are shipped to the large potteries in the East.

Classification.—In the classification of clays their uses as recognized by the trade are given first place. While this is not always a natural arrangement, it is the only one in which the essential properties are taken into account. The following table is that followed by the Missouri Geological Survey in its reports:

1. Whiteware—Kaolin; china clay; ball clay.
2. Refractory—Plastic fireclay; flint clay; refractory shale.
3. Potters'—Plastic clay and shale of moderate fusibility.
4. Vitrifying—Paving-brick clay and shale; sewer-pipe clay and shale; roofing-tile clay and shale.
5. Brick—Common-brick clay and shale; terra-cotta clay; drain-tile clay.
6. Gumbo—Burnt-ballast clay.
7. Slip—Clays of very easy fusibility.

Distribution of the Different Varieties of Clays in Missouri.—That the occurrence of the various clays is somewhat dependent upon the geological formations is to be expected, and that there is a very close relation between the formation and the particular clay found in it is evident from a glance at a geological map of the State. Some districts contain only a few varieties of clay, while others contain nearly all the kinds in abundance. In most of the counties at least one branch of the clay industry has been developed, while in many a number of different lines are in operation, and in a few nearly all kinds of clay ware are made.

Properties and Values of Clays for the Various Commercial Purposes.—In the special investigation concerning the clays of the State which the Missouri Geological Survey has recently completed, particular attention was paid to those physical properties which give each clay its value in the arts. While sufficient chemical work was done to keep pace with the other tests, it was carried on rather as a check for comparisons with other clays. This was done for the reason that heretofore the usual and often the only test which has determined the use of a

clay has been a chemical analysis. In the recent experiments the physical tests were found to be much more reliable than the chemical, and formed very valuable indices for the practical working of the deposits on a commercial scale. The essential features of the testing done may be illustrated by one of the fireclays from a deposit 3 miles west of Clapper, in Monroe County.

A preliminary examination showed:

Color: Varying from light gray to drab, with occasional red and yellow iron stains.
Texture: Nearly uniform, coarse-grained, compact, massive, hard (2.0 to 2.5 sp. gr.).
Taste: Lean, gritty.
Slacking occurred readily and completely, the mass breaking down into irregular granules $\frac{1}{30}$ to $\frac{1}{10}$ in. in size.
Accessories: Pyrite not perceptible; sand as grains from $\frac{1}{30}$ to $\frac{1}{10}$ in. in size, freely disseminated.

A physical examination showed:

Specific gravity: 2.45.
Plasticity: As determined by working a paste in which the clay was ground to 20-mesh and mixed with 14% of water, relatively slight; as determined by the tensile strength of air-dried briquettes there was a maximum of 39 lbs. to the sq. in., the average of 10 samples being 33 lbs.
Water required to make an easily worked paste: 15.5%.
Air-shrinkage, in drying briquettes ranging from ¼ by ½ by 4 in. to 2 by 4 by 8 in., averaged in 10 samples 3.8%. Drying took place rapidly in a warm air-bath at 125° to 200° F., without checking.
Fire-shrinkage, in burning air-dried samples at a vitrifying heat, averaged in 3 samples 1.5%.
Total shrinkage averaged 5.3%, which is remarkably low.
Fusion took place at a temperature above 2700° F., or at a dazzling white heat; incipient vitrification occurred at 2500° F.
Color of burnt clay: White and very light gray.

Conclusions: This fireclay is an excellent refractory material for temperatures as high as 2700° F., or a dazzling white heat. It admits of rapid air-drying without checking, and does not fissure readily in firing. The total shrinkage being very low adds greatly to its value in any practical application of the clay. It is only moderately plastic, but when weathered for some time becomes more highly plastic and molds admirably. The burnt ware is strong. The clay may be used successfully for pottery. When washed and properly treated it is possible that whiteware may be produced.

A chemical analysis gave:

	Per Cent.
Silica	67.76
Alumina	21.96
Combined water	7.80
Moisture	0.43
Iron sesquioxide	0.69
Lime	0.96
Magnesia	0.24
Alkalies	0.24
Total	100.08
Total fluxing impurities	2.13

It has been impossible to make a complete investigation of more than a few of the most important deposits in each county. These will serve as standards of comparison for each community until the work can be pushed forward more in detail.

General Properties of the Principal Types of Missouri Clays.—The investigation was placed in immediate charge of Mr. H. A. Wheeler, and the following information regarding the deposits occurring in the State was largely derived

from his notes, though the field notes of other members of the geological corps have contributed no inconsiderable amount of details.

The Chinaware Clays.—The term chinaware is applied to all those higher grades of ceramic products variously called queensware, graniteware, French china, porcelain, and many other grades. The essential quality of the clays that are used for these purposes is that there should be a white body color after burning. The trade recognizes two general classes of clays for these purposes—the kaolin and the ball clays. The recent investigations in Missouri have disclosed many large deposits of both these varieties.

The kaolinite is of an excellent quality and is found chiefly in the southeastern part of the State in the Paleozoic limestones. The most extensively worked area is in Cape Girardeau and Bollinger Counties, but the belt is much more extensive and reaches as far southwestward as Howell County. Other deposits occur in the central part of the State, in Morgan and Cooper Counties; also in the extreme southwest.

The ball clays, which are the more plastic kaolins, are also well developed in several counties.

The kaolinite occurs in commercial quantities in no less than twenty counties. Besides Cape Girardeau and Bollinger, from which large shipments are made, extensive deposits are known in Perry, Ste. Genevieve, Madison, Lawrence, Morgan, and Cooper Counties. The other counties in which it is found, but in which it is not as yet mined, are Ozark, Howell, Oregon, Ripley, Carter, Shannon, Texas, Reynolds, Iron, Madison, and Wayne.

As yet the working of the kaolin deposits is carried on only in the crudest way. Mr. Wheeler says: "The development of the kaolin belt has been greatly retarded by the lack of capital and knowledge on the part of the owners of the kaolin pits, especially in not knowing how to sort the kaolin for the pottery trade, and this want of appreciation of the demands of the potter has heavily handicapped shipments from Glen Allen. It is necessary not only to carefully sort the pink or iron-stained clay and the flinty clay, but also to grade according to the fineness of the grit or sand in the clay, as it is very essential to have a uniform and reliable, as well as a pure, kaolin; and the Glen Allen shipments have therefore fallen somewhat into disfavor among potters because of carelessness in sorting. The entire district has suffered from the lack of a washer which would secure uniformity as well as purity. Until very recently there were none in this part of the State. Another serious drawback has been the careless system of mining, by which a heavy loss has been incurred by not timbering the pits. This has occasionally resulted in a loss of 75% of the clay. The system of mining usually adopted consists in sinking by a hand windlass a pit or shaft 4 to 5 ft. in diameter to the kaolin, which may be from 5 to 100 ft. deep. On reaching the marketable kaolin, the shaft is widened to 8 or 10 ft., and small drifts, usually 3 by 6 ft., are driven out through the kaolin bed. A small trench is dug around the mouth of the pit to carry off the surface drainage, and a windsail is sometimes erected to aid in ventilation; but the shaft is not timbered, the drifts are not supported, and reliance is placed on large pillars that are left to carry the overlying ground. The pits consequently cave in during wet weather, and are frequently abandoned, as a bailing bucket, worked by man or horse power, is the only way of removing

the water. When the pit is over 20 ft. deep, or the kaolin deposit is very extensive, a horse whim is usually erected to replace the hand windlass. Steam engines and pumping machinery have not yet been introduced, except in one or two instances. This crude system of mining is partly the result of inexperience in not knowing how to cheaply and safely work the treacherous kaolin beds, and partly due to their rather limited size.

"Washing, or the separation of the sand and flint by a careful system of 'floating,' is needed at almost all of the kaolin deposits, and many can never be profitably worked without it. Poor roads is another drawback that greatly hampers development, and usually prevents deposits from being worked that are more than 5 miles from the railroad. The hauling, in consequence, costs from 50c. to $2 per ton, according to the condition of the roads and distance, as it is done with light teams in farm wagons. In shipping, the kaolin has to stand a local freight rate to St. Louis, or the distributing point, of from $1 to $3 per ton, as the distance is from 130 to 250 miles, and a further charge eastward to the potteries of $2.50 to $4, making a total freight charge of $3.50, under the most favorable circumstances, to $7 per ton, usually over $5. As the kaolin brings from $8 to $12 per ton at the pottery, this does not leave a large margin for mining, royalties, and profit, when the heavy waste is considered in sorting out the stained and sandy clay.

"When a local market is secured by the establishment of potteries in or about St. Louis, and ample capital is employed to more economically mine and secure a very much more reliable product by washing, many deposits may be worked that are now out of the market; and it will stimulate prospecting over areas in which thus far there has been no inducement to look for kaolin. Improved haulage by the employment of broad tires and heavy teams will further aid in bringing outlying deposits into the market, especially if the roads are improved. Under the inducements of greatly reduced freights, a much less wasteful system of mining, with cheaper methods of hauling, and the ability to ship a uniform product by means of washing, an impetus will be given that at present cannot be predicted; while a development will result from the prospecting that will be thereby stimulated that will greatly enlarge the producing capacity of this district and very much widen the present zone of workable deposits."

The clays from Glen Allen and other points have been shipped for many years to the East Liverpool, Ohio, and other Eastern potteries.

A physical and chemical analysis of a typical sample of the Missouri kaolin, from a bed over 12 ft. thick occurring near West Plains, in Howell County, are given.

Physical characteristics:

Color: Very dark, almost black, which appears to be caused by the presence of organic matter, as it disappears upon burning; contains a few yellow streaks.
Texture: Massive, compact, soft.
Taste: Fat, slightly of alum.
Slacking took place rapidly and completely, the granules being from $\frac{1}{40}$ to $\frac{1}{10}$ in. in size.
Accessories: Pyrite not noticeable; black calcareous specks sparingly scattered through it.
Plasticity: When ground to 20-mesh and mixed with 30.5% of water, it made a very plastic paste; as measured by the tensile strength of air-dried briquettes, a maximum of 126 lbs. to the sq. in., with an average of 113 lbs.
Shrinkage: In drying, 8.5%; in vitrifying, 9.5%; giving a total of 18%.

Fusion occurred at 2500° F.; incipient vitrification at 2000°.
Specific gravity: 1.76.
Character of ware when burned was pure white, but tender when vitrified unless cooled very slowly.

A chemical analysis (A) gave:

	(A.) Per Cent.	(B.) Per Cent.
Silica	60.55	91.05
Alumina	24.77	5.01
Combined water	12.86	2.74
Iron sesquioxide	0.84	0.69
Lime	0.25	0.24
Magnesia	0.41	0.22
Alkalies	0.68	0.12
Totals	100.36	100.07

Silica.—Along with the kaolin beds occur certain other deposits of "silica," which should be mentioned in this connection, as the substance is an important one in the manufacture of pottery. Cape Girardeau County furnishes the best and most abundant deposits. A chemical analysis of a typical sample (B) taken from the Brooks pit, 5 miles north of the town of Cape Girardeau, gave the above results.

Feldspar.—Another ingredient important to whiteware production is feldspar. In the region of crystalline rocks of Southeast Missouri there are a number of deposits yielding this mineral. Those in Ste. Genevieve County are the most important so far as known. They consist of large pegmatite veins in red granite. The deposits have already been worked to some extent for pottery purposes.

Ball Clays.—These are the more plastic china clays and originate in two ways. They are either accumulations of residual material in limestone or weather from very pure flint clays. They are chiefly confined to Southeast Missouri. They are largely mined in Jefferson, Franklin, Crawford, and Warren Counties. Typical samples collected showed the following composition:

	Regina, Jefferson Co. Per Cent.	Union, Franklin Co. Per Cent.
Silica	45.97	44.14
Alumina	36.35	39.86
Combined water	12.36	13.84
Iron sesquioxide	1.08	0.46
Lime	1.14	0.77
Magnesia	1.09	0.46
Alkalies	1.84	0.76
Totals	99.83	100.29

At present no whiteware is manufactured in Missouri. The clays mined are shipped to Eastern potteries. The natural advantages, however, for carrying on the manufacture of pottery at St. Louis are exceptionally good, among them being: (1) Cheap raw material—kaolin, bond clay, feldspar and flint, which are all found in Missouri; (2) cheap fireclay (from 75c. to $3 per ton, according to quality) and cheap firebrick for the saggers and kilns; (3) excellent cheap fuel (from $2 to $2.25 per ton); (4) large home market; (5) unexcelled shipping facilities.

Fireclays.—The clays suitable for the manufacture of refractory wares are of two principal kinds—the flint or non-plastic varieties, and the plastic or ordinary

fireclays. The former are very pure, compact and hard, breaking with a conchoidal or flinty fracture. Not possessing the most essential characteristic of clay, it is necessary to mix them with a plastic variety in order to mold them.

The plastic varieties of fireclay are those which occur usually associated with coal and constitute the under clays. As might be expected, their capacities for resisting high degrees of heat vary greatly, from those that are rather easily fusible to those which will withstand very high temperatures. The range exhibited by the various kinds is from 1900° to 2700° F. In the trade it is usually necessary to have the manufactured product withstand a temperature of at least 2500°, and this temperature may therefore be taken as the minimum limit in determining whether the clay is capable of being used for refractory products or not. About one-half of the under clays are capable of withstanding this temperature.

The occurrence of the plastic and of the non-plastic fireclays is very different. The former are usually closely associated with the coal seams of the Carboniferous. The flint or non-plastic fireclays invariably occur in isolated basins or pockets in limestone, and appear to occupy old sink-holes.

A comparison of the analyses of the typical plastic fireclay, the washed potclay, and two samples of the flint or non-plastic fireclay is as follows:

	I.	II.	III.	IV.
Combined silica	31.39	26.03	43.56	42.60
Free silica	29.38	37.25		
Alumina	23.56	21.16	41.48	41.88
Combined water	9.25	8.94	14.05	14.00
Titanic acid	0.96	1.07	.....	.....
Ferric oxide	4.69	1.81	0.35	0.62
Ferrous oxide	0.47	0.82	.....	.....
Lime	0.55	0.61	0.45	0.28
Magnesia	0.15	0.30	.....	0.20
Potash	0.92	0.51	0.20	0.54
Soda	0.08	.....		
Sulphur	0.09	0.12	.....	.....
Sulphuric acid	0.35	0.56	.....	.....
Totals	101.84	99.18	100.09	100.12
Total fluxes	7.30	4.73	1.00	1.64
Specific gravity	2.47	2.13	2.45	2.43

I. is a sample of the mine-run of the Cheltenham fireclay seam of St. Louis County, upon which is founded one of the largest and most important fireclay industries in the United States.

II. is the washed potclay of the same fireclay seam.

III. is a sample of flint clay from one of the most extensive beds near Truesdale, in Warren County.

IV. is a sample of flint clay from one of the large deposits near Anaconda, in Franklin County.

The difference between the plastic fireclays and the flint or non-plastic varieties is further shown by a comparison of the results of the physical examinations. A sample of II. gave the following characteristics:

Color: Uniform light gray.

Texture: Uniform, coarse-grained, soft, and rather porous.

Taste: Smooth and fat, with a little fine grit.

Slacking took place somewhat slowly but completely, the mass breaking down into flakes and granules $\frac{1}{40}$ to $\frac{1}{10}$ in. in size.

Accessories: Pyrite was not noticeable with the lens, but considerable fine sand was present.

Water required to make a plastic paste, the clay being ground to 20-mesh, was 20%.

Shrinkage: In drying, 7.7%; in burning, 5.1%; giving a total shrinkage of 12.8%.

Plasticity, as shown by the tensile strength of air-dried briquettes, was 196 lbs. to the sq. in. as a maximum, with an average of 168 lbs.

Fusion: Viscous vitrification occurred above 2700°; incipient at 2200° F

The sample from the beds north of the Missouri River (III.) showed the following physical characteristics:

Color: Mostly white or very light gray, with some iron stains on the joint planes.
Texture: Massive, compact.
Fracture: Conchoidal.
Taste: Extremely lean.
Slacking did not take place at all.
Accessories: No pyrite visible.
Water required to make as plastic a paste as possible, the clay being ground to 20-mesh, was 15 to 20%.
Shrinkage was 2.5% in drying and 9% in vitrifying, giving a total of 11.5%.
Plasticity, as measured by the tensile strength of air-dried briquettes, was 20 lbs. to the sq. in. as the maximum, and 13 lbs. on an average.
Fusion: Viscous vitrification occurred above 2700°; incipient at 2300° F.

The flint clays occur in 15 counties, chiefly those situated around the northern flank of the Ozark uplift. The principal counties now shipping are Warren, Montgomery, Callaway, Franklin, Phelps, Crawford, and Morgan. The plastic fireclays are mined in more than a score of counties, but the chief development is in St. Louis, which contains the largest and most important single seam in the State.

Potters' Clays.—The term as here used is intended to cover all those clays that are highly plastic, and are too fusible for refractory goods, but that contain low percentages of iron. They are suitable for all the common grades of stoneware and earthenware, and for the light-colored building and ornamental brick and terra cotta, though their uses are not confined to these wares.

The stoneware clays occur in four different formations: In pockets, in the southern half of the State; as the under clays of the coal seams, in the northwestern half of the State; in the unconsolidated Tertiary of the extreme southeastern corner of the State; and as limited beds or lenticular masses, in the glacial drift north of the Missouri River. These clays are the most widely distributed of any in the State, and workable deposits occur in nearly every county. The following analyses indicate the main characteristics of the Missouri potters' clays:

	I.	II.	III.
Silica	45.97	71.94	68.50
Alumina	36.35	17.60	20.81
Water	12.36	5.27	7.62
Iron sesquioxide	1.08	2.35	1.79
Lime	1.14	0.62	0.77
Magnesia	1.09	0.56	Trace
Alkalies	1.84	1.51	0.53
Totals	99.83	99.85	100.02
Total fluxing impurities	5.15	5.05	3.09
Specific gravity	1.90	2.34	2.13

I. is from a pocket deposit in Silurian limestone near Regina, in Jefferson County. It is essentially a chinaware clay.

II. is from the coal measures at Calhoun, in Henry County, the largest producing locality in the State.

III. is from the Tertiary beds near Dexter, in Stoddard County. For many years it has supplied the stoneware pottery at that place.

The physical examination of the Calhoun clay showed the following characteristics:

Color: Dark gray, with occasional light iron stains.
Texture: Massive, compact, uniform.
Taste: Lean, with very fine grit.
Slacking took place rather readily and completely, the mass breaking down to granules $\frac{1}{40}$ to $\frac{1}{10}$ in. in size.
Accessories: Pyrite was not noticeable; a very little fine sand was present.
Water required to make a plastic paste, the clay being ground to 20-mesh, was 16.5%.
Shrinkage: 5.5% in drying, and 2.2% in vitrifying, giving a total of 7.7%.
Plasticity, as measured by the tensile strength of air-dried briquettes, gave a maximum of 168 lbs. to the sq. in., with an average of 150 lbs.
Fusion: Incipient occurred at 2100°, complete at 2300°, and viscous above 2500° F. The ware dried rapidly, but required slow heating to avoid checking. It burned to a compact gray to buff body when vitrified.

Regarding the stoneware industry in the State Mr. Wheeler says: "The manufacture of stoneware in Missouri is thus far confined, with a single exception, to small hand potteries. There are 49 potteries in the State, which have only 57 kilns in all. Nearly all the potteries are in out-of-the-way places, that are frequently inaccessible by rail, and depend entirely upon local trade. These small potteries can successfully hold the local trade, especially away from the railroad towns, but when competing in the larger markets they are not prepared to meet the prices that are made by the steam potteries of Illinois, Indiana, and Ohio. The only successful effort that has been made to secure a large shipping business is in Henry County, where there are 1 large steam steam pottery and 3 small hand potteries that are worked with considerable vigor. They are favored with excellent clay, while the fuel is mined almost at their very doors. At most of the potteries the clay is washed, as but few of the clays are sufficiently free from sand and other foreign matter to be safely used without this treatment. The method of washing is crude and takes a long time, while the output is limited, but it suffices for the comparatively small production.

"The usual price realized for the finished ware of No. 1 grade at the kilns is 5c. per gal. It is not an uncommon thing for many of the potteries to sell a large portion of the ware at the kilns direct to those calling there for it. The molding is usually done by contract, at so much a gallon, which ranges from 1 to 2c., according to the character and size of the ware. As most of the potteries are small hand establishments with a single kiln, and as the local market will not generally justify the running of the plant more than a portion of the year, the best of work is not always turned out on account of insufficient practice and lack of demand for the higher grades of goods. Such indifferent, intermittent work does not attract the best potters, so that much of the work is not equal to the better grades that are turned out at the large steam plants."

Shales.—One of the most abundant of the argillaceous materials suitable for the manufacture of various grades of clay ware is the shales, which occur chiefly in the coal measures. They are essentially indurated clays, and when exposed to the weather quickly form beds of soft, plastic clay. The shales furnish material for most of the coarser clay wares. They all contain large percentages of impurities, and are easily fusible. Their areal distribution in the State may be regarded as co-extensive with the lower coal measures, though there are also important beds in the upper coal measures and in the older formations. A typical sample from

Town Creek, near Clinton, in Henry County, shows the following physical characteristics:

Color: Bluish.
Texture: Lamellar and mostly very fine-grained, with some coarse, sandy streaks.
Taste: Lean, slightly of alum.
Slacking did not take place.
Accessories: Pyrite in thin streaks; mica in occasional scales, and sand as fine grit.
Water required to form a paste, in which the material was ground to 20-mesh, was 17%.
Shrinkage: In drying, 2%; in vitrifying, 4.5%; giving a total of 6.5%.
Plasticity, as measured by the tensile strength, averaged 38 lbs. to the sq. in.
Fusion: Incipient at 1800°; complete at 2000°; viscous above 2200° F.
Character of clay: It heated without cracking, dried rapidly, and burned to a dark-red body that was lacking in strength from the leanness of the shale. It is too lean to be worked by itself, but makes a fair quality of paving brick, sewer pipe, and other coarse ware, if mixed with sufficient plastic clay.

A chemical analysis gave the following results:

	Per Cent.
Silica	54.69
Alumina	25.96
Combined water	8.90
Iron sesquioxide	4.97
Lime	0.18
Magnesia	0.15
Alkalies	3.58
Total	98.43
Total fluxing impurities	8.88
Specific gravity	2.26

Among the uses to which the shales are put may be mentioned the manufacture of paving brick, drain tile, sewer pipe, terra cotta, building and ornamental brick and terra-cotta lumber. The most important are for paving and ornamental brick and sewer pipe. Potters' clays and certain brick clays may also be used for these purposes, but the shales are employed almost entirely.

In the manufacture of terra cotta, roofing tiles, sewer pipe and drain tile the annual consumption of clay in Missouri is about 170,000 tons, valued at $115,000, and the value of the product is $1,700,000.

The terra-cotta industry is but in its infancy, though it is rapidly growing. Some of the boldest effects obtained in this country have been produced by the St. Louis works. Single pieces 12 ft. high, 3 ft. wide, with a projection of 5 ft., have been made.

The manufacture of roofing tile is also of very late origin in Missouri, but the use of this grade of ware is rapidly becoming known, and, in the better class of buildings, tiles are being used extensively.

The manufacture of sewer pipe is assuming large proportions. The industry is located chiefly in St. Louis, where there are 3 large companies; at Kansas City, and at Deepwater, in Henry County. The invested capital is over $1,000,000 and the value of the annual production over $1,100,000. The market that the sewer-pipe works of Missouri supplies is a wide one, extending all over the western part of the country west of Ohio.

The drain-tile industry, which supplies the porous earthenware pipes for draining farm lands, is carried on by a large number of small local concerns. The total annual output is about 3,150,000 tiles, valued at about $50,000. Many of the brick yards also manufacture drain tile.

The manufacture of flower pots is confined to local establishments, of which there are only 7 in the State. The capital invested is $30,000 and the annual output is about the same amount.

The paving-brick industry is destined to be one of the most important in the State. The material has been used chiefly in the smaller cities, but of late it has been adopted on an extensive scale in the larger cities. The shales are pre-eminently adapted for the manufacture of vitrified paving brick. The chemical composition, as compiled by Mr. Wheeler from 50 analyses, is as follows:

	Minimum Per Cent.	Maximum Per Cent.	Average Per Cent.	Total Average.
Silica	49.0	75.0	56.0	
Alumina	11.0	25.0	22.5	
Ignition loss	3.0	13.0	7.0	
Moisture	0.5	3.0	1.5	
Total non-fluxing constituents				87.0
Iron sesquioxide	2.0	9.0	6.7	
Lime	0.2	3.5	1.2	
Magnesia	0.1	3.0	1.4	
Alkalies	1.0	5.5	3.7	
Total fluxing constituents				13.0
Grand total				100.0

At present there are 13 plants in the State, the annual output of which is valued at about $350,000. In addition to their use for street paving, the vitrified brick are used largely for sewers, sidewalks, and chemical tanks. The underburned brick are available for all ordinary building purposes.

Building Brick Clays.—The building brick clays are very impure and sandy. They are yellow to brown in color on account of the large amount of iron present. Being the most widely distributed of any of the clays, they occur in every county in the State. They are divisible into four general classes according to origin—the loess, the glacial, the residual, and the alluvial clays.

The loess clays are by far the most valuable. They occur chiefly in a broad belt 5 to 10 miles wide and from 10 to 50 ft. thick along the two principal streams, the Mississippi and the Missouri. The clays are very homogeneous and are almost entirely free from coarse material. The great uniformity shown in the composition and physical characters is shown by Mr. Wheeler's tables:

PHYSICAL TESTS OF LOESS CLAYS.

Locality.	Plasticity.	Size of Grain.	Tensile Strength.		Per Cent. Water in Paste.	Shrinkage.							Fusion (F.).		
						In Drying.			In Burning.			Total.			
			Average.	Maximum.		No. of Samples.	Shrinkage.	Speed	No. of Samples.	Shrinkage.	Speed		Incipient.	Complete.	Viscous
Kansas City	Lean	F...	151	175	18.4	14	5.1	Slow.	1	5.7	Rapid	10.8	2,000	2,200	2,300
Boonville	Very lean	F.C.	97	112	19.3	15	4.7	M. R.	1	5.1	Rapid	9.8	2,000	2,200	2,300
Jefferson City	Lean	F.C.	131	149	17.2	16	5.7	Slow.	1	4 3	Rapid	10.0	2,000	2,200	2,300
Hannibal	Lean	F...	126	148	18.4	14	4.5	M. R.	1	5.9	M. R.	10.4	2,000	2,200	.....
St. Louis	Slightly lean	C...	173	205	17.1	16	5.3	Slow.	1	5.5	Rapid	10.8	1,800	2,000	2,100
Benton	Rather lean.	C...	169	190	19.2	14	3.3	M. R.	2	3.4	M. R.	8.7	1,800	2,000	2,100

ANALYSES OF MISSOURI LOESS CLAYS.

Locality.	Silica.	Alumina	Combined Water.	Iron Sesquioxide.	Lime.	Magnesia.	Alkalies.	Total Fluxes.	Total.	Specific Gravity
Kansas City	72.00	11.97	6.42	3.51	1.80	1.35	3.25	10.11	100.30	2.17
Boonville	71.11	11.62	6.71	3.90	2.37	1.47	3.14	10.88	100.32	2.20
Jefferson City	74.39	12.03	3.17	4.06	1.50	1.53	3.01	10.09	99.68	2.09
Hannibal	73.80	13.19	5.26	3.43	0.86	0.68	2.94	7.91	100.16	2.17
St. Louis	73.92	11.65	5.24	4.74	1.43	0.60	3.13	9.90	100.73	1.98

The glacial clays cover all of the State north of the Missouri River. As in other parts of the Mississippi valley, these clays are predominantly yellow to brown in color, passing below into light blue. They contain pebbles and boulders and considerable coarse sand in pockets or thin layers. Often beds of pure and homogeneous clay suitable for the better grades of earthenware occur. As a rule all the drift clays are so extremely plastic that they cannot be utilized for clay ware.

The residual clays occur in the northern part of the State under the drift, but south of the Missouri River they are surface clays. They are yellow, brown or deep red in color, waxy and very plastic. They are derived chiefly from the decomposition of the limestone.

The alluvial clays are the result of recent stream action. They are usually very impure and mixed with a large percentage of sand, gravel and vegetable matter. Occasionally extensive deposits of high-grade clay occur. They are practically free from the coarse ingredients. The strong plastic clays of the drift and alluvium are termed "gumbo," and are especially well adapted for railroad ballast. Of recent years this has become an important industry, and most of the railroads use the material in preference to all others. The value of the annual output is over $1,000,000. The yearly production of all kinds of brick is about 450,000,000, valued at $3,500,000. The present manufacturing capacity of the St. Louis district alone is over 3,000,000,000 a year.

COAL AND COKE.

THE production of coal in the United States reached in 1896 a total of 186,241,271 short tons, of which 48,855,563 tons were anthracite, 137,331,048 tons bituminous, and 54,660 tons cannel coal. As compared with 1895 there was a total decrease in production of 2,812,119 short tons, or nearly 1½%, which is a remarkably slight change when the general condition of the trade and of the manufacturing industries of the country is considered. Moreover this decrease was almost wholly due to the restricted production of Pennsylvania anthracite. The output of anthracite coal in 1896 was less than that of 1895 by 3,041,734 short tons, or nearly 5.9%; that of bituminous coal showing a small gain of 244,702 short tons, or about ⅕%. This was due mainly to the sustained activity in the Southern and a few of the Western States.

The anthracite production is, of course, almost entirely from the mines of Eastern Pennsylvania. The small output reported from Arkansas showed in 1896 only a trifling change, while that from the Colorado mines showed a decrease relatively large, but small in actual amount. The conditions governing the production from the Pennsylvania mines and the causes which led to so considerable a reduction in the output are shown clearly in the article on the "Anthracite Coal Market" in the following pages, in which the shipments are given in detail. The total production as given in the table herewith is based on the reports of the mine inspectors. Anthracite in 1896 was somewhat over one-fourth of the total coal supply or 26.1%, nearly, while only twenty-one years ago, in 1875, it amounted to 43.48% of the whole. The proportion of anthracite in the total production of coal is steadily and rapidly declining. The production of anthracite in Pennsylvania will before many years reach its maximum and will thereafter decline rapidly. At present rather important quantities of bituminous coal are brought 150 to 200 miles by rail, and are used with economy in the anthracite districts even for the engines at the anthracite collieries. The anthracite trade of Pennsylvania is in a very unsatisfactory condition; the mines are being rapidly worked out, and the future presents but little that is encouraging.

Two of the States which lead in the production of bituminous coal show decreases in production: Pennsylvania having mined in 1896 a quantity smaller by 1,539,456 tons, or 2.9%, than in 1895, while there was a small reduction in Ohio. Illinois, on the other hand, increased its product by 2,050,762 tons, or 1.2%. In West Virginia there was a very slight gain. The Southern States held their own very fairly, the changes shown being small. In Alabama there was a slight

increase shown, which was not expected, in view of the condition of the iron trade, which is the most important consumer of Alabama coal. The mine operators of the State have, however, been making active efforts to extend their trade, with considerable success. Alabama coal now supplies the Gulf ports to a great extent, and a beginning has been made in supplying the lower Mississippi Valley, which in times past depended on Pennsylvania and West Virginia.

The production of the Pacific coast increases slowly. Washington makes the best showing, and seems to be the State having the best prospects for the future in this direction. Oregon mining is not extending. The output of the California mines is small, though an important increase is promised by the San Joaquin Coal Company at Corral Hollow, where extensive developments are in progress. The supply of the Pacific coast continues to come largely from abroad.

The cannel coal reported is entirely from Kentucky, the supply coming from a few mines in the eastern part of the State. This coal has a special market, and commands a much higher price than the average bituminous coal.

TOTAL PRODUCTION OF COAL AND COKE IN THE UNITED STATES.

(In tons of 2000 lbs.)

COAL.

States.	1895.			1896.		
	Tons.	Value.		Tons.	Value.	
		Totals.	Per Ton.		Totals.	Per Ton.
Bituminous:						
Alabama	5,680,410	$5,680,270	$1.00	5,747,698	$5,460,314	$0.95
Arkansas	797,424	1,993,569	2.50	889,750	2,001,937	2.25
California	80,115	240,345	3.00	69,449	158,135	2.25
Colorado	3,449,000	4,242,270	1.23	3,312,000	4,968,000	1 50
Georgia	414,310	364,593	0.88	246,012	214,030	0.87
Illinois	a17,735,864	16,316,995	0 92	a19,786,626	14,246,370	0.72
Indiana	4,312,084	4,096,480	0.95	4,068,124	3,152,796	0.77
Indian Territory	a1,228,440	1,904,082	1.55	a1,235,333	1,914,766	1.55
Iowa	a3,995,836	5,076,434	1.27	a4,004,508	5,085,725	1.27
Kansas	3,190,843	3,590,141	1.13	3,141,343	3,253,583	1.03
Kentucky	3,138,023	2,573,179	0.82	3,128,818	2,346,623	0.75
Maryland	3,479,499	3,061,959	0.88	3,627,425	3,192,130	0.88
Missouri	a2,283,081	2,625,543	1.15	a2,420,147	2,883,169	1.19
Montana	1,104,854	2,300,300	2.08	1,157,722	2,262,768	1.96
New Mexico	367,442	551,162	1.50	332,600	465,640	1.40
North Carolina	15,386	24,771	1.57	14,970	23,203	1.56
North Dakota	79,850	99,475	1.25	94,500	103,950	1.10
Ohio	13,683,879	11,904,975	0.87	12,912,608	10,071,834	0.78
Oregon	65,918	131,836	2.00	60,739	121,478	2.00
Pennsylvania	51,813,112	36,787,310	0.71	50,273,656	33,180,613	0.66
Tennessee	2,319,720	1,809,382	0.78	2,663,714	2,024,422	0.76
Texas	499,668	1,064,293	2.13	585,000	1,228,500	2.10
Utah	530,713	769,533	1.44	429,446	519,801	1.21
Virginia	1,083,229	812,421	0.75	1,018,237	763,677	0.75
Washington	1,184,619	3,553,857	3.00	1,295,313	3,562,110	2.75
West Virginia	12,355,113	8,895,681	0.70	12,541,382	8,778,967	0.70
Wyoming	2,197,914	3,077,080	1.40	2,273,928	2,946,106	1.30
Total bituminous { Short tons	137,086,346	$123,547,936	$0.90	137,331,048	$114,930,647	$0.84
Total bituminous { Metric tons	124,363,917		1.00	124,585,909		0.92
Cannel:						
Kentucky { Short tons	69,747	191,804	2.75	54,660	146,488	2.68
Kentucky { Metric tons	63,274		3.03	49,587		2.95
Anthracite:						
Arkansas	103,247	309,741	3.00	103,420	310,260	3.00
Colorado	133,393	373,500	2.80	59,600	149,000	2.50
Pennsylvania	51,660,657	79,567,411	1.54	48,692,543	87,646,577	1.80
Total anthracite { Short tons	51,897,297	$80,250,652	$1.54	48,855,563	$88,105,837	$1.80
Total anthracite { Metric tons	47,080,919		1.70	44,321,476		1.98
Grand total coal { Short tons	189,053,390	$203,990,392	$1.08	186,241,271	$203,182,972	$1.09
Grand total coal { Metric tons	171,508,110		1.19	168,956,972		1.20

COKE.

States.	1895. Tons	1895. Value. Totals.	1895. Value. Per Ton.	1896. Tons.	1896. Value. Totals.	1896. Value. Per Ton.
Alabama	1,232,242	$2,094,811	$1.70	1,247,028	$2,057,596	$1.65
Colorado	310,950	621,900	2.00	324,700	974,100	3.00
Georgia	90,620	144,086	1.58	55,467	88,737	1.60
Illinois	3,200	5,280	1.65	2,900	4,640	1.60
Indiana	4,760	8,330	1.75	4,890	7 610	1.55
Indian Territory	7,000	16,590	2.38	7,156	16,816	2.35
Kansas	8,350	16,700	2.00	8,147	16,294	2.00
Kentucky	25,453	40,725	1.60	24,519	36,778	1.50
Montana	61,378	506,417	8.25	63,694	461,260	7.26
New Mexico	1,309	3,024	2.31	779	1,791	2.30
Ohio	17,150	27,905	1.59	16,184	27,998	1.73
Pennsylvania	8,922,329	12,848,154	1.44	6,613,180	10,581,088	1.60
Tennessee	371,740	557,610	1.52	332,746	499,019	1.50
Utah	34,265	75,383	2.20	32,491	71,480	2.20
Virginia	137,826	203,982	1.49	131,400	193,138	1.47
Washington	7,318	43,918	6.00	22,368	89,472	1.45
West Virginia	1,248,574	1,785,461	1.43	1,452,392	2,105.968	4.00
Wyoming	9,000	18,000	2.00	19,543	38,086	2.00
Total coke (Short tons)	12,493,464	$19,018,276	$1.52	10,359,584	$17,271,871	$1.67
Total coke (Metric tons)	11,333,996	.....	1.68	9,398,153	.....	1.84

(a) Fiscal year.

UNITED STATES PRODUCTION OF COAL, 1892–96. (IN TONS OF 2000 LBS.)

Year.	Alabama	Arkansas	California.	Colorado.	Georgia.	Illinois.	Indiana.	Indian Territory	Iowa.	Kansas.
1892..	5,314,227	739,300	131,431	3,771,234	165,000	17,862.276	4.494,811	1,004,765	3,820,000	2,794,000
1893..	5,170,042	740,000	75,000	3,946,232	372,191	19,949,564	a 4,583,000	1,229,562	a 3,790,000	2,881,930
1894..	4,381,295	a 723,860	94,754	2,994,028	354,111	a 17,113,576	a 3,065.394	1,072,542	a 3,776,373	2.701,191
1895..	5,680,410	900,671	80,115	3,582,393	414,310	17,735.864	4,312.084	1,228,440	3.995.836	3,190,843
1896..	5,747,698	993,170	69,449	3,371,600	246,012	19,786,626	4,068,124	1,235,333	4,004,508	3,141,343

Year	Kentucky.	Maryland	Michigan	Missouri.	Montana.	New Mexico.	North Carolina.	North Dakota.	Ohio.	Oregon.
1892..	3,020,050	3,036,283	70,000	3,017,285	648,701	434,291	.....	.....	14,599,908	34,720
1893..	3,302.250	3,327,749	22,072	a 3,190,442	783,300	457,085	17,000	b 125.000	14.828.097	41,478
1894..	2,967,195	3,101,082	.....	a 2,383,322	688,780	323,721	13,157	b 65.000	11,902,678	42,509
1895..	3,207,770	3,479,499	.....	2,283,081	1,104,854	367,442	15,886	79,850	13,926,133	65,918
1896..	3,183,478	3,627,425	.....	2,420,147	1,157,722	322,600	14,970	94,500	12,912,608	60,739

Year.	Pennsylvania. Bituminous.	Pennsylvania. Anthracite.	Tennessee.	Texas.	Utah.	Virginia.	West Virginia.	Washington.	Wyoming.	Other States.	Totals.
1892	46,576,576	52,472,504	2,413,678	300,000	363,020	800,000	8,710,878	1,000,000	2,454,449	.....	180,311,504
1893	43,421,898	47,179,563	1,857,432	322,745	421,400	842,933	a 9,838,011	1,211,550	2,243,401	.....	176,180,927
1894	41,867,188	51,828,405	2,589,664	469,904	453,601	925,837	a 10,559,926	1,183,660	2,224,135	8,000	169,875,781
1895	51,813,112	51,660,657	2,319,720	499,668	530,713	1,083,229	12,355,113	1,184,619	2,197,914	.....	189,295,644
1896	50,273,656	48,692,543	2,663,714	585,000	429,446	1,018,237	12,541,382	1,295,313	2,273,928	.....	186,231,271

(a) Fiscal year. (b) Estimated.

The most marked feature shown in our tables is the extremely low average price realized for coal. In 1895 the average price for bituminous coal at the mines was 90c. per short ton, which seemed insufficient to leave any profit for the operators; in 1896 there was a still further fall, and the average price reported, at the mines, was 83.6c. per ton, and in Pennsylvania it was only 66c. and in West Virginia 70c. per ton. A number of producers sold at or below 60c. per ton, and yet made a profit. This result is due to large production, great economy in administration, large beds, and generally favorable conditions; but,

with all that, it is certain that the prices obtained were generally unprofitable. While there was no widespread and general strike, there were many local troubles, and constant discussion as to wages, and a number of reductions in wages were made toward the close of the year and in the early months of 1897. Where the output of coal has been decreased it has usually been done, not by reducing the working forces, but by running short time; so that in many districts the earnings of the miners have been very small and their condition very unsatisfactory.

Foreign Coal Trade.—The imports and exports of coal in the United States are shown in the following tables. The amounts are insignificant in comparison with the production. The exports are chiefly to Canada, though there is a moderate trade with the West Indies and some South American countries. The imports were largely at California ports, the greater part coming from British Columbia, though Australia furnished some of them, and small quantities came from Great Britain and Japan. The only coal imports in the East were from

IMPORTS OF COAL AND COKE INTO THE UNITED STATES.

Year.	Coal.								Coke.		
	Anthracite. (b)			Bituminous.			Total.				
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Metric Tons.	Value.	Short Tons.	Metric Tons.	Value.
1892..	65,058	66,099	$197,583	1,143,304	1,161,597	$3,744,862	1,227,696	$3,942,445	24,482	22,210	$86,350
1893..	53,768	54,628	148,112	1,082,993	1,100,321	3,623,892	1,154,949	3,772,004	33,165	30,080	99,683
1894..	90,068	91,509	234,024	1,242,714	1,262,597	3,785,513	1,352,665	4,019,537	29,137	26,427	70,358
1895..	141,337	143,598	328,705	1,212,023	1,231,415	3,626,623	1,375,013	3,955,328	29,622	26,867	71,366
1896.	102,566	104,212	.....	1,246,991	1,266,948	3,488,115	1,371,155	.....	43,372	39,347	114,713

EXPORTS OF COAL—DOMESTIC PRODUCTION.

Year.	Anthracite.		Bituminous.		Totals.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1892	851,639	$3,722,903	1,645,686	$4,999,289	2,497,325	$8,722,192
1893	1,334,287	6,241,007	2,324,591	6,009,801	3,658,878	12,250,808
1894	1,440,625	6,359,021	2,195,716	4,970,270	3,636,341	9,995,362
1895.	1,470,710	5,937,130	2,211,983	4,816,847	3,682,693	10,753,977
1896....	1,350,000	5,925,506	2,276,202	5,072,818	3,626,202	10,998,324

PRODUCTION AND CONSUMPTION OF COAL IN THE UNITED STATES. (IN LONG TONS.)

Year.	Production.	Imports.	Total Supply.	Exports.	Consumption.	
					Long Tons.	Metric Tons.
1892.....	160,992,414	1,208,362	162,200,776	2,497,325	159,703,451	162,258,706
1893	157,364,399	1,136,761	158,501,160	3,658,878	154,842,282	157,319,758
1894..... ..	151,674,804	1,332,782	153,007,586	3,636,341	149,371,245	151,761,185
1895.....	169,013,968	1,353,360	170,367,328	3,682,693	166,684,635	169,351,589
1896..... ..	166,286,849	1,349,557	167,636,406	3,626,202	164,010,204	166,634,367

Nova Scotia, the mines of that province sending a small quantity to New England; but this is necessarily limited, because the Nova Scotia coal cannot compete with our own supplies either in quality or price.

Coke.—The production of coke is shown in the accompanying tables by States. The total in 1896 was 10,359,584 short tons, showing a decrease of 2,133,880 tons,

or 17.08%, as compared with 1895. The coke trade is so closely involved with and dependent upon the iron industry that this decrease was to be expected. It was proportional to the decrease in iron and steel production.

Pennsylvania maintained its lead in coke production, and will probably continue to do so for some time to come. It is possible, however, that the next five years will see considerable changes, and some of the large operators are already preparing for the exhaustion of the coal measures of the Connellsville field by securing tracts of coal land in West Virginia and Kentucky.

Coal Production of the World.—The following table shows the coal production of the world. In the United States and nearly all the European countries the figures for 1896 are from official returns, or have been collected especially for THE MINERAL INDUSTRY.

Almost everywhere but in the United States there was a gain in production, and all over Europe the demand for coal showed an increase, the only exception being in Great Britain, where production was stationary, and there were some complaints, chiefly because of a smaller export trade.

COAL PRODUCTION OF THE WORLD. (IN METRIC TONS.)

Year.	Africa.	Australasia.					Austria-Hungary	Belgium.	Canada.		France.	Germany.
		New South Wales.	New Zealand.	Queensland.	Tasmania	Victoria.			Alberta, British Columbia	N. Brunsw'k, Nova Scotia.		
1892	250,000	3,841,842	684,155	269,354	36,243	30,445	29,225,004	19,583,173	1,006,721	1,974,059	26,178,073	92,544,050
1893	300,000	3,330,939	691,548	268,260	43,893	93,202	26,548,604	19,407,251	925,298	2,556,085	25,738,073	95,476,208
1894	600,000	3,730,829	745,000	275,036	31,052	174,407	26,905,490	29,534,501	1,327,259	2,168,340	27,416,905	101,485,857
1895	700,000	3,500,000	760,000	275,000	33,000	200,000	27,250,000	20,414,849	1,403,911	1,782,732	28,236,039	103,876,813
1896	1,250,000	3,350,000	730,000	240,000	31,500	195,000	28,125,000	21,250,000	1,072,589	2,324,175	28,870,091	112,437,741

Year.	India.	Italy.	Japan.	Russia.	Spain.	Sweden	United Kingdom.	United States.	All Other Countries.	Totals.
1892..	2,578,299	295,713	3,199,763	6,325,020	1,461,196	199,380	184,713,640	163,657,988	1,500,000	539,554,118
1893..	2,570,332	317,249	3,328,879	6,560,320	1,531,810	199,933	166,971,440	159,919,176	1,500,000	518,456,180
1894..	2,620,910	271,294	3,500,000	7,498,000	1,659,264	213,634	188,277,525	154,211,308	1,600,000	545,246,611
1895..	2,650,000	250,000	3,650,000	7,551,180	1,774,560	205,000	194,350,604	171,727,881	1,750,000	562,341,569
1896..	2,725,000	235,000	3,715,000	7,785,000	1,874,771	210,000	195,272,000	168,957,264	1,800,000	583,450,131

Great Britain continues to be the chief exporter of coal; the foreign sales of British coal in 1896 reached a total of 34,262,077 long tons, or 17.8% of the total production. The increase in exports over 1895 was 1,160,625 tons. No other European country had a large foreign trade in coal, the exports reported from Germany, Austria, and Belgium being largely balanced by imports; this trade, moreover, is in great part with countries immediately adjoining, and is simply an exchange over short distances by rail. Thus certain districts in Germany find it most convenient to draw their supplies from Bohemia, and some of the Luxemburg furnaces use Belgian coal.

The mines of Australia and Japan now supply the Eastern markets very largely, and quite a sharp competition exists between the two countries. The Japanese are making arrangements to work the coal deposits of Formosa, and propose to introduce there and in their home mines the latest improvements in mining methods and machinery.

The Anthracite Coal Trade in 1896.

The opening of the year 1896 found the anthracite coal trade in a condition that in many respects can be characterized as desperate. The year preceding had been one of trials and tribulations, which, however bitter, were of benefit to the producing interests in at least one particular—they furnished experience that indicated the necessity of radical changes in the manner of conducting business affairs. Though experienced men had been at the helm and wise leaders had given direction that, if properly obeyed, would have averted much of the disaster that did occur, the danger of total ruin was imminent at the opening of the year 1896. The trade was beset with undercurrents and false trade winds that carried the ship and its crew in every direction but that in which they desired to go, yet they were compelled to drift, because they had themselves launched their craft upon this sea of trouble.

Put in a nutshell, there are two stubborn conditions which have served to demoralize the anthracite coal business. These are a limited market and a practically unlimited ability to produce.

Like all new materials put upon the market, anthracite coal in its early days had to win its way into the class of commodities which, once used, are used again, and which are first desirable, then needful, and finally indispensable. Anthracite coal in this way came to be looked upon as an absolute necessity which nothing else would be able to supplant. As in many another instance, this sense of security was destined to be rudely shaken. Bituminous coal came to share the field and compete with anthracite for a part of the patronage the latter was enjoying. From the very start bituminous coal had an advantage in its abundance and lower cost of production, and in many cases was on an equal footing with anthracite in the efficiency of results obtained from its use. Mechanical improvements and the gradual perfection of processes in the arts naturally tended to widen the possibilities of the cheaper and more abundant fuel at the expense of the dearer one. Not only, therefore, did anthracite coal fail to win its way further into the essential needs of the times, but it was supplanted by bituminous coal where before it had enjoyed a monopoly. We hear anthracite coal spoken of as a domestic fuel, and this is the field which competition with bituminous coal has forced it largely to occupy to-day, its use as a steam coal having very greatly decreased. So much, briefly, on the question of limited market.

There has never yet been a field which offered good returns for those who were at work in it but that they were joined by others who desired to share the profits, the result being an overcrowding of the field. This has been the experience of the anthracite coal market, in which increase in numbers, however, is not so important a factor in this instance as the ability to produce among those already in the field. Mechanical genius not only served to increase the use of bituminous coal at the expense of anthracite, but it was busy devising apparatus by which the output of the collieries could be greatly increased. Ten years ago an anthracite breaker that could in one day prepare 1000 tons of coal for market was considered a wonder. To-day we have breakers that can without difficulty prepare 1500, 2000, and even 2500 tons of coal in a day of 10

hours. The total output, however, has not increased in the same proportion, for in 1885 it was 31,750,546 tons, while 10 years later, in 1895, it was 46,545,760 tons. And it must be remembered that the latter tonnage was reached when the market was exceedingly sluggish. Coal went begging, had to be disposed of at a sacrifice, and the mines were kept working little over half time.

The advent of 1896 found the anthracite coal interests fully cognizant of their shortcomings during the year just closed, and equally conscious of the fact that "little market" and "much coal" was at the bottom of it all. The question was: "What shall be done to improve matters?" The answer they gave is found in the following brief summary of the year's events:

The Market During 1896.—January opened with such dull market conditions that prices had a tendency to decline. Orders were few in number and small in size, and quotations for the various sizes were: $3.35@$3.40 for stove, $3.10@$3.20 for egg and chestnut, and $2.90@$3.00 for broken, all net on board. Although rumors were abroad that a meeting of presidents was to be held to outline a policy for the year, such rumors were denied by various officials of the producing companies. So far as the public knew, the go-as-you-please methods of carrying on the trade were to be continued. The middle of the month found the demands from buyers as poor, and even worse, than at the beginning, with the result that prices were 10c. to 15c. lower per ton. Just at this time came the first ripple of excitement in the trade. Presidents Samuel Sloan, of the Delaware, Lackawanna & Western; J. Rogers Maxwell, of the Central Railroad of New Jersey, and E. P. Wilbur, of the Lehigh Valley, issued a call to the other coal-carrying roads for a conference to discuss the condition of the coal trade and arrange for a restoration of prices. This announcement had little effect upon the volume of business transacted during the week following its publication, buyers apparently being confident that no immediate change would occur to increase prices. Quotations at the end of the week proved the correctness of their belief. At the meeting of the presidents above mentioned, it was decided to allot to each producing company a certain proportion of the entire output, but when it came to determine these proportions upon the output figures of 1895, there was a disagreement, it being claimed by some that those figures did not fairly represent their capacity. It was decided to appoint a new committee, consisting of President J. Rogers Maxwell, of the Jersey Central; President E. B. Thomas, of the Erie, and Vice-President E. R. Holden, of the Delaware, Lackawanna & Western, to recommend new percentages and other plans to improve trade for the sellers. Up to two days before the close of the month the following were the ruling prices for coal: Stove, $3.20@$3.25; egg, $2.95@$3; chestnut, $2.85@$2.95; broken, $2.75, all net on board. On January 30 the presidents of the coal-carrying companies held a meeting at which the report of the previously appointed committee of three was submitted and adopted. The various companies were allotted the following percentages of the total output: Philadephia & Reading, 20.50; Lehigh Valley, 15.65; Delaware, Lackawanna & Western, 13.35; New Jersey Central, 11.70; Pennsylvania Railroad, 11.40; Delaware & Hudson, 9.60; Erie Railroad, 4.00; Pennsylvania Coal Company, 4.00; Dela-

ware, Susquehanna & Schuylkill, 3.50; New York, Susquehanna & Western, 3.20; New York, Ontario & Western, 3.10.

In addition to agreeing upon percentages for 1896, a standing committee of five, consisting of Presidents Maxwell, of the Jersey Central; Harris, of the Philadelphia & Reading; Wilbur, of the Lehigh Valley; Thomas, of the Erie, and Walters, of the Delaware, Susquehanna & Schuylkill, was appointed to formulate a plan for carrying the anthracite coal under a plan similar to that adopted for the freight traffic of the Trunk Line Association. On January 31, the sales agents, acting under instructions from the presidents, held a meeting and decided on the following tidewater prices: Stove, $3.60; egg and chestnut, $3.35; broken, $3.10—an advance of 35c. to 40c. per ton.

February opened with absolutely no new business doing, purchasers being dazed at the action just taken by the coal-producing companies. In addition the weather was extremely mild, which in itself would have made a slow market. The output for the month had been fixed at 2,500,000 tons, a remarkable restriction considering that between 4,000,000 and 5,000,000 tons had been mined the previous month. The sales agents sat quietly by waiting for orders, but no inducements were considered looking toward the shading of prices below the circular. A few of the individual operators, however, accepted orders at figures 15c. to 20c. below the circular, because, as they said, they had not sold as far ahead at the old rates as the large companies did. As the month advanced business did not increase very materially, but the market acquired an air of firmness that had been unknown to it before. About the middle of February colder weather prevailed for a time, and though it did not last long enough to create new business, it did influence the market so that firmer conditions resulted. This was in part also produced by the general observance of the agreement to restrict the output to 2,500,000 tons. The month ended with mild weather, and dealers continuing to hold off from purchasing, in the hope that conditions would take a turn more favorable to them, on what little business was done the circular prices were obtained.

During March the trade conditions settled into a state of quietude that remained unbroken throughout the month. The first difference of opinion was noticeable at the beginning of the month among the various interests since the "understanding" was reached, but it was of only slight moment. It was on the question of output; the figures given by different sales agents did not agree. Later developments showed that this matter was properly taken care of, the restriction being guided by the market requirements. In this way stocks of coal in hand were not increased, but, on the contrary, some companies materially reduced theirs. Sales were reported rather small, but on all of them the circular prices were said to be realized.

April conditions of the trade differed but little from those existing during March. Business was not large, but prices were firm. Public opinion and the passage of the Robbins Anti-Coal Trust Bill put an end to the meetings of sales agents to recommend restrictions and make other suggestions. Opinions, informally expressed, placed the probable output for the month at 3,000,000 tons. All matters of this sort were from this time forward reported as having been arrived at through "informal talks" that naturally took place between the

heads of the interests involved. A week before the close of the month rumors were general that prices were to be advanced on May 1, the representatives generally expressing themselves in favor of such a step.

May saw the anticipated advances in the schedule of prices, the circular generally taking effect upon the opening day. The advance was 25c. per ton on all prepared sizes, which made the cost of stove coal $4; egg and chestnut, $3.75; and broken, $3.50 per ton. This action was, of course, followed by a dearth of orders, shipments being mostly on orders placed during the previous month. By the end of May it was reported that good companies were able to buy at a net rate of $3.85 for stove, $3.60 for egg and chestnut, and $3.35 for broken, all net on board, the full rates obtaining only on small orders. The production for the month was 3,125,170 long tons.

In June a new element came into the anthracite coal trade, to further depress general conditions, in the shape of the political question. Business was rather quiet during the entire time, and the only point of special interest was the rumor of another advance on July 1. This was announced by the Philadelphia & Reading Company on June 24. The circular rates then became: Stove, $4.25; egg and chestnut, $4, and broken, $3.75. The tonnage for the month was 3,314,196 tons.

July and August were months of midsummer quietude, in which both producer and retailer were as inactive as they well could be without going out of business entirely. The consumption of coal for all purposes fell off very much, which kept the distributing end of the business from warming up over its work, while the prospective 25c. advance, which the producing interests still had up their sleeves, together with the unkind reception which they knew from past experience it would receive, made them wary; not to speak of the assistance given by the August weather. Each month, however, the output increased at a time when market conditions warranted a decrease.

September opened with another new circular, prices having been advanced 25c. a ton on all domestic sizes, becoming $4.50 for stove, $4.25 for egg and chestnut, and $4 for broken. A more active business was reported at this time, principally on orders that had been placed at the July circular of prices before the end of August. With these filled, a reaction set in, and the rest of the month was very quiet. About this time it became generally known that the circular prices were not being realized, and had not been at any previous time as fully and as promptly as the public had been led to suppose. Concessions of all kinds were made to effect sales, and the "understanding" that had been governing the trade since February had developed into a "misunderstanding." In the meantime the production still tended upward, the September output reaching, in round numbers, 3,750,000 tons. October and November conditions, in the main, were a continuation of those in September. The usual fall trade did not materialize because the weather remained uniformly mild and because business was generally depressed. Regardless of these unfavorable circumstances, the output for each month approximated 4,500,000 tons, which amount was fully 1,000,000 tons above what it was to the interest of the producers to mine. No wonder the circular prices could not be obtained, and the trade developed into a condition characterized as "very discouraging."

General Observations.—The year as a whole can, without fear of contradiction, be put down as an improvement over 1895. There was more method in the conduct of affairs, which, in turn, resulted in better prices and, though the total output for the year fell short of the 1895 output by about 3,000,000 tons, the gross receipts resulted in greater net earnings. The "understanding" that the producers arrived at in February promised better things than were fulfilled later in the year, yet this was not altogether the fault of the "understanding." So far as is known, however, the May, July, and September circulars were decided upon at the February meeting and carried out according to programme, when it must have been suggested to most minds that it would have been better to postpone the July circular to September and omit the September circular entirely. The practical results that were obtained by the changes made did not even equal those that would have been required by the modified schedule as suggested.

The matter of tonnages also does not seem to have been regulated with proper judgment. But this feature of the business had in the past been always so grossly in error that it is not to be wondered at that the reforms that were instituted were not carried out entirely. When the figures for the first six months of 1896 showed a decrease in the output of more than 1,750,000 tons compared with 1895, the restraint apparently could no longer be borne, and the dull summer season saw the tonnage steadily increase a quarter million tons each month. But, worst of all, the fall trade, on which the fondest hopes of the year had been based as the time when past shortcomings could be equalized, proved little better than the summer had been. Instead of meeting these conditions with a limited output, the market was given a quantity of coal far in excess of all requirements. Without doubt it was one of the greatest errors of the year.

In a word, the conditions of the trade are not materially changed and have been those which the past would lead us to expect. What these are can be understood by a careful reading of the article on the "Evolution of the Anthracite Coal Trade," in Vol. IV. of THE MINERAL INDUSTRY.

Production.—The anthracite coal-mining and carrying companies have for a number of years maintained a statistical bureau in Philadelphia, which formerly issued monthly statements giving the shipments from the different regions for the information of all concerned. These reports were of material service to the trade, and their regular publication continued for a number of years without objection from any one. Early in 1895, however, the bureau ceased to issue reports, in pursuance of the policy of concealment which all the coal companies seem to have adopted, and this ill-judged course has been closely adhered to.

Notwithstanding the decrease in shipments, that in sales and deliveries for the year was still greater. No report of the actual stocks at tidewater and at Western delivery points was made, but the reports of all the companies published for 1896 show large increases in the "coal on hand."

It has been claimed that the coal consumed at and about the mines amounts to 11% of the shipments, but this is undoubtedly too high, and 7% is more generally accepted; the real figure is probably not much over 6%. Accepting the 7%, however, the actual production for 1896 would be 46,199,907 long tons,

equivalent to 51,743,896 short tons, or 46,939,107 metric tons. The value of this product at the mines was about $93,140,000.

The following table shows the shipments of the different companies for 1896, which is given in comparison with the returns for 1895. The shipments are in long tons of 2240 lbs. as usual with anthracite; the final column shows the percentage allotted to each company for 1896 by agreement.

ANTHRACITE COAL SHIPMENTS BY COMPANIES.

Company.	1895.		1896.		
	Tons.	Per Cent.	Tons.	Per Cent.	Allotment
Philadelphia & Reading	9,905,059	21.47	9,019,533	20.89	20.50
Lehigh Valley	7,360,454	15.81	6,749,128	15.63	15.65
New Jersey Central	5,388,194	11.57	4,999,003	11.58	11.70
Delaware, Lackawanna & Western	6,129,260	13.16	5,627,533	13.03	13.35
Delaware & Hudson	4,347,843	9.34	4,152,273	9.52	9.60
Pennsylvania Railroad	5,025,645	10.79	4,752,120	11.06	11.40
Pennsylvania Coal Company	1,746,832	3.75	1,728,972	4.05	4.00
Erie Railroad	1,820,038	3.91	1,718,262	3.98	4.00
New York, Ontario & Western	1,424,407	3.03	1,321,895	3.06	3.10
New York, Susquehanna & Western	1,492,244	3.06	1,410,080	3.27	3.20
Delaware, Susquehanna & Schuylkill	1,905,784	4.11	1,698,684	3.93	3.50
Totals	46,545,760	100.00	43,177,483	100.00	100.00

An examination of the table will show that the companies generally kept quite closely to their allotment. In January there was no agreement existing and a very large quantity of coal was mined, each company shipping freely, and in February, when the agreement or understanding took effect, there were many existing discrepancies to be made up later. This was generally done, and the variations from the division shown by the actual shipments were very small. This was the more remarkable because of the indefinite nature of the understanding. For reasons well understood, no formal agreement was entered into either then or later. That companies which had heretofore shown themselves quite willing on occasion to disregard written contracts should thus respect an informal one is certainly worthy of note. It is fair to state that the Delaware, Susquehanna & Schuylkill—better known to the trade as Coxe Brothers & Company—refused at the beginning to accept the 3.5% allotted, and continued to work through the year on the basis of the 4% to which the company believed itself entitled. The only other company exceeding its allotment by any considerable proportion was the Reading.

In order to compare the total output for a series of years the shipments to market for 12 successive years are given below.

ANTHRACITE SHIPMENTS FOR TWELVE YEARS.

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1885	31,623,529	1888	38,145,718	1891	40,448,336	1894	41,391,198
1886	32,136,362	1889	35,407,710	1892	41,893,320	1895	46,545,760
1887	34,641,017	1890	35,855,174	1893	43,089,536	1896	43,177,483

Although the shipments in 1896 showed a decrease of 3,368,277 tons or 7.4%, 1895, they were larger than those of any year previous to 1895.

As it is estimated that about two tons of coal are wasted—either left as pillars in the mines or going into the culm piles—for each ton marketed, the year witnessed a reduction of about 138,600,000 tons in the anthracite reserves

The Coal Trade of Buffalo in 1896.

The anthracite coal trade of Buffalo for 1896 was without any particular features of importance. Prices advanced 60c. from opening quotations to the close at wholesale per gross ton, and 25c. at retail per net ton delivered. The receipts by railroads for the year fell off about 200,000 net tons, and the shipments by lake 220,700 net tons. The movement by canal eastward and westward was very light. Lake freights on coal fluctuated from 20c. to 60c. to Chicago and Milwaukee, and from 20c. to 50c. to Duluth and Lake Superior ports; the average rate to Chicago was 39c.; to Milwaukee 37c.; and to Duluth 27c. per net ton free on and off. Trade was only moderate all the year, as so many citizens use natural gas for fuel, and the range of temperature during the winter months was comparatively high.

There was more or less trouble during the year in consequence of dealers cutting prices, in spite of the efforts of the Coal Exchange to prevent it. However, trade was done at a profit—an improvement over 1895 in this respect.

The use of electricity lessened materially the consumption of illuminating gas. At the beginning of December 1000 horse power of electricity was received daily from Niagara Falls, the precursor of a larger supply.

The bituminous coal trade was of only moderate volume, as the consumptive demand was smaller for manufacturing and vessels' use; the supply during the year was plentiful and the assortment good. Railroad companies reaped the benefit of higher freight rates, but operators, dealers, and consumers were not benefited to any extent. Why the special low rate to Canada (15c. less than to Buffalo) was made and put in force was a conundrum to the trade. No labor disturbances of any consequence interfered with the trade in 1896 as far as Buffalo was concerned, and the supply of cars was fully adequate to the requirements of business. Stocks at close were only moderate on railroad sidings and coal on the docks was about all cleared up on the close of navigation. Standard brands of Pittsburg coal sold as low as inferior brands from other districts at times.

The coke trade was quiet early in the year, but improved materially during the latter half.

It may not be generally known, but the retail coal dealers of Buffalo draw their supplies direct from the companies' trestles, thereby saving the expense of keeping yards. The dealers do their own carting.

Natural gas is still extensively used for fuel purposes; the supply is mainly piped from Canada and Pennsylvania, with many small wells within 10 miles of Buffalo. This accounts for the decreased consumption of coal for several years, when it should have increased according to the increase in population.

Fifty-four years have passed since the first record was made of the receipts of anthracite coal at Buffalo. In 1842 only 900 net tons were reported; in 1852, 23,000 net tons; in 1862, 132,500 net tons; in 1872, 521,000 net tons; in 1892, 4,804,700 net tons. Before 1882 statistics of the bituminous receipts are missing, but in that year only 65,000 net tons were received by railroads and lake; 10 years after, in 1892, 2,280,470 net tons were reported.

The following statistics collected by Mr. William Thurstone, secretary of the Buffalo Merchants' Exchange, shows the coal trade for five years:

COAL TRADE AT BUFFALO FOR FIVE YEARS.

	1892.	1893.	1894.	1895.	1896.
Receipts—					
Anthracite, canal	54,760	70,546	42,130	14,038	38,061
Anthracite, rail	4,750,000	4,700,000	4,230,000	4,750,000	4,550,000
Bituminous, lake	.....	.....	.....	240	.....
Bituminous, rail	2,627,441	2,896,614	2,280,470	2,727,308	2,500,000
Blossburg, rail	25,000	25,000	25,000	25,000	25,000
Shipments—					
Anthracite, canal	.....	.....	.....	1,376	1,104
Bituminous, canal	29,216	19,336	8,840	4,259	956
Anthracite, lake	2,822,330	2,681,173	2,475,255	2,612,768	2,391,568
Blossburg, lake	5,000	7,500	7,500	6,000	6,000
Bituminous, lake	25,000	15,000	2,500	2,000	2,500

The nominal or list prices of anthracite coal for 1896 are given below:

LIST PRICES OF ANTHRACITE COAL AT BUFFALO IN 1896.

Date.	Per Long Ton, f. o. b. Buffalo.		Per Long Ton, on Cars at Buffalo or Suspension Bridge.		Per Short Ton, at Retail in City Limits.			
	Grate.	Stove, Egg and Chestnut.	Grate.	Stove, Egg and Chestnut.	Grate.	Stove, Egg and Chestnut.	Pea.	Blossburg.
January 1	$4.45	$4.70	$4.15	$4.40	$4.75	$5.00	$3.75	$4.00
June 1	4.55	4.80	4.25	4.50	.....	.....	.....	.....
July 1	4.80	5.05	4.50	4.75	.....	.....	.....	.....
September 1 to close	5.05	5.30	4.75	5.00	5.00	5.25	4.00	4.00

On bituminous coal the range of prices during 1896 as purchased by gas works, manufacturers, propeller lines, tugs, etc., in car lots, was from $1.90 to $2.35 for gas, $1.50 to $1.75 for run of mine, $1.70 to $1.85 for lump, $1.20 to $1.35 for slack and $1.80 to $1.90 for lump per net ton on track. At retail for family use the prices were from $4 to $5.50 per net ton delivered within city limits. The closing price of Briar Hill lump at wholesale was $3.25; No. 1 Falling Rock cannel at wholesale $5.25, and other cannels $3.75 per net ton in car lots on track. Coke was quoted for the last six months at $4 for Connellsville and $2.65 for Reynoldsville per net ton in car lots on track.

The anthracite coal brought to Buffalo comes by five lines of railroad: The Delaware, Lackawanna & Western brings 25%; the Erie, 23%; New York Central, 18%; Lehigh Valley, 23%; Western New York & Pennsylvania, 9%.

There are seven shipping docks and coal pockets at the port of Buffalo; total average shipping capacity, daily, 27,500 tons; the average capacity of the pockets, daily, 36,800 tons. Outside the city limits at Cheektowaga is the stocking coal trestle of the Delaware, Lackawanna & Western, with a capacity of over 100,000 tons storage. At the same place the Lehigh Valley has its trestles and stocking plant of 175,000 tons storage capacity, with a shipping capacity of 3000 tons daily; and has a transfer trestle for loading box cars, with a capacity of 100 cars daily. And at the same point the Erie has a stocking plant, with average daily capacity of 1000 tons, and storage capacity of 100,000 tons. The Reading has in the city a large trestle and pocket for the convenience of the retail trade, and in connection with the docks, with a capacity of 2000 tons. The Buffalo, Rochester & Pittsburg has terminals fronting on the Blackwell Canal, with a water frontage of 1100 ft.; also a town deliv-

ery yard, with a hoisting plant for loading and coaling vessels, used by Coxe Brothers & Company.

The principal ports of destination received from Buffalo in 1896 the following quantities by lake: Chicago, 1,005,000 net tons; Milwaukee, 648,000 net tons; Duluth, 232,000 net tons; Toledo, 88,100 net tons; Manitowoc, 36,500 net tons; Gladstone, 30,000 net tons; Ashland, 11,000 net tons; Superior, 192,000 net tons; Marquette, 13,500 net tons; Fort William, 12,500 net tons; Lake Linden, 12,000 net tons; Racine, 24,000 net tons; Green Bay, 26,000 net tons; Kenosha, 9000 net tons; Bay City, 15,000 net tons; Saginaw, 13,500 net tons; Detroit, 3150 net tons; Hancock, 5200 net tons; Marinette, 1750 net tons; Washburn, 3500 net tons; Michigan City, 2200 net tons; Port Arthur, 2100 net tons; Menominee, 2100 net tons; Canadian ports, 6000 net tons; Sheboygan, 1250 net tons; Benton Harbor, 1200 net tons; Portage, 1100 net tons, and Port Huron, 1150 net tons.

The Chicago Coal Market in 1896.

Anthracite.—There is nothing to be said in favor of the anthracite coal trade at Chicago during the year 1896. From the very beginning of the year to its end not one real staying boom made its appearance, and the buying of coal was wholly in accordance with the weather. There were weeks of very cold weather in which considerable coal was placed, but when the cold subsided the market conditions became exceedingly quiet. Shipments to out-of-town places ran a great deal below the average, and it is evident that country dealers carried stocks greatly below their usual limit. The amount of hard coal shipped to the city decreased much from the preceding year, through the action of the coal agents in limiting output of mines. There was, however, plenty of coal on the docks, railroads, etc., of the city during the entire year, but the limited receipts reduced the question of demurrage, which was one of the features of 1895. The railroads running north and west from Chicago contended for the privilege of carrying the limited amount of hard coal to points on their lines, and consequently most of the year there was cutting of rates, which finally resulted in a rate of $1 per ton to Missouri River points. This did not hold very long, however, for the roads found it a losing venture and it was soon back to about $2.50 per ton, where it has been since. The shipments of hard coal from Buffalo to Chicago by lake footed up about 960,000 tons, an increase of 20,000 tons over the shipments of 1895. The amount of coal shipped from Buffalo to Chicago in 1895 by water would undoubtedly have been more had it not been for the scarcity of boats, the greater inducements offered in carrying iron ore having made big inroads on the usual coal-carrying fleet. During the opening months of the year hard-coal prices were greatly demoralized, and to obtain business it is known that $4.25 was quoted. Circular rates in January were in the neighborhood of $5 per ton, but for a number of months there were no authorized rates noticed. May circular prices were placed at $5@$5.25; prices advanced in June to $5.10@$5.35. August circular rates increased to $5.35@$5.60, and in September they took another jump to $5.60@$5.75, at which figures they stopped for the remainder of the year. The retail price from that time was $6.50@$7.

Bituminous.—The bituminous coal trade was very poor for the first six months of the year, but on account of the advance in hard coal prices the latter six months showed a considerable increase in volume of sales. There was a large consumption of soft coal by those who have heretofore used hard, the increased cost of anthracite driving them to the use of soft coal. The saving in cost was so large as to lead consumers to put up with the disagreeable features of soft coal. The manufacturing industries bought almost entirely on the hand-to-mouth policy and but few important contracts were booked the entire year. A very large amount of soft coal was shipped to this market and prices were very low. Hardly once during the 12 months of the year was there any real steadiness in them, except toward the end of the year. The better grades of soft coal came to the front largely during the year. Some of the closing prices of the year on bituminous coal were: Pittsburg, \$2.90@\$3.15; Raymond, \$3; Wellston, \$3; Indiana Block, \$2.05; Hocking Valley, \$2.60· Youghiogheny, \$2.85; Blossburg, \$3.65; Piedmont, \$3.15 per ton.

The Pittsburg Coal Shipments by River in 1896.

The Pittsburg coal shipments by river are something very uncertain, having to depend on the stage of water and the weather, which all persons know is not very reliable. We have known of coal shipments every month in the year; this is something very rare and does not happen often. Again there have been years in which there were shipments in May and no others until December. The year 1896 was not a favorable one for the coal interest; the shipments were liberal, but low prices prevailed in Western and Southern markets. The entire year coal sold in Cincinnati as low as 4¾@5½c. per bu.; these are ruinous figures, but were the best that could be obtained. A large supply of Alabama coal is now furnished to the Southern markets; although it is said to be inferior to the Pittsburg article, since consumers make it answer it is sold cheaper.

The following table shows the shipments of coal from Pittsburg by the Ohio River for 14 years, in tons of 2000 lbs.:

SHIPMENTS OF COAL FROM PITTSBURG BY THE OHIO RIVER.

Year.	Cincinnati.	Louisville.	Total.	Year.	Cincinnati.	Louisville.	Total.
	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.
1883	1,261,320	2,258,480	3,519,800	1890	1,304,640	2,042,160	3,346,800
1884	985,240	1,232,040	2,217,280	1891	1,125,000	1,931,600	3,056,600
1885	1,303,600	1,693,360	2,996,960	1892	973,560	1,519,960	2,493,520
1886	1,329,160	1,537,406	2,866,566	1893	879,950	1,617,840	2,497,790
1887	830,800	1,438,920	2,269,720	1894	1,139,920	1,383,280	2,523,200
1888	2,053,560	2,340,520	4,394,080	1895	984,400	1,384,080	2,368,480
1889	1,214,400	1,515,800	2,730,200	1896	2,029,760	2,578,120	4,607,880

The total in 1896 was the highest reached in any of the 14 years, as that for 1895 was the lowest, with the exception of the dry years 1884 and 1887. The total increase in 1896 over 1895 was 2,279,400 tons, or 96.2%. As compared with 1888, the second best year recorded, the increase shown was 213,800 tons, or 4.8%. It is not likely that the river trade will ever exceed this maximum; most probably it will never reach so high a point again, if we consider the competition Pittsburg must meet in all the river towns.

Present Practice of Coal Washing in Germany and England.

By Richard Cremer.

Fifty years ago coal washing was practically unknown, although the apparatus used in metalliferous ore dressing had attained a fairly high degree of perfection. The large lumps of coal were picked out by hand, while the small sizes were separated by fixed screens of the plainest construction. The coal received no other preparation. In our times, however, we try to get all possible value out of the product of our mines, and the success of many, especially those which produce fat coals, depends upon such economy. Naturally we find the most approved appliances for cleaning and sizing coal in the districts where the product has the largest percentage of impurities, and in particular where the product is used chiefly for coke-making, and in smelting. The most rapid progress has consequently been made in Belgium, Saxony, and Westphalia, where coals high in ash are found; while England is comparatively backward, due doubtless to the great supply of coal that England possesses, its superior quality, and the fact that the export trade in coal is chiefly in English hands and will take what is offered even if the quality might be improved. The more part of the English collieries are, besides, small in extent, and under these circumstances we can understand the reluctance of their owners to expend large sums for new and expensive machinery.

Yet it cannot be denied that English managers have made rapid progress during the last 10 years, and a good many English collieries are now provided with as good coal-washing machinery as the best of the continental, while numerous plants are in course of erection, many of them in the best style, and others are being planned, so that in a few years England will doubtless be as far advanced as Germany.

Many different improvements in coal-washing machinery have been introduced during the last decade, yet we find in all essentially the same principles as in the metalliferous ore-dressing. In projecting a first-class plant for coal washing we have to consider: (1) A separation of the lump coal requiring no further treatment; (2) sizing; (3) washing; (4) storage of products, and apparatus for loading and drainage. The separation of the larger sizes is effected by dry sorting; the smaller sizes from which impurities are to be removed are subjected to a wet process.

In many collieries in England and Germany breakers are used to reduce the lumps of coal, either to make sizes required by the market, or to detach pieces of slate, etc., which must be picked out. English collieries employ for this purpose the old-fashioned "coal-mill," but in Germany these have been replaced generally by geared steel rolls. Disintegrators are used in England and Germany, generally those of the Carr type, which is superior to all others. This disintegrator can also be used for mixing various coals.

The trucks used to convey the coal to the breaker or washer in England are usually head-dumpers, in which the box is turned on its transverse axis. This manner of dumping does not compare unfavorably with other arrangements, although the coal is exposed to a somewhat greater chance of breakage than with the circular dumper, which has been introduced in Germany, in which the car locked on a geared platform is inverted bodily.

The sizing of the coal used to be done in Germany by fixed bar-screens and revolving drum-screens, but modern practice has found them inadequate to perform the work demanded, and though they are still to be found in England, especially at small collieries, which are not provided with washers, on the continent they have been supplanted by movable screens, shaking, gyratory, and the so-called Briart grate and its modifications. Kuepper has made a valuable improvement on the Briart grate which avoids crushing the coal and the passage of flat pieces, while retaining the advantages of short length, slight angle of

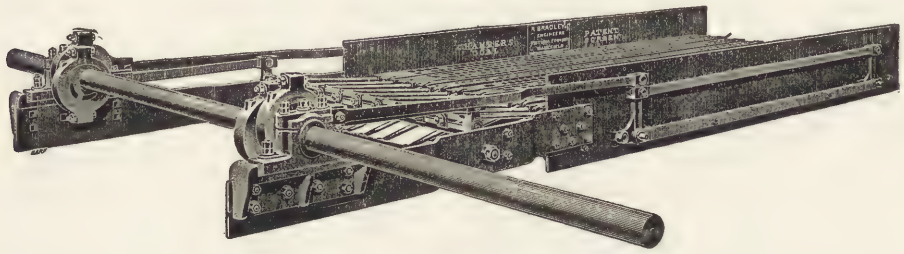


FIG. 1.

inclination, and great capacity. This apparatus consists of two interlocked, reciprocally-moving grates made of ⌐¬ iron, with 80 mm. holes, which are operated by eccentrics. It is set at an angle of 8° to 12°, and so long as the movement is slow and regular the separation is excellent.

A similar apparatus is in use at the Denaby & Cadeby Main Collieries in England. This one (Fig. 1) can be made of any desired length, and can also be used as a picking-table. It is provided with bars of various diameters of holes, which can be exchanged in two minutes.

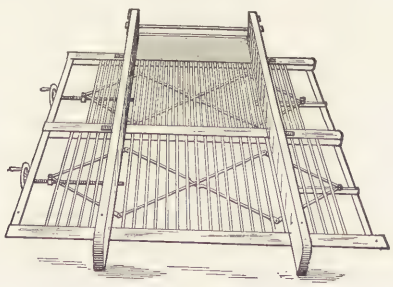


FIG. 2.

A new style of grate, invented by Swift, has been introduced recently by the Ince Iron Works, Wigan, England. This permits of a rapid change of space between the bars, which are controlled by screws at one side of the grate, as shown in Fig. 2.

Frequently, especially if a greater angle than 12° is available, shaking-riddles are used. Messrs. Coulson & Co., Spannymoor, have constructed one of this type, used at the Springwell Colliery and elsewhere, which can be used as a picking-belt and sizing-screen by simply exchanging bands (Fig. 3).

Other less commonly used appliances are the grate-screens constructed by Bergman & Grude, and the combination screen shown in Fig. 4, which consists of a Briart grate working inside of a drum-screen, a device often used because it prevents the breaking of the small pieces of coal by the large ones.

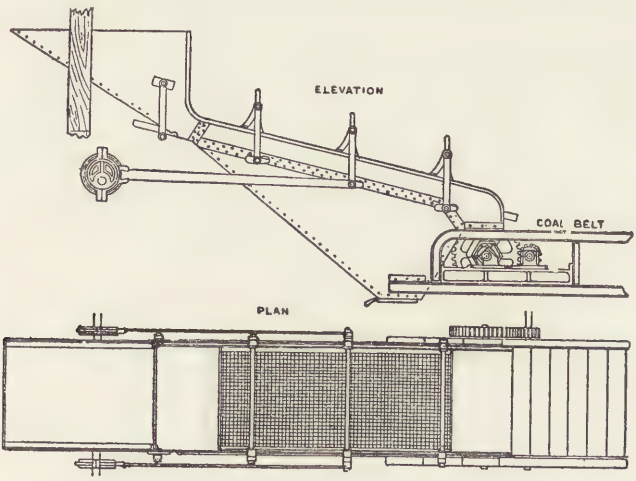


FIG. 3.

Picking-tables are in general use for the separation of the lump coal which is to be conveyed at once to the railway cars. There are many different kinds

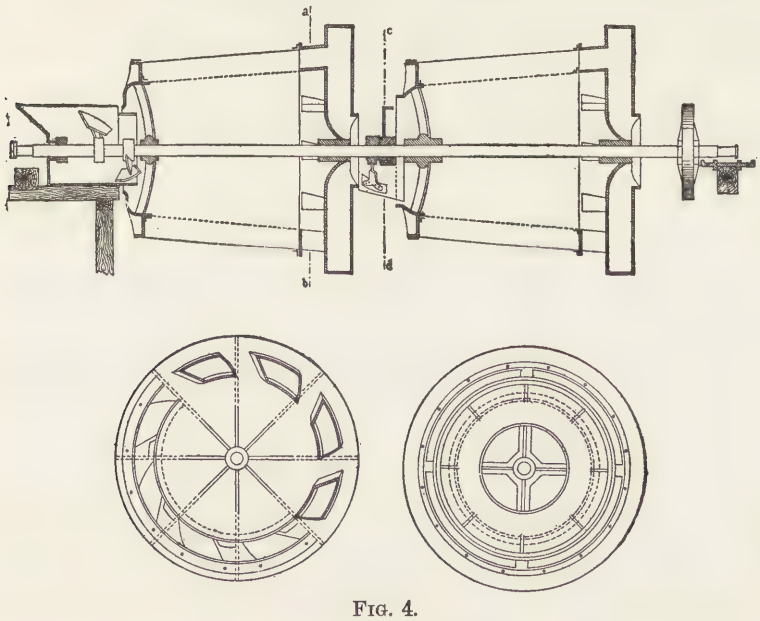


FIG. 4.

on the market, but that introduced by Cornel in 1871 is commonly used. This is a combination of picking-table, bucket-elevator, and bar-screen. The endless belt may be as long as 35 meters and 1 to 1½ meters in width. The Poynton

Collieries, England, use very successfully a combined picking-table and sizing apparatus, which consists of a series of short-linked, endless chains, arranged in parallel and passing between stationary bars set at different distances, the coal falling into cars beneath it according to size, while boys stationed along the belt pick out the refuse. The belt moves 73 ft. per minute and has a capacity of 100 tons per hour, at a cost of 0.5c. per ton. The elevation of the discharge end is adjustable.

The number of sizes into which the small coal is to be separated will depend upon local conditions and the demand of the trade. The German manager aims to utilize the smallest possible sizes, but in England the custom is less economical. The number of sizes called for in England is also much fewer than in Germany, where sometimes 20 or more are necessary in order to secure the best returns for the output, though the tendency there also is to establish a small number of standard sizes. The Westphalian collieries furnish nowadays besides the lump coal only 4 or 5 sizes, exclusive of dust. Many English collieries, especially those whose works have been erected by German firms,

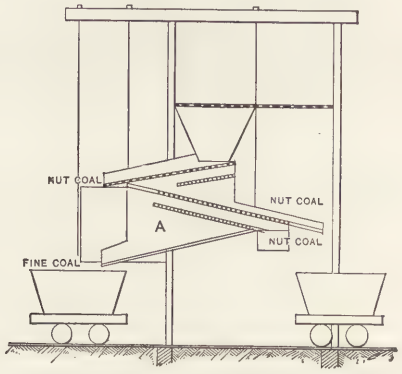


FIG. 5.

make the same division. At the Westphalian the sizes are: Nut No. 1, 90 to 45 mm.; nut No. 2, 45 to 30 mm.; nut No. 3, 30 to 20 mm.; and nut No. 4, 20 to 6 mm.; while all smaller than 6 mm. is considered as dust and is used for coke-making.

The old style of stationary screen (wire cloth or punched steel) is still used in England; in Germany they have been replaced by shaking screens and revolving drums. The material to be sized is generally passed first over the coarsest screen, which saves wear and tear and gives the best sizing. One of the oldest constructions, which is still generally in use, is that introduced by Sauer & Meyer in 1879 (Fig. 5).* This consists of a strong sheet-iron box suspended from a stout iron frame and connected by rods with a crank-shaft. The box contains 3 or 4 screens, 1½ by 2 meters, set at an angle of 10° to 13°.

Many improvements of this style of screen have been made by the German firm of Schluchtemann & Kremer, Dortmund, and by the managers of the Royal Colliery at Gleiwitz. Many modifications have also been made by English engineers.

* *Preuss. Zeits. f. d. Berg- Hütten. und Salinenwesen:* Lamprecht, Die Kohlenaufbereitung, Leipsic.

The shaking screens have been, however, superseded widely by the drum screens, which require only a small amount of power and have a large capacity, while at the same time the sizing is done better, and there is less breakage of coal. A revolving drum screen which is extensively employed consists of a set of punched-steel cylinders placed one within the other, that with the largest holes being innermost.

Messrs. Schmidt & Manderbach have constructed a spiral drum-screen which is rarely used.

A concentric drum screen built by the Humboldt Machine Works of Kalk, which is employed at the Boniface Collieries, Essen, and elsewhere, is shown in Fig. 7. It has a capacity of sizing 8000 to 9000 kgms. of coal per hour at 8 revolutions per minute, while the ordinary drum screen does only 4500 kgms. at 11 revolutions.*

Worth mention also is the shaking screen invented by Karlich,† which has

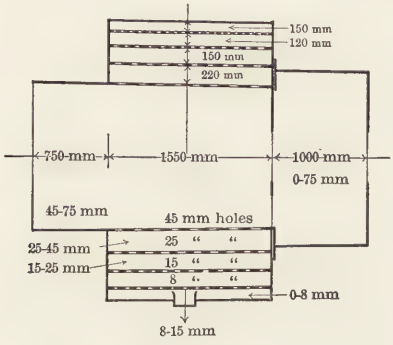


FIG. 7.

been adopted by German engineers in a few cases. This consists of a screen-box suspended from a skeleton, pyramidal frame, which is driven as shown in Fig. 8. The beam *A*, which passes around the screen like a fork and is attached to it by hinge-joints at *C*, rests on a roller at *B* and prevents the screen from revolving, giving it a motion like the shaking of a sieve by hand. The beam alters simultaneously the height of the uppermost sieve, while the charging chute is also connected to it, whereby a regular feed is effected. The advantages claimed for this screen are small consumption of power, regular motion, small floor-space, and low cost.

In washing, so far as the jigs are concerned, there has been no improvement, and principles remain the same as when first established by Lührig. Dust is washed as a rule on feldspar jigs, to which even the refuse of the nut jigs is conducted. Among the engineers who have developed these machines to their present degree of accuracy in work, the Humboldt Machine Works of Kalk, and Messrs. Schluchtemann & Kremer, deserve special mention. They have introduced them in a great majority of the German collieries, and they together with Lührig and certain Belgian engineers may be called the pioneers of coal washing. Baum, assisted by the improvements of Kuepper of Herne, has

* Dick, Die Aufbereitung-Anlage der Zeche Bonifacius, *Zeits. des Vereines Deutsch. Ingenieure.*
† Lamprecht, *Die Kohlen- Aufbereitung*, Leipsic.

successfully undertaken the erection of a number of high-class plants. Baum has also invented a washer, known by his name, and run by compressed air,

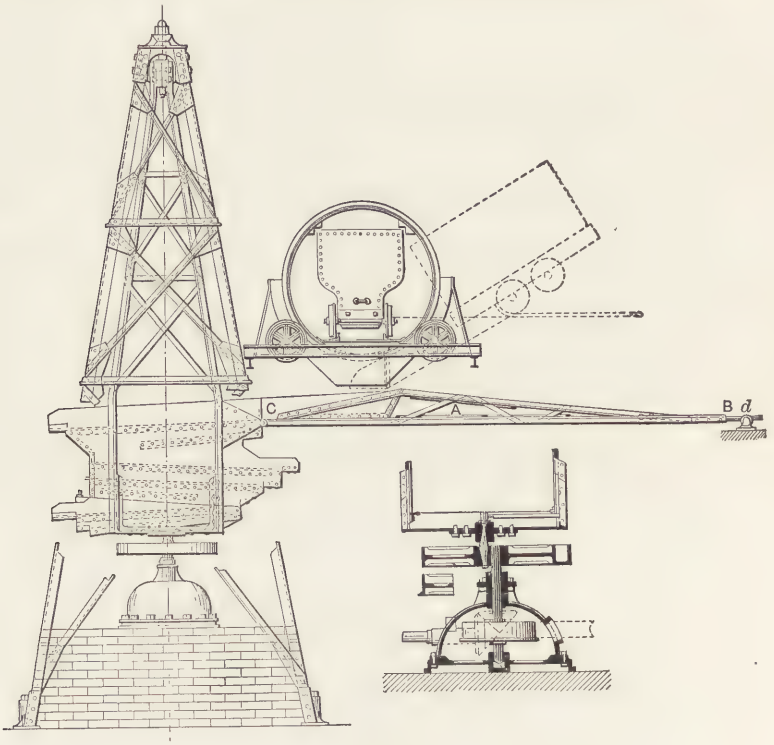


FIG. 8.

which is alternately let in and cut off by a valve in the jigs, forcing the water up through the sieve. The air is blown in at low pressure from a Root's

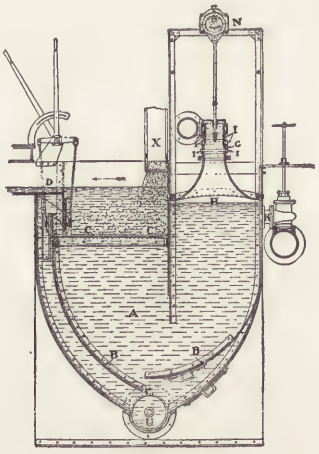


FIG. 9.

blower or other type of compressor. The product of the machine, which is shown in Fig. 9, is not superior to that of the piston jig, notwithstanding the

claim of the inventor. The jig has a double bottom, the refuse accumulating in the lower part, whence it is removed by an endless screw. The air enters through *G* when the valve *H* is at its lowest position, and escapes through the ports *I* when the valve is at its highest, the valve being operated by an eccentric. The water supply is obtained through the pipe *K*. There is no difference between the fine and nut-coal machines, Baum considering the use of feldspar unnecessary.

Another type of machine, constructed by Sheppard & Sons, Bridgend, is shown in Figs. 10 and 11. In the section *B* is the piston and *C* the sieve; the slate passes into the pocket *G*, and the coal over the dam *D*, into the lower pocket *E*, while the water is sucked up through the valve *K* into the upper section again, thus completing the circle. This machine, which is giving very satisfactory results, is in use in more than 100 English collieries. Its advantages are small floor-space required, absence of a pump, and small consumption

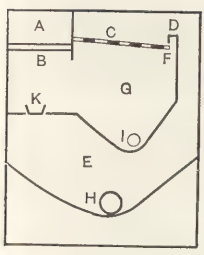


Fig. 10.

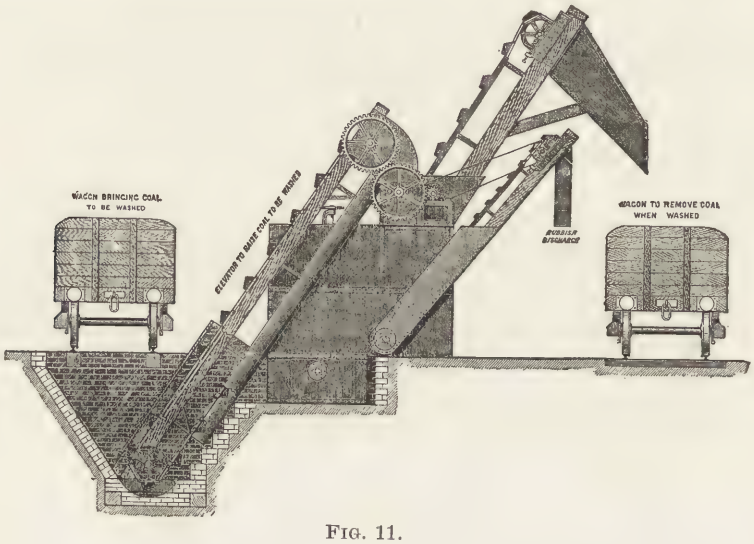


Fig. 11.

of water which has to be renewed only when the machine is cleaned out, a weekly or monthly affair according to the condition of the washed coal. The latter is drained in the elevator, which has perforated buckets, the drippings running back to the jig. The jig has a capacity of 200 to 250 tons per 10 hours.

The Elliot washer, manufactured by the Hardy Patent Pick Company, Sheffield, which has been introduced in a number of English collieries, but not yet in Germany, consists of an inclined steel trough of trapezoidal cross-section, in which works a chain dragging hoes as in an ordinary trough conveyor. The trough is 75 cm. wide at the top, 45 cm. at the bottom, 30 cm. deep, and 18 meters long. The coal to be cleaned is fed at the middle, while the water enters at the head end. The current of water carries off the lighter coal, while the heavier slate works to the bottom and is dragged upward by the hoes to be discharged over the head end. The speed of the hoes and the current of water are variable. The coal and water accumulate in a tank at the

foot of the trough, whence the water is pumped back, a centrifugal pump being used usually for this purpose. The waste of water is only 15%. This apparatus is simple and cheap, requiring little power and attention, but it is doubtful whether its capacity is equal to that of the jig. Much will depend upon the character of the coal and the extent to which it is to be cleaned. Trials of this apparatus in Austria and Germany did not prove satisfactory.

A similar apparatus, patented by Bell & Ramsay, has given good results, however, in Durham and Northumberland. The cost of washing is given by the Washington Iron Works as $\frac{3}{8}$d. per ton, and the total per ton of coke, including repairs, removing duff from settling ponds, and wages of engineman, as $1\frac{1}{2}$d.

Messrs. Wood & Burnett construct an apparatus (Fig. 12) in which the trough itself travels against the current of water. It works well and at a low cost, while the results obtained with it in England in washing nut coal as well as fine coal are favorable.

All our washing apparatus has the serious fault that more or less fine coal, which always exists with the smaller sizes of nut coal, is lost in the process. This has recently been overcome, however, by Mr. Hochstedt, manager of the

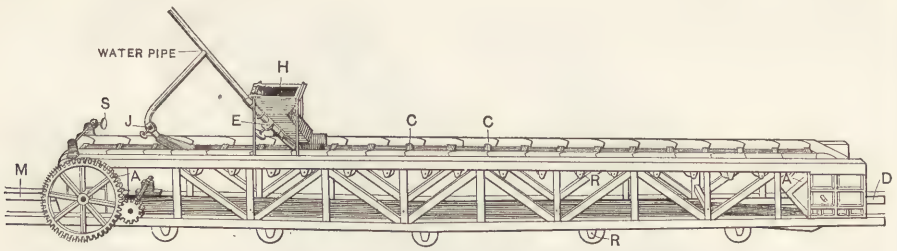


FIG. 12.

Rheinpreussen Colliery, Homberg am Rhein, who separates the fine dust from the small coal by means of an air current. The plan has been successfully carried out at the Anna Colliery, near Aachen, where it is found that the water in the nut-coal washers can be kept tolerably clean by it, which is no inconsiderable advantage.

A similar apparatus is employed at the Wilhelm Colliery at Oberholmdorf near Zwickan with satisfactory results.

The greatest difficulty in coal washing is in cleaning the very fine coal, from which the slate cannot be entirely removed, while the separation becomes less and less satisfactory the finer the coal. The reason for this is that the adhesion of the water increases with decrease in size until the effect of difference in specific gravity is neutralized. This is the case with both the Elliot and Bell & Ramsay machines. Another imperfection of these machines lies in the fouling of the water with coal dust, which makes their work uneven and inaccurate.

If it can be done without increasing the percentage of ash in the product the fine coal screenings are mixed with the washed coal, such a mixture often attaining an increased value, being especially suited for coking on account of its less percentage of water.

Nor can satisfactory results in washing be obtained when the coal carries clay, or similar material which disintegrates in water. Various devices have been constructed to effect this separation, all of them using dry processes, to which class belongs the Hochstedt apparatus previously mentioned. That at the Rheinpreussen and Anna collieries (constructed by the Humboldt Works) is illustrated in Fig. 13. The coal to be treated is discharged by an elevator into the pocket *B*, whence it is fed by the roller *C* on the apron shown; falling in a thin stream in front of the tuyère *I*, the lightest dust is blown into the dust chamber; the heavier material falls before the tuyère *II*, by which any dust remaining is blown into the chamber, and the product falling past the second tuyère goes to the washer. Each tuyère has its own fan, which takes its air from the settling chamber and thereby effects a circulatory current. About 30% of the bulk of the material treated is carried into the dust chamber. By means of this system a coking product with only 7 to 8% water and 5% ash is obtained, with less waste than with a wet process alone, and the possibility of

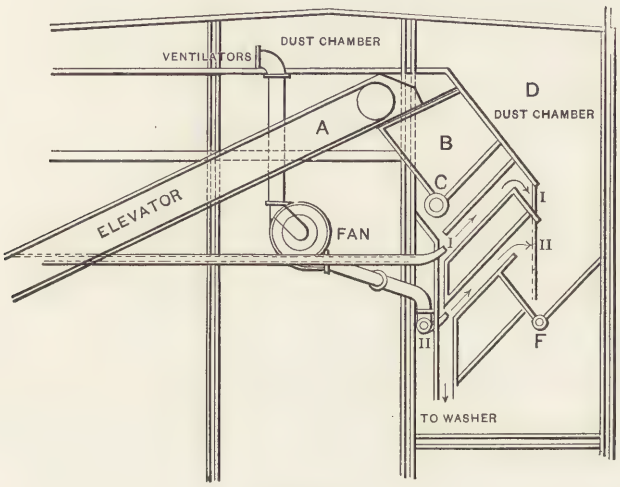


FIG. 13.

carrying out the necessary washing of the dust-free product with clean water. The expense of the plant is small, the two fans costing $600 apiece, and the dust chamber about $750. It treats 500 tons per day at a cost of 5½c. per ton, and effects a net saving of $75 per day. The system operates best where the difference in specific gravity of products is greatest; if the specific gravity of the coal and the refuse is nearly the same it cannot effect a proper separation, although a regular-sized coal for the washers may be obtained and the use of the very fine screens (such as 2 mm.) may be avoided. The separation at the Rheinpreussen Colliery is accomplished with a specific gravity of the coal 1.28, slate 1.86, and schist 1.52.

An experiment was made there in the last part of 1896 with some English dust-coal, which could not be treated properly by the wet process on account of its content of clayey slate, its percentage of ash being about 22. In the tests fine coal with 21.72% ash was brought up to 15.24%; coking coal (2 to

6 mm.) from 21.68 to 4.70%; coking coal, No. 2 and 4 mixed, was cleaned so that it contained only 6.96%. The refuse from the 2 to 6 mm. coking coal that was treated contained only 1.20% coal. The specific gravity of the coal was 1.49; that of the refuse was 3.06.

An apparatus used at the Queen Mary Colliery, Lannisdorf, is shown in Fig. 14. The coal to be cleaned (2 to 6 mm. size) enters at *f*, and falls on the

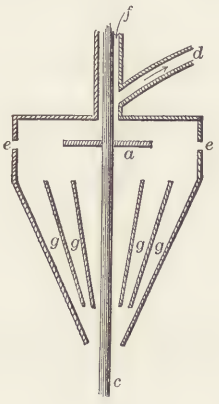


FIG. 14.

disk *a*, the dust being sucked out through the pipe *d;* the disk *a* is rotated by the spindle, from which the coal is thrown off, settling according to size into the divisions *gg*, whence it is drawn off for the washers.

The Sottiaux apparatus is based on the idea of crushing the coal without crushing the refuse and separating them by screening. It is thrown in Fig.

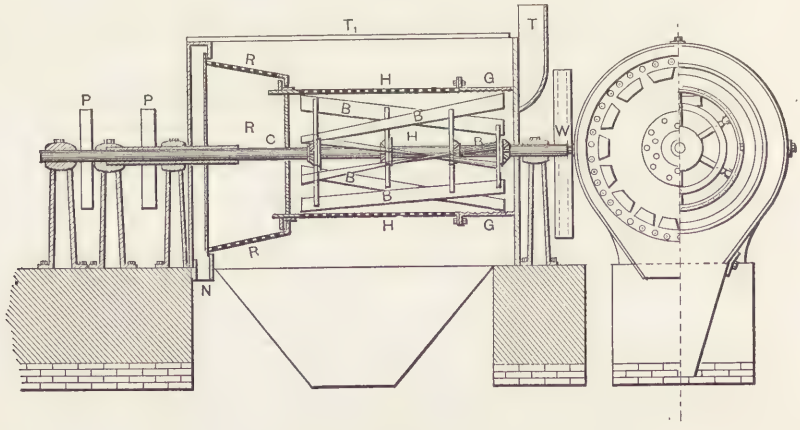


FIG. 15.

15, where *T* is a sheet-iron housing, *G* a corrugated cast-iron drum, and *H* a drum screen, both of which are stationary, *BB* revolving blades which move the coal forward, *R* a revolving screen, *W* a fly-wheel, and *PP* driving-pulleys. Shaft *C* works inside of the drum-screen shaft. The coal is fed at *T* and the refuse drawn off at *N*. The idea has been favored by Belgian engineers and

the machine has been tried for a short time in Westphalia. Apparently the machine is well adapted to its purpose, yet it has not fulfilled the expectations of its inventor. At present it is employed in only one Belgian colliery, the German colliery Frederica, near Bochum, having had to abandon it because its product was too wet, the damp coal obstructing the screens. Obviously it is only applicable when the material is dry and the shale is hard, which is the case at the Belgian colliery (Strépy-Braequignies) where it is used.

Reference may be made to the Pape-Henneberg dry separator, but I am unable to say whether it has been adopted for coal washing.

Of the various methods of treatment of the washed coal, clarification of water, etc., space will only permit mention of a few. In general practice the water is kept in continuous circulation and is run off into settling ponds only when apparatus and cisterns have to be cleaned out. The washed coal is drained

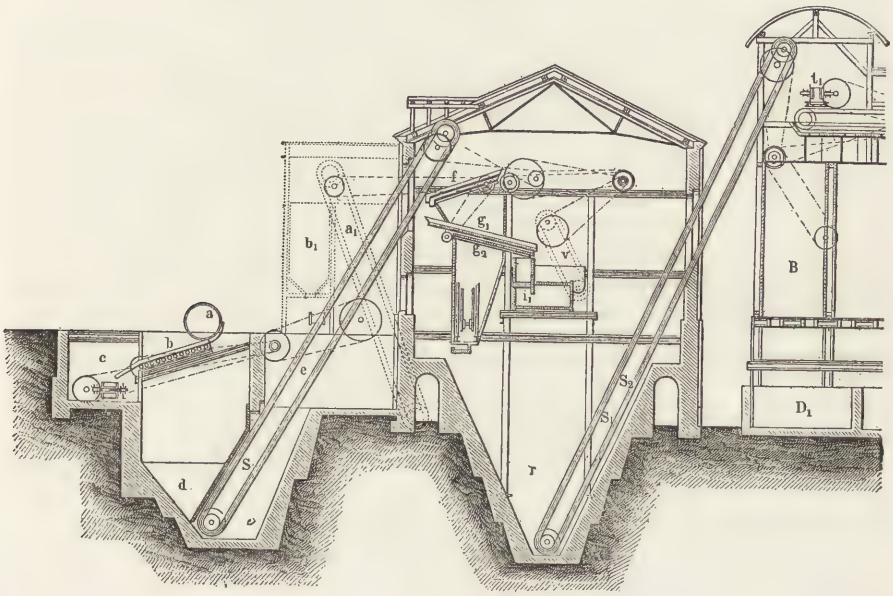


FIG. 16.

by means of perforated chutes, elevator buckets, and even the storage sower may serve for this purpose. In the latter case the method patented by Kuepper is an improvement. He builds the towers with perforated sides, and arranges perforated drainage pipes in the interior. The latest innovation is to use three towers, which are alternately charged, drained, and discharged. Kuepper also uses towers for separating water from the mud, for which sumps and settling ponds are generally used.

Nowadays engineers favor the largest possible storage towers for nut coal as well as for fine coal, on account of the methods of drainage and shipment which are employed. The washed nut coal is drained in passing through one or more chutes to the loading pockets.

Kuepper has also patented a form of tower to avoid breakage of coal in loading filling. In this form the tower, built of wood, stone, or steel, is kept

filled with water with a rising current while the coal from the washer is running to it. In this way the fall of the coal is checked. When the tower is full of coal the inflow of water is checked, and the tower drained by a suitable valve system.

German and English engineers both endeavor to avoid wood as much as possible in the construction of coal-washer buildings, while the corrugated iron roofs which were in general use are being replaced by cement roofs, as being more durable. All new constructions are well lighted by a sufficiency of windows in the daytime, and by electricity at night. Shafts, pulleys, and belts are so arranged that they can be attended to without danger. Special care is taken in design to have all departments capable of proper supervision,

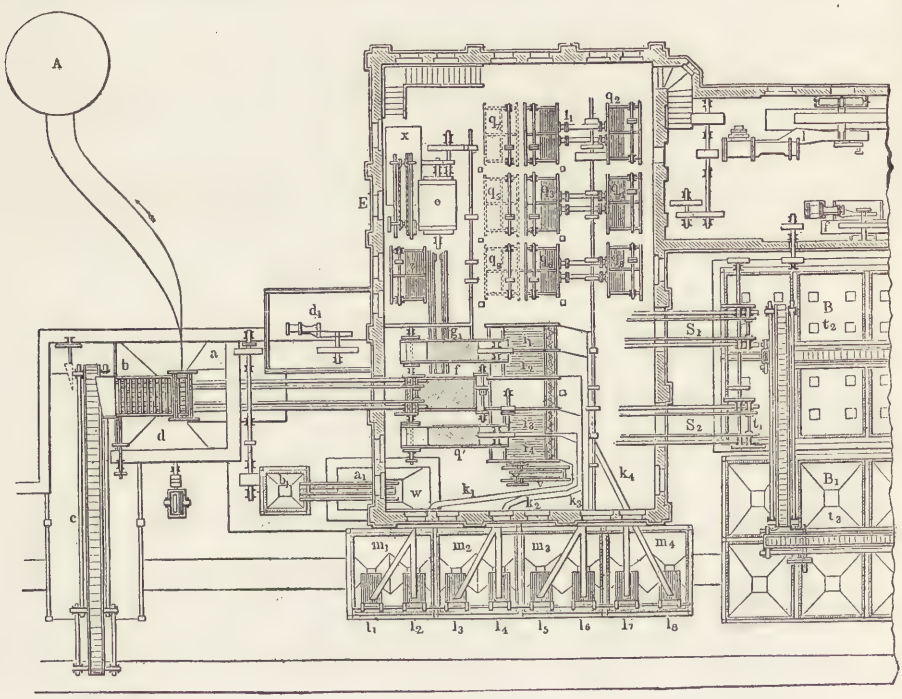


FIG. 17.

and for the introduction of all known labor-saving devices. In this way the cost of coal washing has been brought down to a very low figure.

The washer of the Zollen Colliery, erected by Schluchtemann & Kremer in 1891, on the lines generally followed by that firm, is illustrated in Figs. 16 and 17. The output of the pit *A* is brought to the dumper *a*, and dumped over the grate *b* (Bergmann & Emde pattern), which separates all pieces over 80 mm. size. The latter go to the Cornel loading table *c*, where the slate is picked out. The coal under 80 mm. size falls into the pit *d*, whence it is raised by the elevator and dumped over the grate *f*, which makes three sizes, namely, 0 to 10 mm., 10 to 28 mm., and 28 to 80 mm. The two latter are divided by grates g^1 and g^2 into four nut sizes, going thence with the least possible fall to the nut jigs *i*. From the jigs the coal goes to the shaking

screens l^1–l^8, which drain off the water and remove any remaining particles of dirt, the clean product going thence to the pockets m^1–m^4, ready for shipment. The fine coal under 10 mm. size is raised by the bucket elevator n to the drum screen which divides it into three sizes, namely, 0 to 4 mm., 4 to 7 mm., and 7 to 10 mm. These go to the fine-coal washers p^1–p^6 (three machines of this kind being kept in reserve). The fine coal, together with any mud and the drippings of the nut drainage screens, is discharged into the settling tank r, and from there to the storage and drainage towers BB, by way of the elevators S^1–S^2. The clear water is run back from the settling tank to the pump c, while the dirty water is conducted to tanks at a lower level, where it is left until clean enough for use. The refuse of the nut washers is run into the waste-bin w, whence it is hoisted by an elevator a to the hopper C, from which it is loaded on cars for removal. The capacity of this washer is 80 tons an hour, with a crew of 11 men. The cost of the plant, including the iron buildings, a covered hall for the shipment of the lump coal, and the towers for coke-coal, was $60,000.

The washer of the Minister Stein Colliery, near Dortmund, was designed by Kuepper and erected by the Baum Machine Company. It is constructed throughout of iron and has a capacity of 150 tons per hour. The coal cars are brought into the building and delivered to the dumping machine, which is situated at the same level as the mouth of the shaft, and discharged on a shaking screen. The lump coal (over 80 mm.) passes from this screen to adjustable loading belts, which have a width of 1.6 meters, and transfer the coal to the railroad cars which stand on the second, third and fourth line of rails, while a separate belt loads the cars of the first line. The refuse of the picking-tables is loaded on waste cars, and removed by an elevator. The fine coal (under 80 mm. size) drops from the shaking screen into a pit, whence it is lifted by an elevator to a drum screen, which produces four nut sizes, and the very fine coal. A water jet carries the nut coal through troughs to four fine coal jiggers. From there the coal and water are run to the loading pockets, the sizes being kept separate. The water which carries the coal drains off through screens into troughs which carry it back to the centrifugal pump, together with any mud from the same source. The overflow of the nut pockets runs also to the same troughs. The washed fine coal is carried, together with the water used for washing, to a centrifugal pump, which lifts them into the storage towers, of which there are three, of a capacity of 1200 tons each. The fine coal settles slowly in the towers, on account of the small velocity of the mixture, while the clear water overflows into troughs, which conduct it back to the washers. These towers are charged, drained, and discharged in rotation. The slate is carried off by a screw-conveyor beneath the washers and an elevator with draining buckets to the waste-bin.

The Collieries of France and Belgium.

By Paul Schneider and Gaston Le Bel.

France can furnish from her own mines all the combustibles that she consumes and is capable of consuming. Her territory includes the most diverse kinds of coal—anthracites, bituminous, lignites, and peat. The coal regions are situated as follows:



The Coal Basins of France.

North: Basin of the North and of the Pas-de-Calais, which is a prolongation of the Belgian basin.

East: Basin of Ronchamp (Haute Saône).

West: Basin of the Lower Loire; basin of the Vendeé.

Center: Basin of Bourgogne (Saône and Loire), Blanzy, Le Creusot, Epinac; Basin of the Nivernais (Allier), Commentry, Bezenet; numerous lacustrian

basins of which the principal are those of Deeize, Ahun, St. Eloi; and finally the basin of the Loire (St. Etienne, Rive de Gier).

South: Basin of L'Aveyron (Decuzeville, Aubin), basin of the Tarn (Carmaux, Albi), basin of the Gard (La Grande-Combe, Bessègés), basin of L'Herault (Graissesal).

The richest basin of France is that of the North and Pas de Calais, a long straight zone, which traverses Belgium and penetrates the department of the North. Its length is more than 400 kilometers, its width 6 to 10 kilometers, and its area 250,000 hectares. In this stretch there are considerable variations in the thickness of the coal beds and in their richness.

The character of the coals of the French basins is very varied.

The anthracites (4 to 5% volatile matter) are found at Marais (Allier), La Mure (Isére), and Messeix (Puy de Dome).

The anthracitic coals (6 to 7% volatile matter) occur at Vicoigne, Fresnes, and Vieux Condé (Nord).

The short-flame, lean coals (10% volatile matter) are also found in the department of the North.

The semi-fat coals (11 to 12% volatile matter).

The fat coals, called blacksmiths' coal, and the coking coal (13 to 14% volatile matter) of which the best types exist at St. Etienne.

The gas coals (15 to 16% volatile matter) at Montrambert (Loire) in the basins of the Nord and of the Tarn.

The long-flame lean coals (21 to 22% volatile matter) found especially at Blanzy (Saône and Loire), Commentry, and Decazeville.

The true lignite at Fuveau in Provence.

The ligneous lignite at La Tour du Pin (Isére), Chambery, in La Manche, and La Dordogne.

Peat at a large number of places, but especially in La Somme.

Historical.—The first work in France having to do with the extraction of coal appears to have been at the commencement of the fourteenth century in the districts of Roche la Molière (basin of the Loire); it has been found that certain *seigneurs* of that region had arrogated to themselves a claim on all the minerals in their lands. This claim was for the half of the coal extracted at a charge to the seigneurs of half the cost of extraction; or if they so preferred it they could reserve a quarter of the coal extracted without paying anything for extraction. The most ancient of these titles that exists bears the date February 18, 1321.

The reign of Louis XIV. gave a great impulse to the manufacture of arms, inaugurated important works for the construction of canals, and consequently increased the prosperity of the basin of St. Etienne; but the coal industry did not really develop itself until the birth of industrial liberty.

In the basin of La Bourgogne the first exploitations appeared in 1528, 1610, and 1640, but they did not acquire any importance until 1782. It is at this epoch that the extraction of coal for the glass-works and later for the crystal works of Marie Antoinette at the mines of Creusot was begun. These mines ultimately supplied the Royal foundry, and after that the works established by MM. Schneider fréres at the beginning of the century.

The history of the first explorations in the basin of the North is one of the most interesting in our industry. The coal-bearing strata are covered by beds of chalk, or *morts terrains,* in which are subterranean sheets of water, or *niveaux.* These *morts terrains* exist also in Belgium, where several collieries

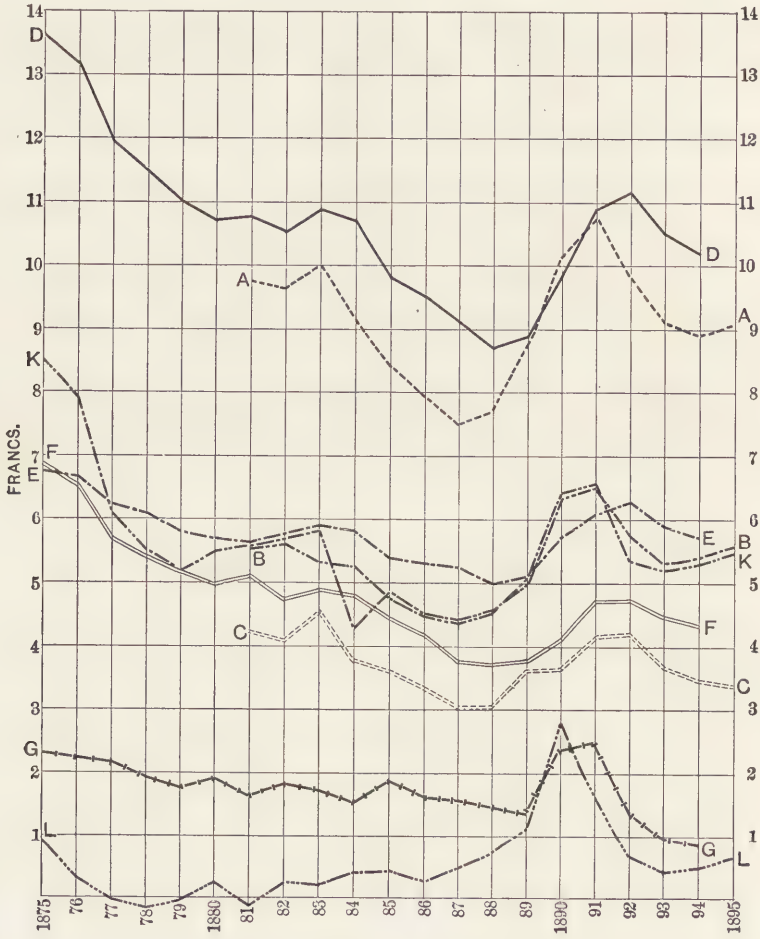


DIAGRAM I.

AA—Total cost of production, Belgium.
BB—Analysis of cost of production, Belgium; cost of labor.
CC—Analysis of cost of production, Belgium; expenses other than labor.
DD—Total cost of production, France.
EE—Analysis of cost of production, France; cost of labor.
FF—Analysis of cost of production, France; expenses other than labor.
GG—Collieries of France, part of the capitalist per ton of coal; the part of the workman coincides with line *EE*.
KK—Collieries of Belgium; part of the workman per ton of coal.
LL—Collieries of Belgium; part of the capitalist per ton of coal.

of the Mons bed had to pass through them, and from the study of the formations there resulted the probability of the continuation of the coal-bearing zone in the territory around Valenciennes.

It was this idea which guided the first explorations undertaken by Comte

Desandrouin. This man, to whom the department of the North owes a great part of its prosperity, had determined the east and west course of the coal-bearing area and its progressive dip under the *morts terrains*. Assured by his first reconnaissance and the identity of the superficial conditions, he began the exploration in 1716, and in 1717 discovered coal at Fresnes-Mais, but the

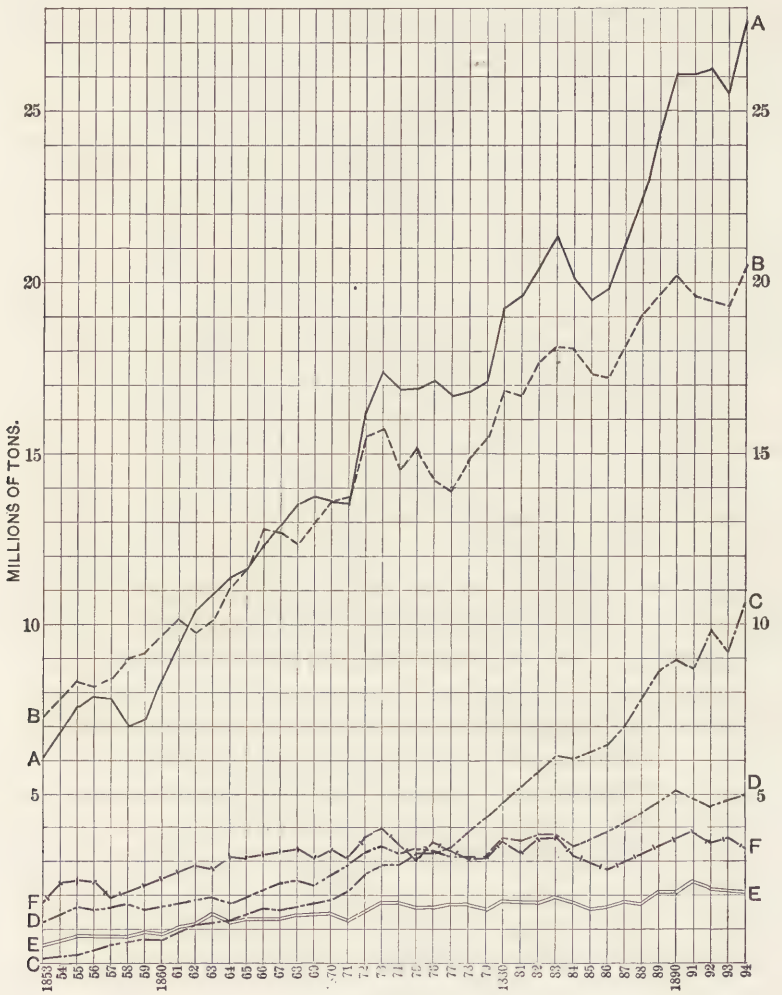


DIAGRAM II.

AA—Total production of coal in France.
BB—Total production of coal in Belgium.
CC—Production of the Pas-de-Calais basin, France.
DD—Production of the basin Du Nord, France.
EE—Production of the basin of the Gard, France.
FF—Production of the basin of the Loire, France.

presence of the subterranean bodies of water, very difficult to hold back, opposed great difficulties to the working of the mines. The first pit at Fresnes was flooded and had to be abandoned, and it was not until 1723 that the exploration of these beds was accomplished. This first conquest was a great thing; however, it was not yet a success, for the coal of this territory proved to

be lean and only suitable for the lime-burners. But the relative position of the fat and lean coals had already been determined in Belgium; it was known that the former were superior to the others, and that according to their stratification it was necessary to seek them more to the south. New attempts were made, and in 1734, that is after 17 years of effort, and the sinking of 17 soundings in the districts of Etræux, Quarouble, Bruai, etc., the beds of Anzin were finally discovered.

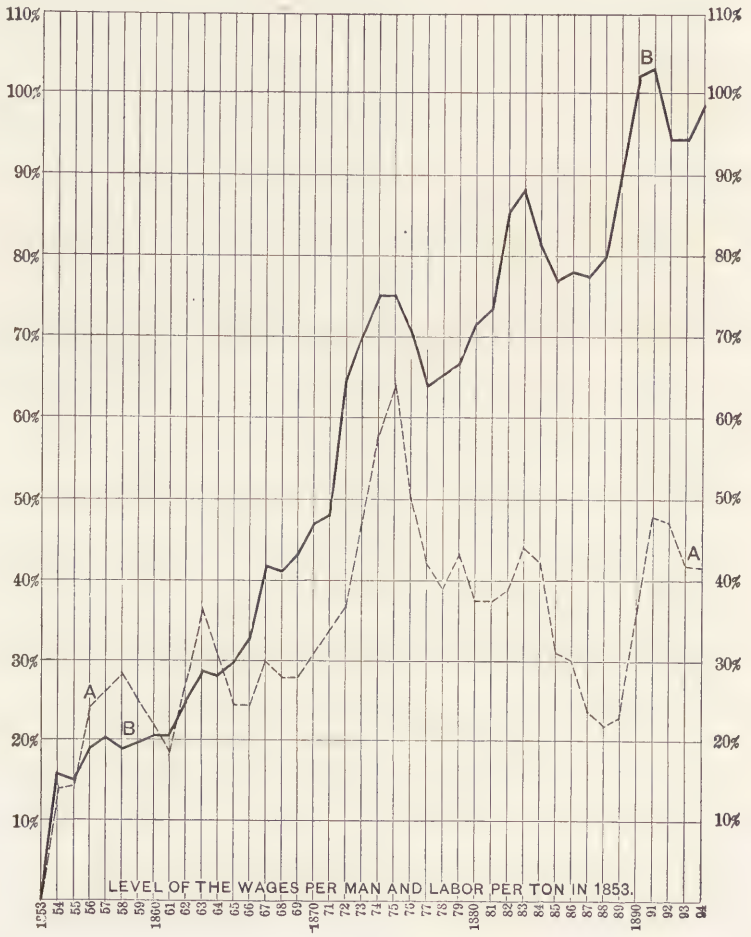


DIAGRAM III.

Comparison of wages and cost of labor per ton of coal extracted in France.
AA—Labor per ton extracted. *BB*—Average wages per man.

The extreme parts of the basin of the North were not recognized, however, until 1845. Numerous soundings made in the Pas-de-Calais cut again the coal-bearing strata, first at Oignies in 1841, and some explorations were undertaken, but it was not until 1855 that the basin of the Pas-de-Calais was practically delimited.

WORKING CONDITIONS OF THE VARIOUS BASINS.

North.—The number of beds in this basin is estimated at 150, their thickness varying from 0.10 to 1.60 meters; many a seam exploitable at a certain point, that is having more than 0.30 meters, ceases to be so a little further away. The number of exploitable seams is reckoned as 80. The mean depth attained by the works is 350 meters, certain pits having gone to 650 meters.

Pas-de-Calais.—Conditions in general are analogous to those of the North.

Center.—The most important is the basin of the Loire, which holds the third place in point of production. It has the form of a triangle of which the base

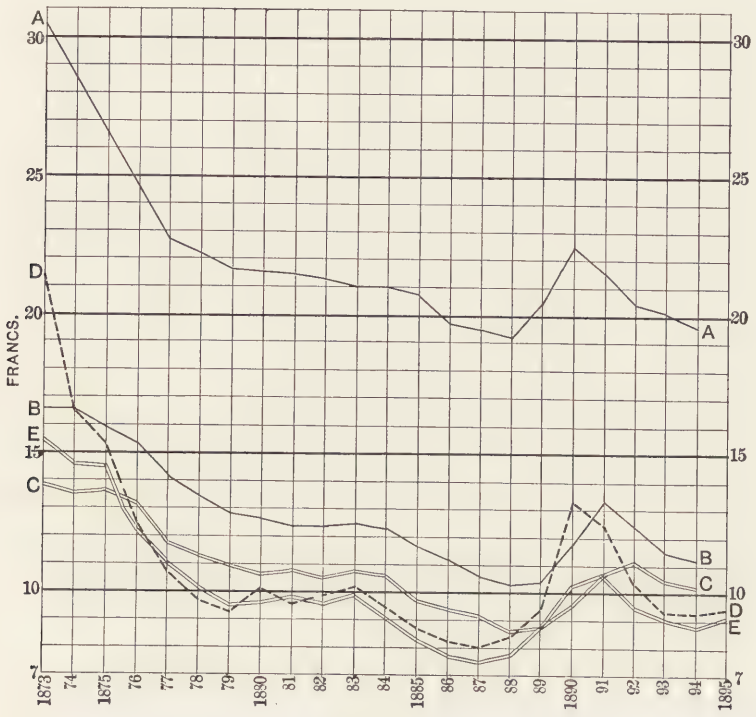


DIAGRAM IV.

AA—Selling price at point of consumption, France.
BB—Selling price at mine, France.
CC—Cost of production, France.
DD—Selling price at mine, Belgium.
EE—Cost of production, Belgium.

rests on the Loire for a length of 12 kilometers. The surface of the basin is very rugged, the characteristic point of the contour of the coal-bearing district being the divide between the basins of the Loire and of the Rhône.

In the region of Rive de Gier, which includes the straightest part of the basin, the order of the beds is simple and perfectly defined. In the opposite region, that of St. Etienne, where the basin is 10 kilometers wide, the beds are numerous, continually broken and uplifted to the surface by the movements of the earth. In the same region the general structure of the formation is that called *Fond de Bateau*, the value resting principally in a powerful seam of 8 to 18 meters thickness called the *Grande Masse*. This seam overlies generally two

others called *Batardes*. The aggregate of these beds gives 10 to 25 meters of coal.

In the region of St. Etienne, on the contrary, there are three distinct levels, constituting a formation superior to that of Rive-de-Gier. This region comprises 20 to 24 beds of coal, giving a total thickness of 47 to 58 meters.

The basin of Bourgogne is identified by two series of parallel outcrops. It is almost entirely covered by Triassic sandstone. The surface of the region is not very rugged, but it is, however, elevated so as to form the divide between the Loire and the Saône.

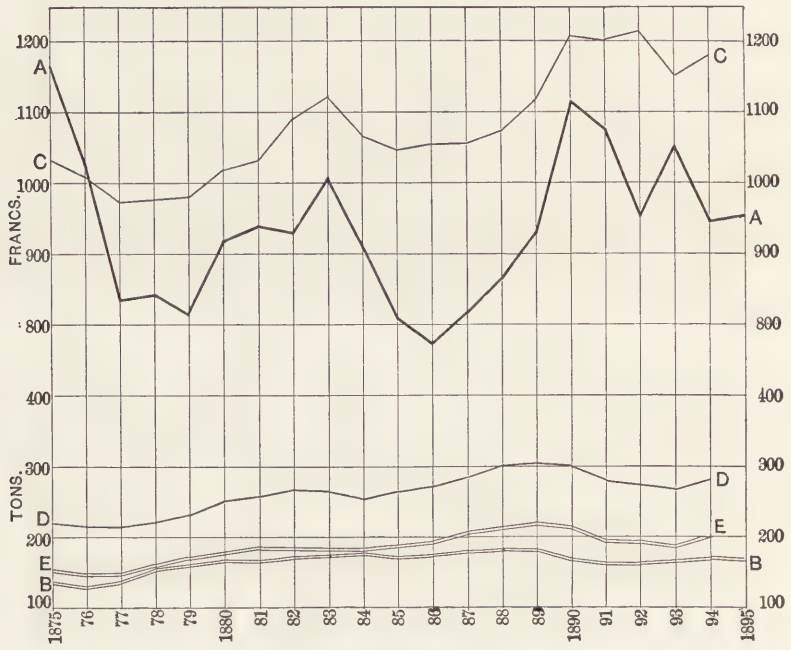


DIAGRAM V.

AA—Average annual earnings per man, Belgium.
BB—Annual production per man, Belgium.
CC—Average annual earnings per man, France.
DD—Annual production per man, calculated on basis of men employed underground.
EE Annual production per man, calculated on total number of employees.

The coal deposit at Blanzy may be considered as one of the most remarkable. The shafts first cut three beds of 1 to 3 meters thickness; below these the principal deposit consists of two beds of 10 to 20 meters thickness. These two beds are broken by step faults, which throws them down to greater depth than they would otherwise attain. The exploitation of these great masses of coal presents difficulties. It is necessary to fill up the chambers and waste the wood required to support the enormous pressure of the overlying ground. The inflammable nature of the coal adds another difficulty, fires originating very quickly; they are fought by isolating them.

South.—The basin of the Gard in this region holds the fourth rank in production. The coal-bearing district is singularly abrupt, its altitude

exceeding 500 meters. From the heights of the Pinèdes one sees that the coal-bearing strata are dominated by mica schists; and at some distance are the granitic summits of the Mont Lozère, of which micaceous and talcose schists form the buttresses. The rain waters furrow violently these coal-bearing slopes, and the torrents of water cause a constant difficulty to the human industry on the flats and at the bottoms of the valleys.

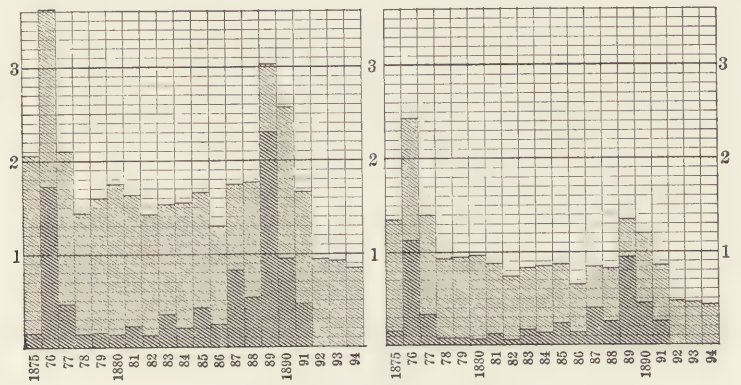


DIAGRAM VI.—FRANCE.

Accidents per 1000 workmen. Accidents per 100,000 tons.

Total number and number due to fire-damp.

It is in this savage place that the forges of Tamaris were founded in 1832, and some years later coal was sought in the ravine of the Grande Combe, which showed numerous outcroppings. The basin is divided into two parts by a promontory of talcose schists. To the south of this the two parts unite, but it is covered by the later deposits of the Trias and the Lias, and by the Neo-

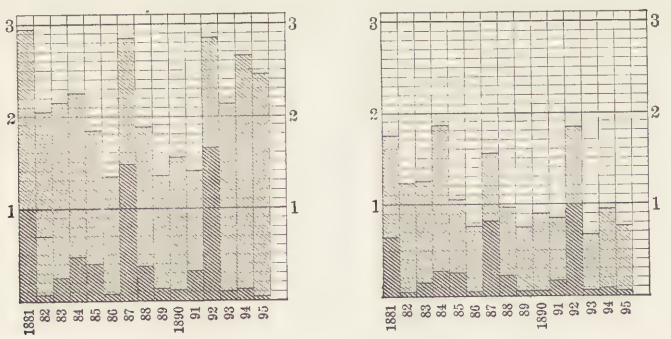


DIAGRAM VII.—BELGIUM.

Accidents per 1000 workmen. Accidents per 100,000 tons.

Total number and number due to fire-damp.

comian and Tertiary limestones. This basin shows 18 seams of coal, giving a total thickness of 20 meters. Several of these seams are less than 1 meter in thickness, but the principal ones are 3 to 4 meters.

Basin of the Tarn and the Aveyron.—In the Aveyron the coal-bearing district can be divided into four groups. The uppermost comprises the great

seam of Decazeville. It is covered by schistose sandstone and bituminous schist. The second is separated from the great seam of Decazeville by 50 meters of sterile sandstone. It comprises three beds of 1 to 5 meters thickness. The third stage is characterized by a great seam of 6 to 15 meters thickness. Finally the lowest group comprises a series of seams of from 1 to 3 meters in thickness.

In the Tarn the basins of Carmaux and Albi are remarkable for their regularity. They contain at Carmaux four seams of 1 to 3 meters thickness, and at Albi four seams of which the most important is 14 meters thickness.

Finally, the basin of Graissessac is one of those long, straight valleys filled with coal deposits, then uplifted above all the Secondary and Tertiary deposits in such manner that the coal-bearing strata have remained uncovered. Its length is 18 kilometers and its mean width 1 kilometer.

Basin of the Alps.—The coal-bearing strata, more or less uplifted, on the slopes of the Alps and placed even in the most rugged regions contain seams which are wholly anthracitic. The anthracites of La Mure (Isére) are types of this class.

Basin of the West.—The coal-bearing strata of the Lower Loire form a zone of 500 to 1200 meters in width and not less than 100 kilometers in length. The formation of the beds is regular and *en chapelets*. The quality of the coal is variable, ranging from anthracite to the semi-fat.

The basin of La Vendeé presents itself in the form of a zone of about 1 kilometer mean width and 20 kilometers length, consisting of beds strongly inclined and resting upon the primitive rocks.

PRODUCTION.

The production of the French collieries is shown in the accompanying tables and diagrams, from which it will be remarked that since 1853 it has not ceased increasing, although there have been temporary declines, brought about generally by political events. Among the various geographical groups certain have remained stationary or nearly so in point of production, while others have developed considerably.

In 1894 the number of persons occupied in coal-mining was as follows:

	Men.	Boys. (*a*)	Women.	Children. (*b*)	Totals.
Interior.....	86,400	5,500	.....	4,500	96,400
Exterior.....	27,600	2,500	3,700	4,400	38,200
Total.....	114,000	8,000	3,700	8,900	134,600

(*a*) From 16 to 18 years. (*b*) From 13 to 16 years.

The small number of women is due to the interdiction of their employment in night-work and the limitations in day-work prescribed by the law of November 2, 1892, whereby the labor of women and children in industrial establishments is restricted to 8 hours daily between 9 A.M. and 6 P.M.

The number of persons occupied above ground is not more than 29% of the total, while in 1853 it was 35%. The decrease is due to the improvements in mechanical preparation of the coal, picking, sizing, etc.

STATISTICS OF COLLIERIES IN FRANCE AND BELGIUM; 1853–72—TABLE I.

Year.	Production, in Tons.		Number of Men Employed.		Year.	Production, in Tons.		Number of Men Employed.	
	France.	Belgium.	France.	Belgium.		France.	Belgium.	France.	Belgium.
1853.....	6,163,919	7,172,687	40,958	54,204	1863.....	10,972,205	10,345,330	73,357	79,187
1854.....	7,056,071	7,947,742	46,766	62,194	1864.....	11,506,908	11,158,336	77,342	79,779
1855.....	7,715,407	8,409,380	54,322	70,980	1865.....	11,847,190	11,840,703	7[illegible],950	82,368
1856.....	8,172,398	8,212,419	58,821	73,585	1866.....	12,516,220	12,774,662	79,909	86,721
1857.....	8,148,572	8,383,902	59,467	72,577	1867.....	13,012,715	12,755,822	83,492	93,339
1858.....	7,563,198	8,925,714	56,035	73,850	1868.....	13,532,389	12,298,589	84,909	89,382
1859.....	7,694,717	9,160,702	58,310	77,293	1869.....	13,787,927	12,942,894	84,499	89,928
1860.....	8,551,584	9,610,895	59,240	78,232	1870.....	13,659,689	13,697,118	82,673	91,993
1861.....	9,664,639	10,057,163	65,619	81,675	1871.....	13,606,884	13,733,176	83,649	94,286
1862.....	10,526,547	9,935,645	69,382	80,302	1872.....	16,245,834	15,658,948	91,899	98,863

STATISTICS OF COLLIERIES IN FRANCE AND BELGIUM; 1873–95—TABLE II.

Year.	Production, in Tons.		Number of Men Employed.		Mean Yearly Earnings		Annual Production per Man.		Selling Price at Mine and Place of Consumption.			Cost of Production.						Profit per Ton.	
									Mine.		Place	Labor.		Other Ex.		Total.			
	France.	Belgium.	France.	Belgium.	France.	Belgium.	France.	Belgium.	France.	Belgium.	France.	France.	Belgium.	France.	Belgium.	France.	Belgium.	France.	Belgium.
1873	17,479,341	15,778,401	105,513	107,902	.....	.....	....	....	16.61	21.40	31.83	6.90	....	7.00	....	13.90	15.47	2.71	5.93
1874	16,907,913	14,669,029	106,289	109,631	.....	.....	....	....	16.59	16.45	28.33	6.83	....	6.87	....	13.70	14.85	2.89	1.56
1875	16,956,840	15,011,331	108,712	110,720	1,030	1,163	156	136	15.93	15 31	26.56	6.80	....	6.88	....	13.68	14.65	2.25	0.86
1876	17,101,448	14,329,578	110,802	108,543	1,010	1,031	154	130	15.33	13 55	24.82	6.60	....	6.53	....	13.13	13.29	2.20	0.26
1877	16,804,529	13,938,523	108,907	101,343	1,120	835	154	136	14.06	10.97	22.87	6.20	....	5.61	....	11.91	11 05	2.15	*a*0.08
1878	16,960,916	14,899,175	106.415	99,032	1,206	842	159	150	13.46	9.92	22.19	6.10	....	5.41	....	11.51	10.02	1.95	*a*0.10
1879	17,110,979	15,447,292	102,472	97,711	1,203	809	167	154	12.93	9.39	21.84	5.85	....	5.15	....	11.00	9.40	1.93	*a*0.01
1880	19,361,564	16,866,698	107,237	102,930	1,221	920	180	164	12.71	10.06	21.74	5.75	....	5.00	....	10.75	9.83	1.96	0.23
1881	19,765,983	16,873,951	106,410	101,351	1,146	931	186	164	12.43	9.70	21.61	5.70	5.59	5.10	4.20	10.80	9.79	1.63	*a*0.09
1882	20,603,704	17,590,989	108,269	103,701	1,181	926	190	170	12.36	10.00	21.42	5.85	5.66	4.70	4.09	10.55	9.75	1.81	0.27
1883	21,333,884	18,177,754	113,003	106,252	975	1,006	189	170	12.50	10.17	21.00	5.95	5.88	4.95	4.04	10.90	9.92	1.60	0.25
1884	20,023,514	18,051,499	109,426	105,582	980	914	183	172	12.33	9.53	21.00	5.90	5.38	4.85	3.82	10.75	9.20	1.58	0.35
1885	19,510,530	17,437,603	101,616	103,095	990	812	192	168	11.73	8.87	20.89	5.40	4.78	4.45	3.67	9.85	8.45	1.88	0.46
1886	19,909,894	17,285,543	102,354	100,282	1,020	783	195	172	11.19	8.25	19.79	5.35	4.50	4.20	3.45	9.55	7.95	1.64	0.30
1887	21,287,581	18,378,624	103,163	100,739	1,030	815	206	183	10.63	8.04	19.65	5.30	4.47	3.85	3.07	9.15	7.54	1.48	0.50
1888	22,602,894	19,218,481	104,959	103,477	1,100	869	215	186	10.31	8.43	19.12	5.00	4.67	3.80	3.08	8.80	7.75	1.51	0.68
1889	24,303,509	19,869,480	110,919	108,382	1,120	932	219	183	10.42	9.45	20.45	5.08	5.10	3.87	3.75	8.95	8.85	1.47	0.60
1890	26,083,118	20,365,960	121,555	116,779	1,070	1,117	215	173	11.94	13.18	22.54	5.67	6.40	4.14	3.75	9.81	10.15	2.13	3.03
1891	26,024,893	19,675,644	131,832	118,983	1,050	1,068	197	165	13.25	12.58	21.61	6.10	6.55	4.85	4.20	10.95	10.75	2.30	1.83
1892	26,178,701	19,583,173	133,193	118,578	1,060	940	197	165	12.40	10.28	20.38	6.25	5.38	4.85	4.80	11.10	9.68	1.30	0.60
1893	25,650,981	19,410,519	132,644	116,861	1,060	1,050	193	166	11.49	9.34	20.03	5.95	5.30	4.55	3.72	10.50	9.02	0.99	0.32
1894	27,416,905	20,534,501	134,557	117,103	1,084	926	204	175	11.22	9.32	19.53	5.80	5.35	4.45	3.57	10.25	8.92	0.99	0.40
1895	.....	20,457,604	.....	118,957	.....	948	....	172	.....	9.45	.....	....	5.50	....	3.55	.....	9.05	0.97	0.40

(*a*) Loss.

RELATIVE PART OF WORKMAN AND CAPITALIST IN A TON OF COAL. BELGIUM—TABLE III.

Year.	Workman.	Capital.	Year.	Workman.	Capital.	Year.	Workman.	Capital.	Year.	Workman.	Capital.
	Francs.	Francs.		Francs.	Francs.		Fra cs.	Francs		Francs.	Francs.
1861.....	5.90	0.97	1870.....	5.82	0.91	1879.....	5.15	*a*0.01	1888.....	4.67	0.68
1862.....	5.60	0.78	1871.....	5.95	1.04	1880.....	5.54	0.23	1889.....	5.10	0.60
1863.....	5.35	0.72	1872.....	6.58	2.27	1881.....	5.59	*a*0.09	1890.....	6.40	3.03
1864.....	5.10	0.96	1873.....	9.30	5.93	1882.....	5.66	0.27	1891.....	6.55	1.83
1865.....	5.45	1.17	1874.....	8.60	1.56	1883.....	5.88	0.25	1892.....	5.38	0.60
1866.....	5.80	1.81	1875.....	8.50	0.86	1884.....	5.38	0.35	1893.....	5.30	0.32
1867.....	6.48	1.60	1876.....	7.90	0.26	1885.....	4.78	0.40	1894.....	5.35	0.40
1868.....	5.85	0.81	1877.....	6.10	*a*0.08	1886.....	4.50	0.30	1895.....	5.50	0.40
1869.....	5.78	0.62	1878.....	5.45	*a*0.10	1887.....	4.47	0.50			

(*a*) Loss.

A similar improvement since 1853 is shown in the wages and proportion of labor in the cost of getting the coal. In 1894, the wages having increased 100%, the cost of labor in a ton of coal had only increased 42%, owing to the

progress in the art of mining. The average annual wages of the workman in 1894 were 1084f.; in 1875, 1030f.; consequently the increase in that period has been about 6%.

Cost of Production, Selling Price, Profits, etc.—The ceaseless efforts of the colliery managers in face of the constant necessity of increasing the wages of the workmen and of struggling against the competition which caused the selling price to fall have had for effect the progressive lowering of the mean cost of production, which in 1894 was as follows: Labor, 5.80f., other expenses 4.45f., total 10.25f. It is, thanks to mechanical improvements, and to good ventilation, that the proportion of labor in the cost of production has been reduced while wages have been increased.

The cost of production in France is high, which is due to the particularly difficult conditions of exploitation. We have, it is true, some great seams in which the workman can produce as much as the English, but this great thickness requires an elaborate system of supporting the roof, and the cost of timbering and filling absorbs the economies realized in breaking the coal. In France the collieries consume 0.75 to 1.50f. worth of timber per ton of coal extracted, while the English collieries use only 0.10f. to 0.20f. worth, because of the general solidity of their formations. In the French mines not only is one obliged to abandon in the chambers the most of the props, but the number of working galleries to be maintained being for a given output greater than is necessary in the English collieries, the consumption of wood is a charge which bears heavily on the production.

The general expenses in France are commonly very high, in spite of the fact that since some years the mining has been concentrated in the smallest possible number of pits, the complication of the works and underground service requiring a considerable staff of engineers and master-miners. These general expenses are augmented, moreover, by the very complex conditions of the works above ground.

From 1873 to 1894 the mean selling price on the platform of the mine has decreased 33%. and the cost of production in the same time has been reduced only 26%. The profit of exploitation having thus diminished constantly, we have shown in Diagram I. the proportional parts of the workman and his employer in the production of a ton of coal.

While the annual wages of the workman have increased 6% in the last 20 years, as we have previously pointed out, it is not the same with the profits, which have fallen 58%, and if to-day the cost of labor in the cost of production has fallen, thanks to the better conditions of exploitation, the part of the laborer in the value of a ton of coal extracted is still 5.80f., while that of the principal is no more than 0.95f.

The workman alone then has profited in the progress realized, progress due to the capital, which has had not only to create engineers and *savants*, but also to make the experiments, the experience, and the expensive schools. What has been for the capitalist the result of the ameliorations, of which the workman now enjoys increased wages and better welfare in his work? A loss of 58% in a period of 20 years!

Before reverting to the statistics of accidents in the collieries, we may point

out in the tables and diagrams the same steps in Belgium as in France, in so far as wages, cost of production, and market price are concerned.

However, in Belgium, that which we have just said on the subject of the social question in comparing the part of the capital to that of the workman is still more striking, for we see that in certain years the capital finds itself at a loss, and actually the profit per ton is no more than 0.41f.

Statistics of Accidents.—The statistics of the accidents contain much data of very unequal value. The number of workmen killed is known exactly; the number of wounded and the corresponding accidents are known less exactly. In principle, according to the instructions of the mines-administration, the statistics ought to include all the wounded who have suffered 20 days' incapacity of work. But there is much failure on the part of the colliery managers to make the necessary declarations to this effect. It is then only on the number of men killed that true statistics can be based.

STATISTICS OF ACCIDENTS—TABLE IV.

Year.	Number.		Number of Victims.					
			Mine.		Surface.		Total.	
	Mine.	Surface.	Killed.	Wounded	Killed.	Wounded	Killed.	Wounded
1889	597	79	322	531	12	69	334	600
1890	734	96	292	679	20	70	312	758
1891	715	97	191	646	29	72	220	718
1892	793	101	112	733	15	93	127	826
1893	763	111	105	692	19	93	124	785
1894	794	146	93	736	21	130	114	866

STATISTICS OF ACCIDENTS PER 1000 EMPLOYED, ACCORDING TO CAUSE—TABLE V.

		Cave-ins.	Fire-damp.	Shafts.		Blasting.	Underground Railways	Manual Labors.	Various Causes.
				Falls in Shafts.	Falls of Tubs.				
1889	Accidents..	35.9	0.8	3.4	0.5	2.5	20.9	4.3	7.4
	Killed	5.7	28.5	2.4	0.5	0.4	1.6	0.4	1.3
	Wounded	31.5	1.5	1.7	0.3	2.7	19.3	3.9	6.6
1890	Accidents	37.0	0.8	3.1	0.7	3.5	25.0	3.9	10.7
	Killed	7.1	13.5	1.6	0.1	0.7	3.3	.....	2.3
	Wounded	30.4	5.4	1.6	0.8	3.2	21.5	3.9	11.3
1891	Accidents	29.5	0.9	4.9	0.4	2.9	22.7	4.7	10.1
	Killed	6.3	6.9	2.4	0.1	0.4	2.6	.....	1.6
	Wounded	24.6	2.1	3.2	0.5	3.5	20.8	4.7	9.4
1892	Accidents	35.5	0.4	5.0	0.6	5.2	25.4	5.2	9.2
	Killed	6.0	.....	2.2	0.3	0.9	1.4	0.1	0.9
	Wounded	27.7	0.4	3.8	1.3	6.3	24.1	5.1	8.5
1893	Accidents	32.0	0.1	6.4	0.4	2.4	23.3	7.8	9.0
	Killed	4.7	.....	3.0	0.1	0.1	1.9	0.1	1.3
	Wounded	28.0	0.1	4.2	0.3	2.4	21.6	7.7	9.2
1894	Accidents	33.7	0.5	4.3	0.6	3.5	25.1	4.8	9.8
	Killed	5.3	.....	1.7	0.3	0.1	1.3	.....	0.9
	Wounded	30.0	0.8	3.0	0.4	4.3	24.0	4.8	9.1

The year 1889 was in France the worst of the period. Indeed that year was marked by an accident of exceptional gravity, which caused more deaths than in other mines during the whole year. At 11:30 A.M., July 3, a terrible explosion of fire-damp occurred in the Verpilleux and St. Louis mines of St. Etienne, where there were 213 miners. The explosion was heard on the surface. Only 6 miners were saved. This catastrophe surpassed in number of

victims any recorded in many years. That which particularly affects the diagram in 1876 happened at the Jabin pit of the Terre Noire mine, and had 186 victims. The works of the Verpilleux and St. Louis pits were wrecked for their whole extent, and the work of recovery lasted several weeks, a number of bodies being abandoned finally.

In 1890 an explosion at the Pélissier pit of the Villeboeuf mine was excessively grave, causing the death of 113 miners. It was due to the ignition by an unknown cause of a fire-damp mixture, accumulated in the mine without doubt on account of insufficient ventilation.

With respect to the accidents of fire-damp in Belgium, the year 1881 was marked by two explosions, one in No. 6 pit of the Marsinelle colliery, the other on December 8, at the Marie pits of the Cockerill colliery at Seraing.

In 1887 the number of victims of fire-damp was considerable because of two explosions in the Mons district; on January 4 at the Escouffiaux collieries (35 victims), and on March 4 at the Colliery of the 24 Shares (113 victims).

Finally, in 1892, the catastrophe of Anderlues had 169 victims, of whom 160 died, and that of the No. 2 pit of the Agrappe, caused by an instantaneous and violent disengagement of fire-damp, killed 35 miners.

Although the various coal centers of a single country have particular dangers, arising among other things from fire-damp, character of the coal-bearing strata, etc., it is interesting to compare the number of men killed with the total number employed in various countries. France appears to hold the best position, and England has also made sensible progress in point of security of the miner.

It would be unjust not to attribute the more part of the decrease of accidents to the incessant efforts of the colliery managers. The advance in the instruction of the master-miners and overseers, the execution of the rules of the mines-administrations, and the scientific study of accidents has contributed also to more security in the work of the miner.

In comparing the results of the last decennial period with those of the preceding it is proved that the annual production of the workman has increased. One arrives thus at the price of the coal which was fixed without value in the bosom of the earth. This sum of money, created by the intelligence of the engineer united to the two great forces of industry—capital and labor—represents annually in France 302,042,400f. with 134,600 persons employed, and in Belgium 200,158,000f. with 18,957 employees.

In France, without doubt because of the new methods of exploitation employed in the new basin of the Pas-de-Calais, which represents alone 41% of the French production, it appears indeed that the coal raised represents a more important sum than in Belgium for the same number of persons employed, and if the French colliery owner is under considerable charges (aid funds, hospital funds, workmen's lodgings, etc.), that the foreigner does not have to bear in the same proportion, he succeeds, by the less amount of labor employed and the development of the mechanical means (as in the United States), in occupying a place which reflects as much honor to the engineering schools as to the constant efforts made by him in a country where the coal-riches are very far from approaching those proved since the middle of the century in other parts of the world.

Modern Coke Ovens and Their By-Products.

By George Lunge.

The only rational method of coke manufacture is that which utilizes to their full extent the products of the destructive distillation of the coal instead of wasting them by burning at the mouth of the distilling apparatus, as is done in the antiquated "beehive" oven. The Germans were the first to practically develop by-product coke ovens on a large scale, although Carvès, in France, was the first to build and successfully operate coke ovens of this description. England and France, the largest coke producers in Europe, are considerably behind in the introduction of this branch of manufacture.

COKE OVENS UTILIZING THE HEAT OF THE GASES.

The first step in the way of improving the methods of coke making was the utilization of the gases and vapors issuing from the mouth of the coke ovens for the purpose of furnishing the heat required to effect the destructive distillation of the coal. Everybody knows that this heat in the ordinary "beehive" oven is produced by the burning of part of the coke itself, some air being admitted into the oven for that purpose. In this case the heat is, therefore, both produced and utilized within the oven, at the expense of a considerable part of the produce. In the improved ovens, on the other hand, the necessary heat is provided by burning the gaseous products of the distilling process outside the oven, the flame being made to travel round the ovens in flues, none of the coke itself being consumed and thus wasted in the process. The increase of output of coke in the improved process ranges from 10 to 15%, and sometimes even more.

Various types of ovens, constructed for this object, all of them very different in shape from the "beehive" pattern, have been known for about 40 years, and especially within the last 20 or 30 years they have come into very extended use in Belgium, Germany, and somewhat less in France, but they have obtained a very slight footing in Great Britain, or until quite recently in the United States. In these last-named countries, although the advantage of an increased yield could not be denied, nearly all coke-makers maintained the opinion that this advantage was more than counterbalanced by certain drawbacks, and that it was a bad speculation to give up their thoroughly tried "beehive" ovens for the new patterns, which in any case involved considerable outlay for plant, and perhaps also for repairs. Indeed, even in Germany until a few years ago a good many "beehive" ovens were still at work side by side with the improved ovens, but their number is constantly decreasing.

A few words will dispose of the alleged drawbacks of the modern coke ovens which utilize the heat of their gases. The principal objection, tenaciously upheld until quite recently by most of the leading English and American ironmasters, has been that the coke made in the new ovens is inferior to the beehive-oven coke, both in respect to crushing strength and fuel value. Some of these gentlemen roundly maintained that such coke was simply unfit for use in the blast furnace; others were at least positive that much more of it than of "beehive" coke was needed for making pig iron. These views might at any time have been refuted by a visit to some Belgian or Rhenish-Prussian iron works; but at that

time most of the gentlemen concerned would have scorned to admit that they could learn anything about iron making from continental people, and would have deemed such visits wholly superfluous. It is useless to waste time in refuting those utterly erroneous assumptions as to the inferiority of the coke from the improved ovens, for this has been already quite sufficiently done by Mr. Blauvelt in THE MINERAL INDUSTRY, Vol. IV. I may assume it as now fully known and generally acknowledged in the United States that the coke made in externally fired ovens is absolutely equal to "beehive" coke for the purpose of iron making, as well as for any other.

A second objection has been made to the new ovens, and this is somewhat better founded than the first, but it should not be stretched beyond its proper extent. It cannot be denied that externally fired coke ovens cost much more to erect than "beehive" ovens, and the difference is again greatly enhanced, if the appliances for saving the by-products are to be added, which we shall consider later. There are now about 45,000 "beehive" ovens in existence in the United States; their abolition would of course mean the annihilation of a very large amount of capital. There is something in the argument taken from this fact against the introduction of the new style, but not very much. It is a simple matter of calculation whether the interest on the capital required for building new ovens is greater or less than the profits corresponding to the increased yield of coke and that represented by the value of the by-products. This calculation will, of course, run very differently according to whether it is a question of pulling down existing "beehive" ovens and erecting new ovens in their place, or whether the manufacture of coke is to be started in some fresh place where the new style of ovens are more in favor. Data for judging of the comparative costs of erecting and working various systems of coke ovens have been given in Mr. Blauvelt's paper.

BY-PRODUCT COKE OVENS.

Very soon after the first invention of outside-fired coke ovens a very important further step was taken, viz., the recovery of by-products in the coking process. Instead of immediately burning the gases and vapors issuing from the mouth of the oven, and merely utilizing the heat of their combustion, the gases, etc., are first subjected to processes for extracting their tar and ammonia similar to those practiced in the manufacture of ordinary illuminating gas, and only the residual permanent gases are used as fuel in the coking process, for which, with proper appliances, they are far more than sufficient, so that the excess may be employed for steam raising and other purposes. A brief history of the development of this industry is given in THE MINERAL INDUSTRY, Vol. IV., pp. 216 and 217. The reader is also referred to the more complete account in my *Coal Tar and Ammonia*, second edition (1887), p. 38 *et seq.*, which, as I have reason to assert, is more correct in some details, though none of them are of much importance.

The priority in this field undoubtedly belongs to France, where Carvès, a little after 1856, altered the original externally fired "Knab" ovens in such a way as to recover tar and ammonia. A number of such ovens were built at Commentry (1862), Bessèges (1866–1873), and Terre Noire (1879); and these ovens have been at work there ever since. It is very remarkable that this example, open as

it was to the knowledge of the whole world, did not lead to any further imitation even elsewhere in France, or anywhere outside of France for a whole quarter of a century after the invention of Carvès. In 1879 Dr. R. Angus Smith, chief inspector of alkali works in Great Britain, in the *Fourteenth and Fifteenth Reports on the Alkali Acts*, p. 49 *et seq.*, minutely described the Bessèges plant, pointing out the immense saving which could be effected to British coke manufacturers by this system, and warmly recommended its adoption to his countrymen; but neither he, nor Mr. Henry Simon (a German engineer residing in Manchester), who in 1880 read a paper on this subject to the Iron and Steel Institute, at that time produced any visible impression in the United Kingdom, partly owing to the conservative spirit so frequently exhibited by Englishmen toward foreign inventions, partly in consequence of the direct, strenuous opposition made by some of the most highly respected English ironmasters to the quality of the coke made by the new process. It did not matter to them that these objections had been found to be unjust in the practice of a good many continental iron-makers, for this was held of no account in England at that time.

In 1881 Germany awoke to the importance of the Carvès process, and with characteristic skill and enterprise her technologists and manufacturers at once took the lead in the new departure, not merely by patenting a great many improved constructions of coke ovens fitted for the recovery of by-products, but by setting to work to carry that system into practice. Some hundreds of ovens of the new type were erected within the next few years, and thousands have followed since. It is calculated that there are about 4000 of these ovens in Germany at the present time; comparatively few of the old ovens are now left in that country, and no new "beehive" ovens have been built there for a number of years. Belgium has also made a good deal of progress in this direction, and so has France, although seemingly not to the same extent. But, in spite of numerous patents and public lectures, papers, etc., issued in Great Britain, very little real progress was made there with the new system, partly because some of these British patents were on the wrong tack, in attempting to retain the "beehive" ovens with some modifications, which were in all cases found to be altogether inadequate for producing the proper quality and quantity of by-products; in other words, because capital was too shy to resolutely go in for the new system, which entailed the entire abolition of the old plant. A few good plants on the new system were indeed erected in England, but although the reports made about the results obtained were very favorable, and were widely circulated, the causes alluded to above prevented the vast majority of British coke-makers from following that lead, and up to this date the state of matters has remained the same.

The readers of THE MINERAL INDUSTRY know well enough that the coke-makers of the United States, until a year or two ago, have been behind even their British colleagues in this matter. When visiting the Chicago Exhibition in 1893, I saw, of course, the pretty model of the Connellsville coke ovens in the Mining Building, with the gas flames representing the spectacle offered by the oven mouths on the large scale, and in the presence of Mr. H. Clay Frick himself I was asked if I did not admire that exhibit; but I was bold enough to say in that gentleman's presence that I could not bestow any admiration upon the demonstration of an utterly irrational and antiquated method of conducting a technical

process. I was then informed that not a single coke oven of the novel type was in existence in the United States, and the same objections were repeated to me which had been made and refuted over and over again in Europe. To this I could, of course, only shrug my shoulders. I had then, as now, no personal interest whatever in the matter and so left it alone.

Things have improved since, but how much is still to be learned in this respect can be seen from Mr. Blauvelt's paper in THE MINERAL INDUSTRY, Vol. IV. The reader will there find a very good account of some of the principal types of modern coke ovens as they exist in Europe, and to a small extent also in the United States; he will there also find details about the cost of erection and of working these ovens, forming a very welcome supplement to the statements and diagrams to be found in my *Coal Tar and Ammonia.*

I am in a position to bring all this up to date by communications which I have received from Mr. Hilgenstock, one of the foremost German metallurgists, who is now connected with the Otto Company. Part of the information to be now given is contained in a lecture delivered by Mr. Hilgenstock, another part in a German patent taken out quite recently, and part is derived from letters and other written documents which I have received permission to use in this place.

THE OTTO-HOFFMANN IMPROVED PROCESS.

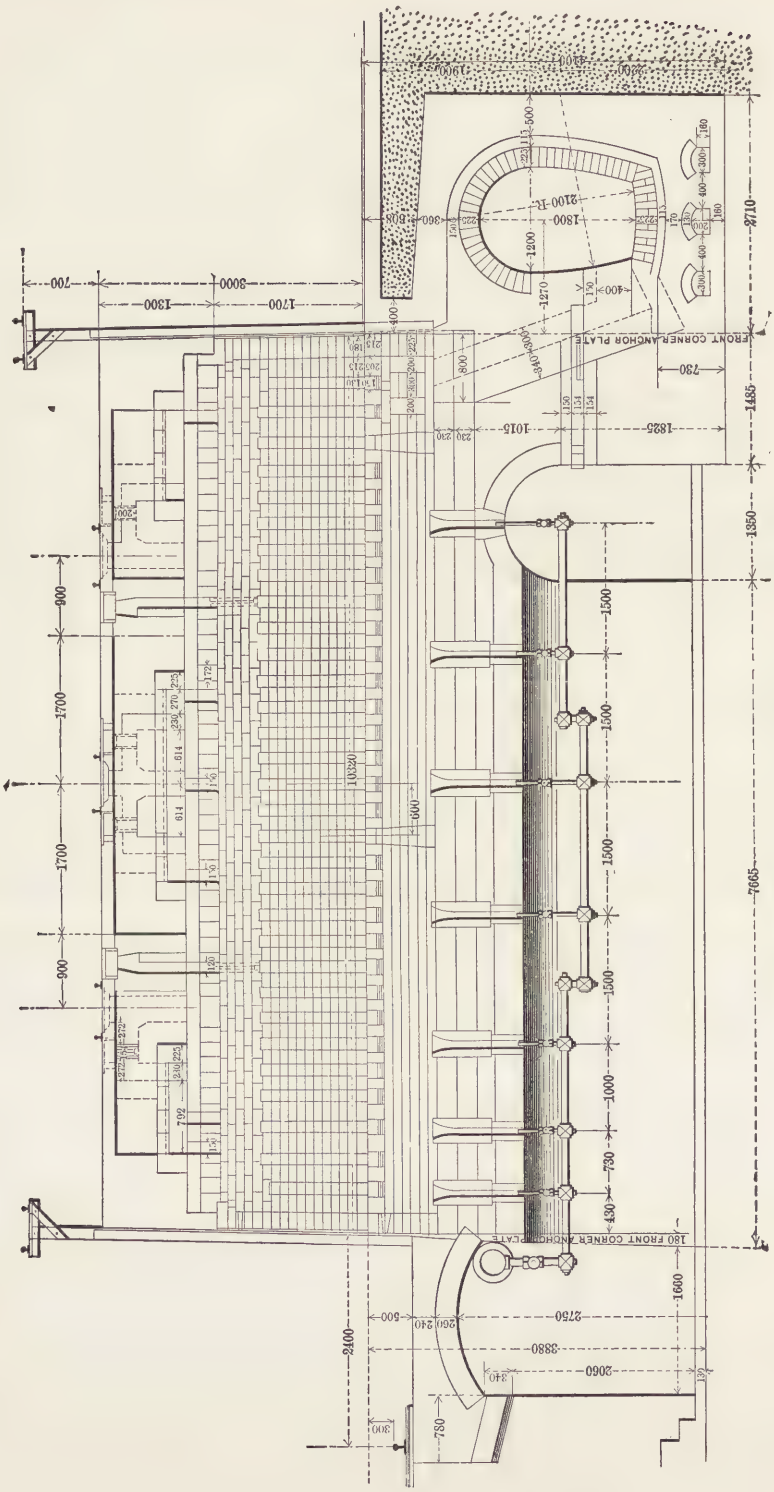
Before everything else we must consider that the recovery of the by-products involves a considerable loss of heat, which we may calculate as follows: The gases evolved from coke ovens will never be as free from extraneous products as those from the same description of coal in an ordinary gas retort, consisting of a single piece. Coke ovens are built up from firebrick, which cannot possibly be joined in an absolutely air-tight manner, and a little air will thus always penetrate through the joints. We may take it that a metric ton of ordinary coking coal yields from 280 to 300 cu. m. of gases, containing from 50 to 55% hydrogen, 35 to 40% methane, 2.5 to 3% heavy hydrocarbons, 6.5% carbonic oxide, and 1.5% carbon dioxide; together with 2.5 kgms. of pure ammonia (equal to 10 kgms. of sulphate of ammonia), and 30 kgms. of tar. In order to recover the tar and ammonia we must renounce burning the gases in the hot state and must cool them down. Taking the heating value of the pure oven gas as 4500 calories per cu. m., each kilogram of coal furnishes 1350 calories, of which 200 calories (15%) are lost in the cooling process. Another loss is involved in the necessity of conducting the operation more slowly, in order to avoid the internal burning of coal as much as possible, and this we may put at 150 calories per kgm. of coal, or 11%. The heating value of the tar itself is 190 calories, or 14%. The total loss of heat in the recovery of the by-products is thus $15+11+14=40\%$ of the total heat of the gases as they leave the ovens, a loss which is avoided where the gases are burned immediately in the flues surrounding the ovens. The remaining 60% should be still more than sufficient for carrying on the coking process, but it is evidently not quite so easy to do this as when the recovery of the by-products is not demanded, and this led inventors to apply the well-known principle of heat recuperators. Various kinds of such apparatus have been tried, but the old Siemens recuperators, which were adopted in the Otto-Hoffmann ovens, seem to have answered best in the long run. The great success of the Otto ovens seems to have been due to still another cause,

viz., to their adopting the Coppée system of many short vertical flues, acting simultaneously, while the Carvès, Carvès-Simon, Hüssener, and Semet-Solvay ovens possess long, horizontal, zigzag flues. In the Coppée ovens the gases have a much shorter time to stay in the heating flues, and they are exposed to much less friction. Now, unless the gases within the proper coking chamber (the oven) and those traveling in the outside flues are exactly under the same pressure, it is impossible, looking at the unavoidable leakage of the joints, to avoid either the entrance of fire gases into the oven, which means burning coal instead of distilling it, and at the same time destroying part of the tar and ammonia, or else the escape of oven gases into the fire flues, which means the loss of the corresponding amount of tar and ammonia through the chimney. It is practically impossible to work the exhauster in such a manner as to maintain a perfect equilibrium of the gases inside and outside the ovens throughout the flues. Owing to frictional resistance the flue gases will be always under a plus pressure in some places and under a minus pressure in others; and this irregularity must increase with the length of flue the gases have to traverse. The longer the route of the gases the better the opportunity of utilizing their heat, but the worse the chance of recovering a maximum of by-products; and as there is a considerable amount of waste heat in all good modern ovens, the latter saving is much more important than the former.

These theoretical considerations are borne out by the fact that the Otto ovens are increasing in numbers faster than others. In Germany there exist only about 100 Hüssener ovens, and 48 Semet-Solvay ovens, with a few ovens of other systems, but in 1895 there were already 2237 Otto ovens at work, and in 1896 480 new ovens were erected by the Otto Company in the Ruhr district alone, to the exclusion of all other systems.

As far as the style of work is concerned, it would be very unfair to compare the best new ovens of other systems with the Otto ovens of 10 years ago, for the latter have been greatly improved in the course of time. Otto ovens of 10 m. length and 600 mm. width require only 40 hours for finishing a charge; those of less width can be worked in 30 hours and less. Thus, for instance, a set of 60 ovens at the Hibernia pit, each 10 m. long, 1.8 m. high and 0.530 m. wide, has a daily output of 240 tons of coke, and a monthly output of 100 to 110 tons of sulphate of ammonia and upward of 300 tons of tar.

The following improvements have been made in the newer ovens. The heating gas and air are conveyed in two different channels, the air is heated to 800° C., and the combustion divided over a large space. A fresh supply of heating gas is continually supplied in the center of the top flue. The coking chambers a made narrower, and the heating flues wider than before, causing less friction for the fire gas, and thus increasing the quantity of by-products. Sometimes, where the nature of the coal calls for it, the ovens are provided with a double row of heating flues between each two ovens, formed by placing two simple walls side by side, in a manner somewhat similar to the Semet-Solvay ovens, but differing from these especially in the way the flues are carried vertically, not horizontally. This arrangement permits of taking one of the walls away for repairs, leaving the other one standing; it also allows of the walls expanding and contracting independently. The new ovens have twice the output of the old ones, and it is maintained that no other system comes near them.



NEW FORM OF THE OTTO-HOFFMANN COKE OVEN.

The greatest improvement in the Otto ovens is that which has been recently protected by the German patent No. 88,200. Starting from the principle of shortening the path of the fire gases still more than in the former ovens, a number of gas burners are arranged all along underneath the ovens in underground passages. The Siemens recuperators, which involve the trouble of constantly changing the direction of the draught, are abandoned, the air enters without artificial pressure, and is heated up in the foundations of the ovens in an extremely simple manner. The pressure in the fire flues being now reduced by 2 mm., much less air enters the ovens through the chinks of the walls, the oven gases are much purer than before, and the yield of by-products is correspondingly increased. The charge is also coked in a shorter time. Already upward of 400 such ovens are finished, and it is claimed that they have surpassed even the expectations of the inventors themselves. At all events there is no doubt that this is one of the best, if not the very best, type of ovens now existing. It is shown in the annexed drawing.

RECOVERY OF BENZOL FROM COKE-OVEN GASES.

A last and most important step is the recovery of benzol from the gases as they leave the apparatus in which tar and ammonia are condensed, before the gases are conveyed back to the coke ovens in order to heat these by their combustion. This idea, self-evident as it now appears to any expert in this matter, was not discovered, or at least not clearly enunciated and carried out in actual manufacturing practice, until, in 1889, once more Carvès took the initiative, and accomplished its realization at a French works on a comparatively small scale. This beginning was immediately followed on an immense scale in Germany, which up to this time had a practical monopoly of this system. Nothing more rational can be conceived. It had been known for a long time that only a small proportion of the benzol and toluol, evolved in the destructive distillation of coal, is found in the tar; by far the greater portion remains in the state of vapor, and we know perfectly well that the illuminating power of ordinary coal gas as it is made in retorts is principally due to these hydrocarbons.

It is shown in my *Coal Tar and Ammonia*, p. 31, that the gas made from a certain quantity of coal contains twenty-three times as much benzol and toluol as the tar produced in the same operation. The idea of extracting the benzol from ordinary coal gas is very old, and has been several times tried on a large scale, as described in my book; but it had to be abandoned every time, for the very good reason that the gas, after the extraction of the benzol, is worthless as an illuminant. But the matter is entirely different with coke-oven gas. This gas differs from retort gas in containing a smaller proportion of illuminants, due in a great measure to the difference in the quality of the coal employed, and is therefore not so well adapted for lighting purposes. But then it has never been devoted to that object, nearly all of it being simply utilized as fuel. The benzol and toluol contained in it amount to only about 1% (by volume), and their abstraction from the gas but slightly reduces its fuel value. But since the modern ovens are provided with heat regenerators, the coke works obtain much more gas than they can use in heating the ovens and raising the steam required in the manufacture, including the distillation of the ammonia from the aqueous liquor.

The value of the benzol, less the cost of extraction (which is not very considerable), is therefore a clear gain. Nor was it necessary to seek very far for extraction processes, as these had been fully worked out in connection with the previous attempts at treating ordinary coal gas, and had been described in my book several years before they were first applied to coke-oven gases. In practice two processes have been employed for coke-oven gases—condensing the benzol, etc., by cooling the gases under pressure, or absorbing it in heavy mineral oils, and recovering it by distillation, leaving the absorbing oils behind to be used over and over again.

As already remarked, the Germans were not slow to carry out these processes on the largest scale, both in the Westphalian and the Silesian districts; and such enormous quantities of benzol soon appeared in the market that the price of this article was forced down to about half of the lowest figure ever before touched. Indeed, some of the benzol-extracting plants were temporarily stopped, in order to cause the price to rise again, which it did. It is now certain that even if the manufacture of water gas should become much more general than it is at present in Europe, and if thereby the quantity of ordinary gas tar, available for manufacturing those necessary raw materials for the industry of artificial coloring matters, were greatly diminished, the supply of tar from the coke works would be equal to, and that of coke benzol even far in excess of, the demand of the aniline color works, and of all other consumers of coal-tar products.

Quite new uses are already found for benzol in consequence of its cheapness, such as raising the illuminating power of ordinary retort gas, thus doing away with the necessity of employing cannel coal or other enrichers in the gas-making process.

It is true that in America the consumption of the products extracted by distillation from coal tar, including benzol, has been hitherto greatly inferior to that of the industrial European nations, and that the whole industry of coal-tar distilling has flourished but little in the United States. But nobody doubts that the time will come when the finer chemical industries, for which the above-named products furnish some of the most important raw materials, will be cultivated in America to the same extent as they are at present in Germany or France or England. Indeed, as a gas enricher it is probable that there would now be a market in the United States for all the benzol which could be produced from coke-oven gases if it could be sold at a reasonable price. It should therefore be of sufficient interest to American readers to learn rather more of the quality of the by-products of coke making. Indeed, the excellent information which Mr. Blauvelt gives about the construction of coke ovens, their cost, etc., is only an introduction to the study of the important industry which the products recovered by these new ovens have given birth to. I shall endeavor to supplement the information given by Mr. Blauvelt by treating of the tar, ammonia, and benzol obtained as by-products from coke ovens.

COKE-OVEN TAR.

It has been very emphatically pointed out in my *Coal Tar and Ammonia*, and is also mentioned in Mr. Blauvelt's paper, that the tar obtained from a certain class of by-product coke ovens (namely, from all those which are merely

adaptations of old "beehive" ovens, not entirely discarding internal firing) does not at all resemble ordinary gas tar, but is more of the nature of an oil, and is useless for the purposes to which gas tar usually is put. No benzol, anthracene, etc., can be obtained therefrom, as is also the case with the tar obtained in Scotland from blast furnaces worked with splint coal, which is one of the reasons why all ovens of that class have turned out failures economically.

The case is quite different with the tar recovered by means of the Carvès-Hüssener, the Carvès-Simon, the Otto-Hoffmann, the Semet-Solvay, and other "horizontal" ovens, forming long, narrow chambers, which are worked at a very high temperature, like gas retorts. The tars here produced are similar to ordinary gas tar—sometimes almost exactly like it. Inferior to this, but still immensely superior to the tar from imperfectly adapted "beehive" ovens, is that which is made in ovens similar to the latter class, but heated exclusively from the outside.

The following analyses are from various quarters. No. 1 is by A. Sadler; Nos. 2, 3, and 4 are by Watson Smith; Nos. 5 and 6 by Knublauch; Nos. 7 and 8 by myself (in conjunction with Jac. Schmid).

No. 1.—Tar made from Pease's West coal (Durham) in Carvès-Simon ovens:

First distillation—	Per Cent.		Per Cent.
Light oil	4.50	Solvent naphtha	0.60
Creosote oils	20.40	Heavy naphtha	0.40
Thick anthracene oil	34.20	Crude carbolic acid	0.05
Further treatment—		Creosote oil	46.50
Water	10.00	Anthracene	0.74
Benzol, 50/90%	0.50	Pitch and loss	41.21

If this analysis is correct this tar would be of somewhat inferior quality. The large quantity of water is very singular.

Nos. 2, 3, and 4 represent tar obtained in coking at Bear Park, Durham, England, with Carvès-Simon ovens:

	No. 2.	No. 3.	No. 4.		No. 2.	No. 3.
First distillation—	Per Cent.	Per Cent.	Per Cent.	Further treatment—	Per Cent.	Per Cent.
Ammonia water	.....	.....	7.44	50/90% benzol	0.70	.....
Light oils	5.12	5.12	6.28	Benzol boiling, 80° to 100° C.	.....	0.53
Creosote oil	7.44	.....	11.99	Tuluol, 100° to 120° C.	.....	0.29
Naphthalene	.....	.....	10.17	Solvent naphtha, 90%	1.03	1.25
Thick anthracene oil	34.03	.....	.....	Burning naphtha, 30%	0.43	0.20
Crude anthracene	.....	.....	11.57	Crude carbolic acid	0.30	.....
Naphtha	3.05	.....	.....	Crude cresol	0.35	.....
Pitch	45.17	.....	57.29	Crude naphthalene	.....	1.00
				Anthracene, 28%	2.40	.....

This tar, although obtained by the same kind of ovens, and at a locality not far distant from Pease's West coal, is much better than the tar from the latter (No. 1), owing probably to its being obtained at a somewhat higher temperature.

As can be seen from the statements here reproduced, the foregoing were not quite complete analyses, even in the sense of that word here employed. The following analyses are far more complete.

Nos. 5 and 6.—Tar made in coking the same description of Westphalian coal, but No. 5 in ordinary gas retorts, No. 6 in Otto-Hoffmann coke ovens.

	No. 5. Gas Tar. Per Cent.	No. 6. Coke-Oven Tar. Per Cent.
Water	2.90	2.20
Light oil up to 200° C	4.00	3.40
Benzol for aniline	0.92	1.10
Solvent naphtha	0.20	0.32
Creosote oil	8.60	14.50
Crude naphthalene	7.40	6.70
Anthracene oil	17.40	27.30
Pure anthracene	0.60	0.70
Pitch	58.40	44.30
Carbon in pitch	15 to 25	5 to 8
Yield of tar on coal	.....	3.00

According to these analyses, which proceed from a thoroughly competent source, the tar made from the same kind of coal in the Otto-Hoffmann ovens is at least equal to or indeed rather better for the purposes of tar distillers than that obtained by treating the coal in ordinary gas retorts; but the yield of 3% of tar on the coal is low, the yield in gas-making being rarely under 5%.

Nos. 7 and 8.—Tar made from Westphalian coal. No. 7 in Otto-Hoffmann horizontal ovens (the ordinary kind, as described in Mr. Blauvelt's paper); No. 8 in "beehive" ovens, adapted by the same firm to external firing.

	No. 7.	No. 8.		No. 7.	No. 8.
First distillation—	Per Cent.	Per Cent.	Final product—	Per Cent.	Per Cent.
Light oil up to 170° C	7.43	4.88	Aniline-benzol	1.68	1.12
Middle oil up to 230° C	10.54	14.14	Heavy naphtha	3.64	2.06
Creosote oil up to 270° C	7.62	9.99	Crude naphthalene	7.69	1.53
Anthracene oil	44.35	22.65	Crude phenols	2.23	6.93
Pitch	30.55	43.41	Anthracene, 33%	1.59	0.72
Water	Trace.	3.84	Creosote oils (remaining)	8.24	15.67
Loss	0.39	1.09	Anthracene oil (filtered)	42.76	21.93
Specific gravity at 15° C	1.1198	1.1368	Pitch (very hard) 165° C. fusing p't	30.55	43.81
			Water	.....	3.84
				98.38	97.21

These analyses, made with the greatest care in my laboratory, according to identical methods, are very instructive. Tar No. 8, which was obtained from modified "beehive" ovens, is decidedly inferior in every respect to No. 7, which was made in horizontal ovens. This is still further accentuated by the observation that the "aniline-benzol" in No. 8 contains but little real benzene, and a notable proportion of non-nitrifiable hydrocarbons, of which the benzol No. 7 was free. Tar No. 8 was also found to be dehydrated with great difficulty, while this was easily effected with No. 7, as shown by our analysis. For all this, the quality of No. 8 still permits working it up along with No. 7, or, which comes to the same thing, with ordinary gas tar.

The final conclusions to be drawn from these analyses are: That the tar produced in externally fired ovens of the long, narrow type, in which the temperature is at least as high as in the gas retorts, and the work is carried on quickly, is, to all intents and purposes, equal in quality to ordinary gas tar; it has indeed been worked together with the latter by the German tar distillers all along, and enjoys an excellent reputation. To this kind of ovens belong the Carvès oven in its various modifications, the Otto-Hoffmann, and the Semet-Solvay. I have not

come across an analysis of the tar made from Semet-Solvay ovens, but according to all analogies it should be similar to that from the Carvès or the Otto ovens. Next in quality is the tar from such "beehive" ovens as are so far modified as to be heated exclusively from the outside, to which category belongs No. 8. High temperature is essential to the production of good tar, and owing to the fact that the temperature in such modified "beehive" ovens is decidedly lower than in the long, narrow ovens the tar is inferior to that from the latter. Still it retains the general characteristics of gas tar, while the tar made in modified "beehive" ovens, which retain to a certain extent internal combustion, as well as the tar obtained from the Scotch blast furnaces, is of a totally different kind, and not to be classed with gas tar at all. It has at present hardly any value except as fuel.

COMPOSITION OF BENZOL FROM COKE-OVEN GASES.

The recovery of benzol from the gases remaining after the condensation of tar and ammonia from coke-oven gases is the most recent achievement in that field, and the details of this industry are as yet surrounded with a great deal of mystery. This fact is probably owing to the circumstance that it was not found possible to obtain a good patent for the extraction of benzol from coke-oven gas. It is too evident that this gas, although not identical with retort gas, is obtained in an altogether similar manner, and is in other respects entirely analogous to it. Seeing that so many proposals had been made, and partially carried out, for extracting benzol from retort gas, there was no actual novelty in transferring that operation to coke-oven gas. Probably more than one before Carvès had conceived the idea, or even uttered it, to treat coke-oven gas by one of the various methods formerly described for the extraction of benzol from retort gas, patented or otherwise, but there seems to be no doubt that Carvès was the first to do this in actual practice, and that the coke-gas benzol shown by him in the Paris Exhibition in 1889, for which he obtained a medal, was really the first produced in this manner. It seems, however, that his operations were only carried out on a restricted scale. The special method followed by him has not been made public, and the German works, which soon after took up this industry on a very large scale, have also kept their methods rigorously secret—perhaps all the more as they must have realized very substantial profits. They may have justly assumed that every effort was required to keep this new industry within moderate limits, lest by its general adoption the price of benzol would be forced down to an intolerably low limit. This was actually the case during a certain time, notwithstanding all their precautions, and prices have only improved since the adoption of benzol as a coal-gas enricher, which has certainly caused a very large increase in the value of that product.

The secrecy in this field has indeed been carried to a very unreasonable length in withholding from scientific men samples of coke-oven benzol, when asked to supply it for scientific purposes. For two or three years I have made unsuccessful endeavors in this respect. I was unable to procure an authentic sample of this article, either because the factories would not own to its origin, preferring to sell it in the same market as ordinary benzol, or because they were unwilling to run the risk of the quality being found to differ from ordinary benzol. At last, in 1894, I succeeded in obtaining authenticated samples of two different brands of

coke-gas benzol, several gallons of each, so that I was enabled to make a thorough examination of these products. One of these (No. 1) was made by "absorption," the other (No. 2) by "compression."

In the *Zeitschrift für angewandte Chemie*, 1894, p. 637 *et seq.*, I described the analytical methods and results, applied for this purpose, in all details; here I shall only quote the ultimate composition of the benzols in question.

No. 1.—Benzol obtained by absorption. This is a clear liquid of tarry smell, sp. gr. at 16° C.=0.875. It commences to distil at 18°; up to 150° C. nearly 98% have passed over.

Benzol No. 1.	Per Cent.
Benzene	85.10 by weight.
Toluene	11.63
Xylene	1.54
Higher homologues	0.09
Non-saturated fatty hydrocarbons	0.41
Bases (pyridine, etc.)	0.08
Phenols	0.08
Thiophen	0.46
Carbon disulphide	0.01
Methylisocyanide	0.0024
Mercaptanes	0.00018
Residue of distillation	0.62

Benzol No. 2.	Per Cent.
Benzene	67.03 by weight.
Toluene	15.61
Xylene	2.18
Higher homologues	6.41
Naphthalene	3.79
Higher boiling aromatic hydrocarbons	1.42
Non-saturated fatty hydrocarbons	0.96
Thiophens	0.71
Phenols	0.11
Bases (pyridine, etc.)	0.28
Residue (pitch)	1.50

We must call No. 1 an excellent sample of crude benzol, far superior to most grades of "crude naphtha" from coal tar.

No. 2.—Benzol made by compression. This is a much darker liquid than No. 1, very much like crude coal-tar naphtha (first runnings). Sp. gr. at 16° C.= 0.893; commences to distil at 30°; up to 320° C. 96.5% had passed over. This sample is very different from No. 1, and much less valuable; but it would still figure in the market as first rate coal-tar naphtha.

No. 3.—I also examined a "washed benzol," made at a Silesian factory from coke-gas benzol, obtained by the absorption method. Sp. gr. at 16° C.=0.883. It is colorless, of a pure smell. A careful examination proved the entire absence of basic substances, phenols, unsaturated hydrocarbons of the fatty series and carbon disulphide. The oil was easy to nitrate completely, without leaving a residue of non-nitrifiable substances. It began to boil at 78° and at 97.5° C. it yielded 90%, by volume. It is therefore a first-class sample of commercial so-called 90% benzol.

It is proved by the preceding analyses that the benzol extracted from coke-oven gases is absolutely equal in all respects to the very best grades of benzol made in the ordinary way from coal tar. Now, as far as the proper use of coke-oven gases is concerned, namely, for heating the coke ovens and raising all the steam required at the coke works (including the driving of the machinery for washing the coal, preparatory to coking), the removal of the benzol causes no perceptible loss whatever. Its heating power cannot amount to more than a few per cent. of the total gases, and this loss is of small consequence, as there is always much more than that going to waste, not being required at the works. If at any future time the proposal should be realized, to revolutionize the treatment of bituminous coal by coking it at or near the pits, selling the coke as fuel instead of the raw coal, and carrying the gases to more or less distant places to be used in the manner of natural gas, this plan would not be interfered with in any way by the previous extraction of the benzol. Its removal would not affect the heating value of the

gas to an appreciable amount, and even if some of such gas was destined for lighting purposes it could be just as easily carbureted with petroleum naphtha, as is at present done with water gas or natural gas.

Probably at the present moment there is not in the United States a sufficient market for the immense quantities of benzol which can be extracted from coke-oven gases, unless it should be found to be cheaper for gas enrichment than petroleum naphtha. There is, however, very little doubt that at no distant future plenty of uses will be found for this benzol, and entirely new industries might be introduced on the strength of obtaining such a valuable substance at a low price. At all events it seems appropriate to direct the attention of both technical and business men to such possibilities.

The yield of benzol is certainly not as high as stated by Mr. Blauvelt, viz., 1% of the weight of coal (which is an impossible figure), but it amounts to 0.3% up to 0.4% of crude benzol, containing 93½% of 90% benzol, according to the results obtained in several years' practice in Germany.

AMMONIUM SULPHATE AND CAUSTIC AMMONIA.

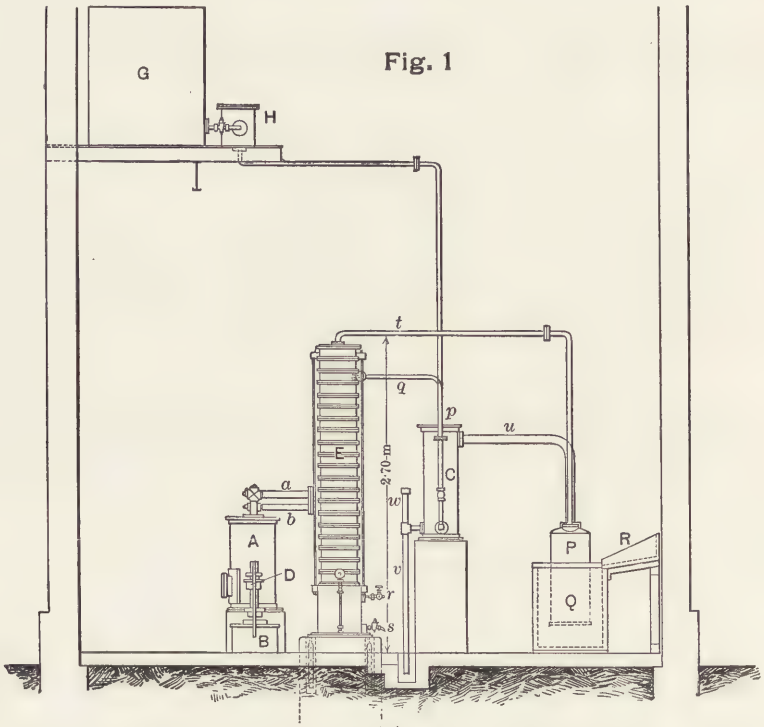
Although the manufacture of both caustic ammonia and sulphate of ammonia is well understood in America, and is, moreover, fully described in my book on *Coal Tar and Ammonia*, it may be of interest to give here a description of the most modern style of apparatus for the manufacture of both these articles, according to the system of Heinrich Hirzel, of Plagwitz-Leipzig, which is not described either in my own book or in Mr. Blauvelt's paper.

We begin with the manufacture of sulphate of ammonia from ammoniacal liquor, such as is produced in the manufacture of coke by the Otto-Hoffmann or similar processes, as well as in the ordinary manufacture of coal gas. The apparatus is shown in figures 1 and 2. It consists of the distilling column *A*, the lime mixer *B*, the lime pump *D*, the preliminary heater *C*, the high-level tank *G*, the inflow regulator *H*, the acid tank *Q*, the lead bell *P*, the draining stage *R*, and the necessary connections. The style of work is as follows: The gas liquor flows out of *G*, the flow being regulated by means of *H*, so that a regular jet, according to the capacity of the apparatus, enters at *n* into a worm, contained in the heater *C*. It issues at the top through pipe *p*, and through *q* it enters into the column *E*, in which it is brought into intimate contact with steam, meeting it as introduced by valve *r*. While the steam ascends the column, the ammoniacal liquor gradually flows in the opposite direction, descending from chamber to chamber, and thus yields to the steam all its free ammonia, as well as that which it contains in the state of carbonate and sulphide. This is accomplished about midway down the column. Here the liquor is turned away through pipe *a* into the lime mixer *A*. Here it is mixed with milk of lime from tank *B*, by means of pump *D*, and now re-enters the column through pipe *b*. In the lower half of the column the ammonia present in a combined state is liberated by the lime, and is fully expelled by the steam, so that the liquor, when it issues at the bottom through *s*, is entirely spent.

At the top of the column a pretty hot mixture of steam and ammonia issues through *t*, and passes into the lead bell *P*, which dips into sulphuric acid, filling tank *Q* to about half its height. Here the ammonia is entirely retained by the

acid in the state of sulphate. At the top of the bell hot vapors escape through pipe *u* into the heater *C*, where they are cooled by fresh ammoniacal liquor, rising in a set of pipes from *n* to *p*. In this way the fresh liquor is heated up

Fig. 1



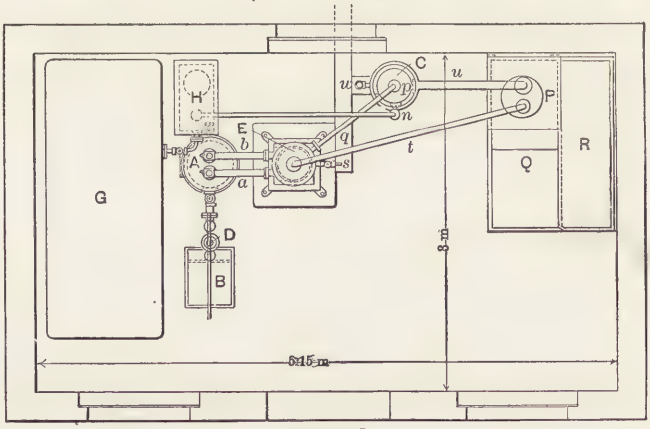


Fig. 2

at the same time as some liquid is condensed from the vapors in the worm, and runs away into a drain at *v*. The non-condensable part of the vapors, which has a very bad smell, is carried away into a chimney, or into a fire where it is burned. (This mode of disposing of the waste gases from ammonia stills is not practicable

in populous districts.)* During the operation crystallized sulphate of ammonia separates in *Q;* it is scooped out with a copper ladle, and the consumed acid is replaced. The crystals are drained on the stage *R,* and are then ready for the market. The work is carried on continuously.

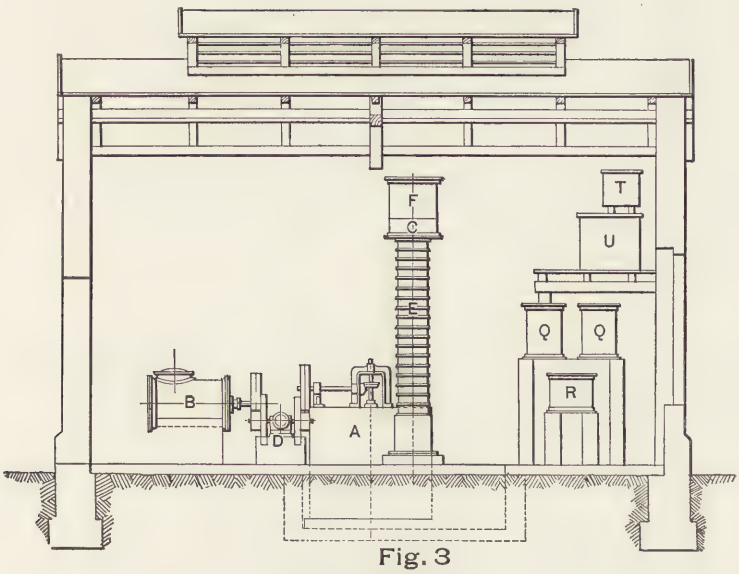


Fig. 3

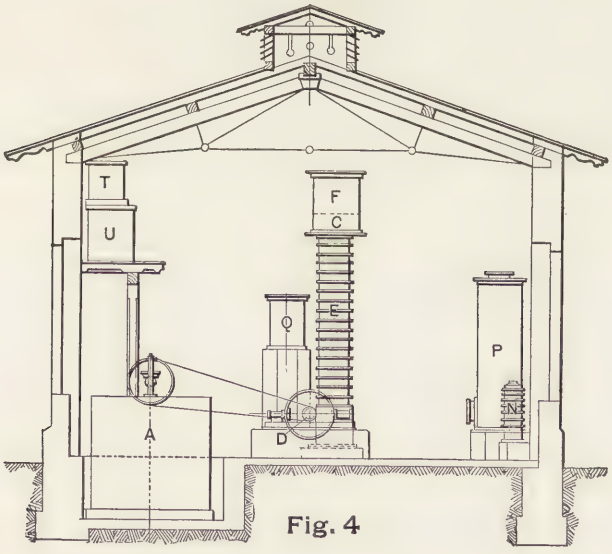


Fig. 4

The apparatus, as shown in the diagrams, will serve for a small production, say for working 3 tons of liquor in 24 hours. The firm builds apparatus for handling 100 tons of liquor daily and upward, with columns of 2 m. diameter. In the large apparatus the lime mixer is constructed in such manner that the excess of

* For full directions for disposing of the noxious gases in such cases, see *Coal Tar and Ammonia.*

lime is removed and cannot obstruct the lower part of the column. The large apparatus is also provided with an arrangement for utilizing the boiling hot spent liquor, leaving the column at *s* for a preliminary heating of the fresh liquor. This avoids sending a large quantity of hot, steaming liquor into the drains. In this case the heater *C* is removed to the foot of the column and the hot vapors escape directly from the saturator *P* into the chimney. I cannot approve of dispensing with the cooling of the vapors from the saturator; without such cooling it is impossible to remove their smell by burning, or to deal with them in any other reasonable way.

While the apparatus just described does not differ essentially from others previously known, Hirzel's apparatus for the manufacture of caustic ammonia (liquor ammonia) is a very complete piece of work, and especially distinguished from others by a new style of distilling column. The general arrangement of the apparatus is shown in Figs. 3 to 5, the details of the column in Figs. 6 to 8. *A* is

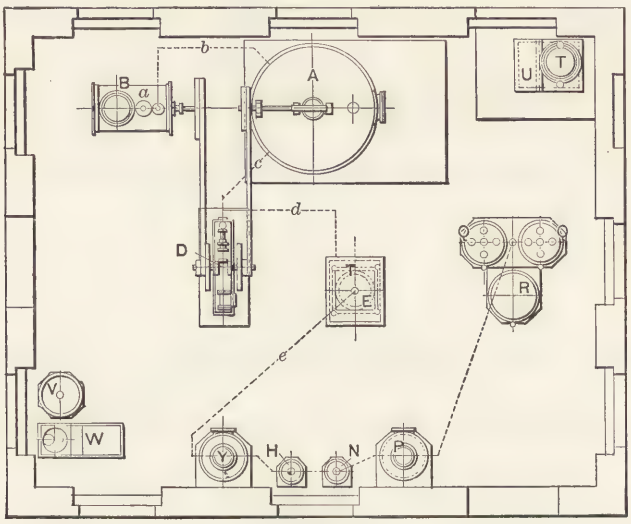


Fig. 5

the decomposing vessel; the large-sized apparatus is provided with two vessels of this kind, to secure continuous work. *B* is the lime mixer, provided with stirring gear; *D* is a heater (dephlegmator); *F*, the cooler belonging to it; *P*, a pump for the mixture; *E*, the distilling column; *G*, the coke purifier (this is duplicated for large plants); *H*, the permanently acting soda purifier; *N*, the similarly acting oil purifier; *P*, the bone-char purifier (duplicated for large plant); *Q* and Q^1, absorbers, provided with powerful cooling worms; *R*, a collecting tank. Cooler *T* and tank *U* serve for preparing distilled water for the absorption from the boiler steam. The small steam still *V* and acid saturator *W* serve for working up the escaping ammonia into sulphate.

The ammoniacal liquor to be treated is supplied either from a high-level reservoir, or by means of a pump, and enters the lime mixer *B* at *a*. This vessel had been previously charged with slaked lime, in the form of thick putty, about 5 or 6 parts for each 100 of liquor. Pump *D* moves the mixture on to vessel *A*,

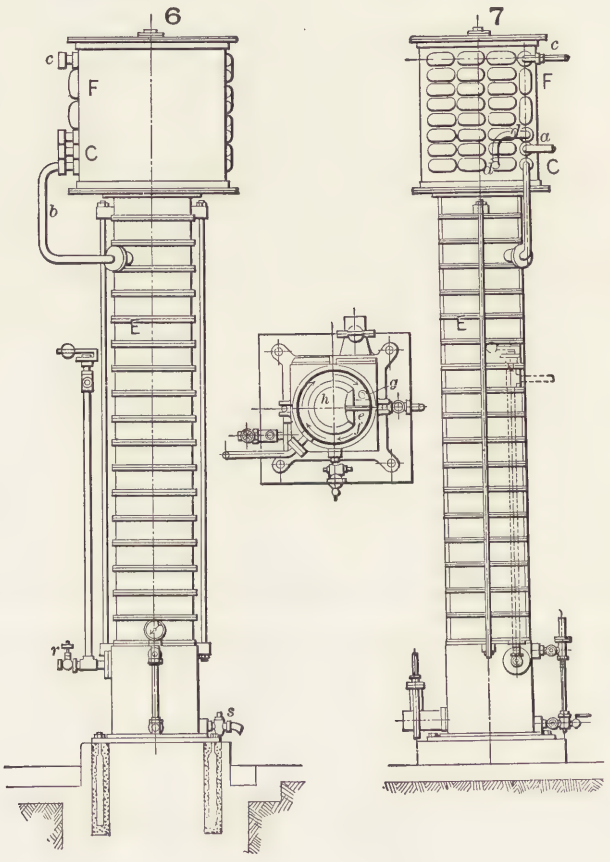
and as both *B* and *A* are provided with stirring gear, the liquor is intimately mixed with the necessary quantity of lime for decomposing all the ammonia compounds; also the carbonate and sulphide (in distinction to the previously described apparatus for producing sulphate of ammonia). This requires a slight excess of lime, and is promoted by keeping the temperature in *A* at about 40° C. by means of a little steam. When a charge has thus been prepared in *A*, ready for distillation, it is forced by pump *D* through pipes *c* and *d* in a regular stream into the heater *C* at the top of the column *E*, which had been previously heated up by steam. From *C* the mixture flows downward, and on its way is constantly in contact with steam, this contact being made most intimate by the construction of the single chambers, so that the liquor leaves the column at the bottom, completely deprived of its ammonia. At the top the mixture of steam and ammonia passes first through the heater *C*, in which it effects a preliminary heating of fresh liquor, and then enters the cooler *F*. Here the aqueous vapor is condensed by external cooling, forming a weak solution of ammonia, which re-enters the column; at the top of the cooler nearly dry ammonia escapes through *e* into the coke purifier *G*, where it is deprived of the liquid particles mechanically carried over. It now enters the purifying column *H*, where it is purified by means of a solution of caustic soda from sulphur compounds, carbolic acid, etc. A second column purifier *N* is charged with mineral oil, which retains hydrocarbons, etc. The last purifier *P*, charged with animal charcoal, delivers the gas from empyreumatic substances, and it now enters the absorbers *Q* and Q^1, where it is dissolved in distilled water. These two absorbers are connected by a two-way cock, and are worked in such manner that a solution of pure ammonia of the requisite strength is formed, which is run into tank *R*, ready to be sent out in carboys, etc.

By the same apparatus anhydrous liquid ammonia can be manufactured. In this case the gas issuing from *P* is first passed through an apparatus filled with a substance by which the last moisture is removed (dry lime), and is then passed into a small gas holder, serving as regulator. From here it is pumped away by means of an air compressor into a cooling apparatus, under such pressure that it condenses into a liquid, which is then sent out in the ordinary iron bottles.

The most important part of all this apparatus is the distilling column. Figs. 6 to 8 show this column on an enlarged scale, and of the shape intended for liquor ammonia; if sulphate of ammonia is to be manufactured, a modified column is employed without the heater *C* and cooler *F*. In *C* the mixture of ammoniacal liquor and lime, pumped in through *a*, and passing through a set of pipes, serves as cooling water, is thereby heated up, and enters the column through *b*. The proper cooler *F*, on the other hand, is supplied with ordinary cooling water, which enters through *c* into a set of pipes, and leaves this through *d*. The chambers of the column, which are separately shown from the top in Fig. 8, are so constructed that the liquid is exposed to a maximum of contact with the gas. For this purpose they are each provided with a partition *e*, forcing the liquid arriving from the upper chamber through *f* to circulate in the direction shown by the arrows, and at last, behind the partition, to run into the next lower chamber through *g*. All this time the liquid is exposed to the action of the steam escaping round the margin of cap *h*. Thus, with a very small expenditure of steam, the liquor is deprived of its ammonia up to a minute trace. The foot of

the column is formed as an iron reservoir, fitted with water gauge and pressure gauge; the outlet tap *s* is regulated in such manner that no steam can ever escape there. The steam cock is at *r*.

Other European patents of the same firm describe special arrangements for preventing the obstruction of the chambers by the lime descending in the column during all its length. Such obstructions occur only with some particular descriptions of lime, and are as a rule sufficiently removed by the numerous cleaning-holes provided. In special cases a shaft is made to traverse the column from the



top to the bottom, which is provided with scrapers for each chamber, and is turned from time to time by hand or by machinery. Another contrivance consists in an annular shovel, moving below the caps *h* to and fro, and thus preventing the accumulation of muddy obstructions.

STATISTICS.

The following statistics of coal-tar products and sulphate of ammonia are taken partly from official sources, partly from market reports as published in the leading technical journals. The German market prices ceased to be quoted in the

Journal für Gasbeleuchtung since the end of 1894, except those of sulphate of ammonia, but the English prices are quite sufficient for showing the drift of the market. The most remarkable facts to be gathered from these tables are: First, the enormous rise in the value of benzol during the year 1896, following the greatest known depression in 1895; the latter being caused by the competition of coke-gas benzol, which stimulated the endeavors for seeking new markets, and led to the recognition of the great value of benzol as an enricher for coal gas—thus causing a demand exceeding the supply, and a corresponding rise in the price of that article, which, however, cannot continue to the same extent, as it will lead to a more general adoption of the recovery of benzol from coke-oven gas. The second salient fact is the gradual decline in the value of sulphate of ammonia during the last two years, so that in October, 1896, the price stood about half of that ruling in October, 1894. The increased production of that article (as shown in the tables annexed hereto) was not in this case taken up by new uses for it. On the contrary, the bad times for the corn growers undoubtedly restricted the purchases of fertilizers, and hence the unprecedented shrinkage of value, still lasting at this time.

PRICES OF COAL TAR AND AMMONIA PRODUCTS IN ENGLAND AND GERMANY.

	1892.				1893.				1894.	
	Jan.	April.	July.	Oct.	Jan.	April.	July.	Oct.	Jan.	April.
Coal tar (London), shillings per ton	19	12½	.....	.....	12½	.....	.....	.....	.....	13–16½
Coal tar (Germany), marks per ton	20	15	.....	.....	25	.....	.....	.....	.....	20
Benzol, 90% (London), pence per gal	36	24	20½	19½	25	20	18½	18	17	14
Benzol, 90% (Germany), marks per liter	0.60	0.44	0.35	0.36	0.46	0.36	0.34	0.33	0.31	0.26
Benzol, 50% (London), pence per gal	24	18	16½	15½	21	17½	17	17	18	16
Benzol, 50% (Germany), marks per liter	0.43	0.34	0.28	0.28	0.38	0.33	0.31	0.31	0.33	0.29
Solvent naphtha (London), pence per gal	15½	15	15	14½	14½	14½	12½	15	16	14
Solvent naphtha (Germany), marks per liter	0.30	0.27	0.27	0.24	0.27	0.23	0.23	0.27	0.25	0.24
Pitch (London), shillings per ton	34	28	28	27	27	26	23½	26½	28	27
Pitch (Germany), marks per ton	34	28½	25	24	24	23	21	23½	25½	24
Anthracene, 30%, quality A (London), pence per unit per lb	13	11	10½	9½	12	13½	12	13	13	13
Anthracene, 30%, quality A (Germany), marks per unit per kg	2.00	2.02	1.72	1.48	1.96	2.21	1.97	2.13	2.13	2.13
Carbolic acid crystals (London), pence per lb	5	4¾	5	6¼	7½	8	7	5¾	6	.....
Carbolic acid crystals (Germany), marks per kg	1.00	0.87	0.92	1.15	1.38	1.50	1.28	1.05	1.10	.....
Sulphate of ammonia (Hull), shillings per ton	205	195	200	200	200	250	262½	290	267½	275
Sulphate of ammonia (Germany), marks per ton	205	220	207	216	212	260	274	308	287	278

	1894—*Cont.*		1895.				1896.			
	July.	Oct.	Jan.	April.	July.	Oct.	Jan.	April.	July.	Oct.
Coal tar (London), shillings per ton	.....	14	18–22	.....	.....	17–20	.....	18–22	.....	.....
Coal tar (Germany), marks per ton	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Benzol, 90% (London), pence per gal	11½	14½	12½	13½	12	12	27	25	33	55
Benzol, 90% (Germany), marks per liter	0.22	0.28	.....	.....	.....	.....	.....	.....	.....	.....
Benzol, 50% (London), pence per gal	14½	14	12½	13	11½	12	24	20	28½	39
Benzol, 50% (Germany), marks per liter	0.27	0.26	.....	.....	.....	.....	.....	.....	.....	.....
Solvent naphtha (London), pence per gal	14	15	.....	.....	.....	12¾	15	14	15	18
Solvent naphtha (Germany), marks per liter	0.25	0.25	.....	.....	.....	.....	.....	.....	.....	.....
Pitch (London), shillings per ton	30	38	35	32½	37	34	34	32	32	31
Pitch (Germany), marks per ton	25½	.....	.....	.....	.....	.....	.....	.....	.....	.....
Anthracene, 30%, quality A (London), pence per unit per lb	14	14	13	13	13	13	11½	10½	10¼	9¾
Anthracene, 30%, quality A (Germany), marks per unit per kg	2.29	2.29	.....	.....	.....	.....	.....	.....	.....	.....
Carbolic acid crystals (London), pence per lb	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Carbolic acid crystals (Germany), marks per kg	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Sulphate of ammonia (Hull), shillings per ton	280	260	220	220	195	175	172½	165	160	150
Sulphate of ammonia (Germany), marks per ton	290	288	228	234	220	200	187	180	170	155

PRODUCTION OF SULPHATE OF AMMONIA IN GREAT BRITAIN AND IRELAND.

(In long tons.)

Made from	1889.	1890.	1891.	1892.	1893.	1894.	1895.
Gas works	100,700	102,150	107,000	110,748	112,179	113,634	119,695
Blast furnaces	6,150	5,050	6,500	11,000	8,833	10,075	14,588
Shale works	23,950	24,750	27,000	23,105	28,485	32,891	38,335
Coke ovens	2,800	2,300	3,000	4,973	3,265	3,448	7,083
Totals	133,600	134,250	143,500	149,826	152,762	160,148	179,651

AVERAGE MARKET PRICES OF SULPHATE OF AMMONIA FROM 1869.

(F. o. b., Hull, England.)

Year.	Price.	Year.	Price.	Year.	Price.	Year.	Price.
	£ s. d.		£ s. d.		£ s. d.		£ s. d.
1869	15 15 0	1876	18 12 6	1883	16 10 0	1890	11 9 0
1870	16 0 0	1877	19 17 6	1884	14 8 0	1891	10 15 0
1871	19 0 0	1878	20 5 0	1885	11 8 0	1892	10 2 0
1872	21 0 0	1879	18 7 6	1886	11 3 0	1893	12 10 0
1873	18 2 6	1880	19 0 0	1887	11 18 0	1894	13 5 0
1874	17 2 6	1881	20 4 0	1888	11 18 0	1895	9 16 0
1875	18 10 0	1882	20 7 6	1889	12 0 6	1896	8 5 0

The Semet-Solvay Coke Ovens and their By-products.

In the Semet-Solvay ovens full consideration has been given to the importance of maintaining an equilibrium of pressure between the work-chamber and the flues. This result has been obtained by making the flues of a suitable section, and by regulating within them the direction of the current of gas.

It should be noted that the minus pressure in the chamber of the oven increases from top to bottom. The current of gas circulates from top to bottom within the flues, so that the maximum and minimum minus pressures are found at the same relative points as within the chamber of the oven. To establish this equilibrium it is only necessary to regulate the operation of the extractor and the draft of the chimney.

The Phœnix Company, at Ruhrort in Westphalia, in 1896 built a third group of Semet-Solvay ovens, which was put in operation in November. In France and Belgium there were on January, 1896, 377 Semet-Solvay ovens in operation, and during the year 86 new ovens were fired, and the construction of 235 new ovens undertaken. The last will be put in operation in 1897. The yield in by-products depends of course upon the coal treated. The Semet-Solvay ovens erected at Brymbo, North Wales, constantly obtain per ton of coal charged 112 lbs. of tar, 30 lbs. of sulphate of ammonium, and 3 gals. of light oil, density 0.89.

In the United States there are three Semet-Solvay coke-oven plants, and a new installation is being made at Birmingham, Ala. In the American practice, benzol is recovered by absorption in "dead-oil."

Benzol-winning from Coke-oven Gases.*

By Chr. Heinzerling.

Before describing the method of recovering benzol it should be mentioned that a far greater amount of the aromatic, volatile compounds is contained in the gas than in the tar. From experiments by Deville† on the division of aromatic hydrocarbons between the illuminating gas and the tar it appeared that 100 kgms. of coal developed an average of 1179 gms. of hydrocarbons. The same amount of coal gave 5.8 kgms. of tar with a maximum tenor of 1.5% aromatic hydrocarbons, or 87 gms. Consequently the gas contained 1179 less 87, or 1092 gms., of benzol. There was therefore 92 or 93% of the easily volatile hydrocarbons in the gas and 7 or 8% in the tar. We shall not go far astray if we assume that a similar division takes place in the by-product coke ovens.

Supposing that 80 Otto ovens treat about 67,000 tons of coal per annum of 300 working days we have an average of 233 tons per day. In the Saar and Ruhr district the latter deliver about 60,000 cu. meters of gas per day; in Upper Silesia, 67,000 to 68,000 cu. meters. Assuming the benzol contents of a cubic meter of gas at 20 gms. (according to the results of an experimental plant in Upper Silesia) the yield of the above-mentioned volumes of gas would be respectively 1200 and 1340 to 1360 gms. per day. For this there would be required an expenditure of 240 horse power for 24 hours. This power could be furnished by gas-motors cheaper than by steam, using the coke-oven gas. With a consumption of 0.7 cu. meter per horsepower-hour about 3600 cu. meters per day would be required. In the Sahr and Ruhr district a plant of 60 Otto ovens produced per day about 24,000 cu. meters more gas than was necessary for heating the ovens, and in comparison with this amount the 3600 cu. meters required to extract the benzol would be insignificant.

The determination of the benzol contents of coke-oven gases is accomplished by several methods.

The older Bunsen method,‡ which consists in passing the carefully dried gas through absolute alcohol, wherein the benzol and the other heavy hydrocarbons are absorbed, gives too high results.

Deville separates the aromatic hydrocarbons by condensation. The gas dried by chloride of calcium is passed through a 2-mm. worm, cooled in a mixture of salt and ice to —22° C., and then through a second and third worms cooled by methyl chloride to —70° C.§

The worms are weighed before and after the experiment, the difference giving the weight of the benzol obtained, which can be purified by fractional distillation. At —70° C. the tension of the benzol is *nil*, and all must therefore be separated. This method has been employed at a works in Upper Silesia, with the substitution of liquid carbonic anhydride for methyl chloride for the last cooling—with satisfactory results.

A simpler but less exact method consists in passing the dried gas through several wash-bottles filled with cumene and through a glass cylinder filled with beads

* Abstract of an article in *Chemische Rundschau*, March 1 and 15, 1897.

† *Journ. für Gasbeleuchtung*, 1889, p. 692.

‡ Bunsen, *Gasometr. Meth.*, 2 Aufl., p. 114.

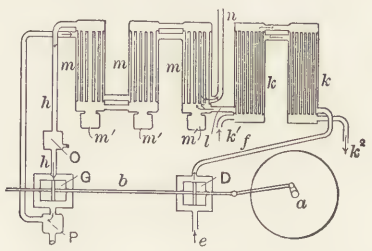
§ *Journ. für Gasbeleuchtung*, 1889, p. 652.

and cumene, the apparatus standing in a frigid mixture. After passing ½ to 1 cu. meter of gas the cumene is collected and the volatile hydrocarbons distilled off at a temperature of 140° to 145° C. The distillate is then subjected to fractional distillation and weighed.

The extraction of benzol from coke-oven gases was first attempted in Westphalia about 10 years ago. The method which was then employed, and upon which several patents were taken out in England,* depended upon the absorption of the benzol in middle or heavy tar-oils, which when saturated were subjected to distillation, and their benzol contents having been expelled were then available for use over again.

If the temperature of the absorbing medium and that of the gases are kept moderately low, and if the former is renewed at proper intervals, the separation is very good. This method has the disadvantage, however, that the oil soon becomes thick from tar and coal-dust mechanically carried over, and consequently unsuitable for further use.

The author patented in 1891 a new method† which involves the separation of benzol from coke-oven gases by condensation at low temperature. The system is illustrated by the following sketch:



(*D*) Compression cylinders. (*m'm'*) Benzol taps. (*mm*) Tubular coolers.
(*kk*) Water-coolers. (*G*) Expansion cylinder. (*O*) Benzol separator.

The coke gases coming from the washer are compressed under 3 to 3½ atmospheres, then reduced to 10° or 12° C. in water-coolers, and next in tubular coolers to —30° or —40° C., after which the gas is allowed to expand, whereby the temperature falls further to —70° or —80° C. At this point the separation of the benzol is complete, since at —70° C. the tension of benzol vapor is zero. The expanded gases flow back around the tubular cooler (thereby serving to effect the second reduction of temperature, namely, to —30° or —40° C.) to the coke-ovens. About two-thirds of the benzol separates in the tubes, partly in fluid, partly in solid state. The former collects in drainage cisterns, and the latter remains hanging in the tubes. The last part of the benzol separates, after expansion of the gases, entirely in the solid state. It is advisable to have two independent series of condensers, so that there may be no interruption of operations when it is necessary to clean the solid benzol out of the tubes. The general arrangement of the apparatus is shown in the accompanying sketch.

* Patent No. 4888 of 1887, by Fritz Friedländer and Jul. Quaglier; Patent No. 11,202 of 1884, by John Coates; Patent No. 4312 of 1882, by Josiah Hordman.

† D. R. P., No. 66,644.

This system was put in use in a large experimental plant in Upper Silesia where about 40,000 cu. meters. of gas was treated per day and about 145,000 kgms. of crude benzol were produced. The cost of production (exclusive of amortization of plant) was 13.20 marks.

The arrangement and construction of the apparatus in the experimental plant was not good, owing to the lack of experience of designers and builders, but many difficulties were overcome in the course of the experiments, and there is no doubt that with further modifications the system will prove economical. The most trouble was experienced in adjusting the relative size of the compression and expansion cylinders. This was finally arranged and the apparatus worked very well until it broke down, developing a temperature as low as —82° C., and for a long time an average of —75° to —79° C.

In order to obviate the difficulties entailed by the expansion cylinder the author proposed to abandon it entirely, and instead to conduct the gas into a chamber where the benzol might separate. Further study of the method led to the conclusion that it was not especially advantageous to recover energy from the expansion cylinder.

If for example 350 horse power is required for the treatment of 100,000 cu. meters of gas in 24 hours, about 100 horse power can be saved by the use of the expansion cylinder. At a works where the value of a ton of coal does not exceed 7 marks, assuming that 1 kgm. coal develops 1 horse power per hour, the saving in coal would amount to 18 or 20 marks per day, which would be of very small account in comparison with the value of the benzol won. If in place of steam-engines motors operated with coke-gas were used, the loss of the expansion-energy would be quite unimportant.

The gases before entering the compression cylinder must be completely dehydrated, which is easily done by sulphuric acid in a Rohrmann-Lunge plate-tower or by calcium chloride. The complete removal of the water presents the advantages of (1) non-formation of ice in the cooling tubes, (2) less compression-energy required, and (3) the possibility of winning as a by-product the not inconsiderable amount of ammonia contained in the gases. It is also to be recommended that all traces of coke-dust be removed from the gases before entering the compression cylinder by passing them through a suitable filter.

The benzol won by this method had, according to H. v. Keler and G. Lunge,* the following composition: Benzol, 67.03%; toluol, 15.61%; xylol, 2.18%; higher homologues, 6.41%; naphthalene, 3.78%; aromatic hydrocarbons of higher boiling points, 1.42%; unsaturated fatty hydrocarbons, 0.96%; thiophene, 0.7%; phenol, 0.11%; tarry bases, 0.28%; residue, 1.50%. The benzol won by absorption with heavy stone-coal-tar-oil had: Benzol, 85.1%; toluol, 11.63%; xylol, 1.4%; other constituents, 1.87%.

* *Zeitschr. der Angewandte Chemie*, 1894, p. 637.

COPPER.

THE copper mining industry in the United States in 1896 was extremely prosperous. The average price of the standard Lake copper was 10.88c. per lb. as compared with 10.76c. per lb. in 1895, and the production increased nearly 37,000 metric tons. Although the gain was mainly in the United States there was also an increase in Chile, Japan, and Australasia. Spain showed a small decrease. The indications are for a further expansion in the output. Two new producers, the Mountain Copper Company, of California, and the Mt. Lyell Mining and Railway Company, of Tasmania, appeared in the field in 1896, and their production is likely to become important in 1897, while it will not now be long before the Seven Devils Company, of Idaho, and the Washoe Company, of Montana, will enter the list.

COPPER PRODUCTION IN THE UNITED STATES.

(Pounds of fine copper.)

States.	1893.		1894.		1895.		1896.	
	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.	Pounds.	Long Tons.
Arizona	43,773,675	19,542	44,531,108	19,880	48,329,403	21,575	73,745,321	32,922
California	2,825,773	1,261	120,156	54	225,650	101	1,971,545	880
Colorado	7,121,257	3,179	6,528,214	2,914	6,125,000	2,734	9,539,245	4,259
Michigan	113,537,793	50,686	114,526,555	51,128	129,740,765	57,920	144,058,524	64,312
Montana	154,300,100	69,330	183,094,755	81,739	194,768,925	86,950	228,958,164	102,213
New Mexico	273,515	123	154,730	69	.....	.....	.....	.....
Utah	1,312,171	585	1,183,098	528	2,664,757	1,190	3,550,050	1,585
Eastern and Southern States	415,025	185	2,437,986	1,088	3,255,000	1,453	3,750,124	1,674
All others	3,441,168	1,535	927,712	414	1,344,350	601	2,050,000	915
Total domestic production	327,000,477	146,426	353,504,314	157,814	386,453,850	172,524	467,622,973	208,760
Stock January 1	55,434,000	24,743	57,418,689	26,073	78,738,689	34,698	86,961,280	38,822
Imports bars, ingots, old, and ores	13,260,077	5,920	11,726,485	5,235	13,875,560	7,194	a27,404,087	12,234
Total supply	395,694,554	177,089	422,649,488	189,122	479,068,099	213,416	581,988,340	259,816
Deduct exports	180,066,880	80,387	167,651,153	75,737	121,703,978	54,332	281,905,217	125,851
Deduct consumption	158,208,985	70,629	176,259,646	78,687	270,392,841	120,262	216,483,123	96,644
Stock December 31	57,418,689	26,073	78,738,689	34,698	86,971,280	38,822	83,600,000	37,321

(a) This includes copper imported in low-grade Spanish and other pyrites chiefly for sulphur, and the copper imported from Canada in copper-nickel matte, in which the nickel is the metal of chief value; also the copper in certain gold and silver ores. These items do not appear in the United States statistics of imports, since in these materials the copper is not the substance of chief value.

Alabama.—The old Woods, or Copper Hill mine, in Cleburne County, idle since 1879, was unwatered at considerable expense, which resulted in showing a body of pyritous ore averaging 20 ft. in thickness at the 80-ft. level, to which depth it had been worked out for a distance of about 80 ft. northerly from the

shaft. The work of sinking to a deeper level was then undertaken. At present these mines are 20 miles from the Southern Railroad Company's nearest station. When they were being worked in the seventies the ore was hauled to Carrollton in Georgia, a distance of about 30 miles, for shipment to Eastern smelters. The average grade of the ore body exposed at the Copper Hill mine is reported as being about 6% Cu.

Arizona.—The increase in the copper output of Arizona in 1896 was remarkable, being largely attributable to the Copper Queen Company, which has already extended its capacity for production a good deal beyond its present actual outturn. This company has completed the installation of its Bessemer plant.

Mr. John F. Blandy contributes the following review of the copper industry in this Territory in 1896: "In the copper industry there has been an increased production in every camp and changes made in the smelting plants show that they are up to date in the improvements in their line of work. Much work is being done in opening and equipping new copper mines in the Globe district and in the erection of smelting works. The latter will be in blast in 1897. Smelteries are also being erected near Crittenden in Pima County to work ores from the Harshaw and Washington districts. The smeltery at Tucson handled 1030 tons of ore in 1896 and produced 239,268 lbs. of copper. Copper ores are known to exist at many places in the Territory, outside of the old camps, but none has yet been sufficiently promising to induce capital to exploit them, except some near the Grand Cañon, about 60 miles north of Flagstaff, which are now being thoroughly examined by a strong company."

COPPER PRODUCTION IN ARIZONA.

(Pounds of fine copper.)

Mines.	1891.	1892.	1893.	1894.	1895.	1896.
Arizona Cop. Co..	5,673,611	5,893,533	7,871,819	9,935,812	11,308,910	13,042,000
Copper Queen...	13,022,957	12,916,416	13,795,618	12,688,372	15,741,731	23,298,150
Commercial.....	.....	282,451	273,330	90,805	.....	46,040
Detroit.....	4,194,672	1,918,594	4,942,728	5,777,744	3,790,128	7,016,348
Old Dominion....	6,982,101	7,698,297	7,665,293	4,839,386	.....	5,650,000
United Verde....	7,350,087	9,524,492	9,121,146	10,904,453	16,491,402	22,327,950
United Globe....	2,302,765	.....	.....	.....	.....	2,020,565
Other mines.....	2,368,506	149,333	103,741	214,536	997,232	344,268
Totals.....	41,894,699	38,383,116	43,773,675	44,531,108	48,329,403	73,745,321

California.—The Union and Satellite mines were idle in 1896 and the production of this State was almost entirely from the property of the Mountain Copper Company, which is the successor of the Mountain Mines, Ltd., on Iron Mountain, near Shasta. The mines, which have been opened systematically, show a large body of gold and silver-bearing sulphide ore, averaging 7% Cu. They were purchased by an English syndicate in January, 1895. The mines are connected with smelting works at Keswick by a narrow-gauge railway 10.7 miles long. The Keswick smelting works, which were originally designed for the pyritic process, were put in operation in the early part of the year, but the method not proving wholly successful, the works were altered to follow the Montana blast-furnace practice, with a preliminary stall-roasting of the ore, since which they have been run to good advantage. The original practice was

described fully by Herbert Lang in the *Engineering and Mining Journal* of July 25 and August 1, 1896. The roasting-stalls now in use are 6 ft. deep, 7 ft. wide, and 14 ft. long, holding about 35 tons of ore, which is roasted from 47% S to 10% S, with ¼ cord of wood in 4 weeks' time. The blast furnaces (3) are 42 by 150 in., with shaft 9 ft. high from tuyères to charge-door, and a 6-in. bosh. Connected with each furnace is a No. 7 positive blower, driven by water-power. Each furnace smelts about 200 tons of ore per 24 hours, averaging 25 tons of 50% matte. The ores being basic it is necessary to add silica as flux and gold-bearing quartz is purchased for this purpose.

The Mountain matte is shipped by railway to the Atlantic seaboard for refining. The present capitalization of the Mountain Copper Company is $6,250,000. The production of the State in 1896 was 1,971,545 lbs. of which the Mountain Copper Company shipped 1,821,545 lbs.

Colorado.—The copper production of this State is derived chiefly from Leadville and the mines of the San Juan region. There is so much duplication in the returns of the different smelters of Colorado, and so much of their product comes from other States, that it is impossible to arrive at the Colorado production exactly; but by care in avoiding duplications and full information courteously furnished by the producers we have found that the production of copper from Colorado ore amounted in 1896 to fully 9,539,245 lbs.

Idaho.—The Seven Devils Copper Mining Company, a strong concern, is making preparations to open the copper deposits of the Seven Devils district. It is the intention to construct eventually a railroad from some point on the Union Pacific system to the district, the initial work being the building of a smelting plant and the development of the mines. The smelting plant is now being constructed under the supervision of the Nichols Chemical Company, of New York, and will comprise Herreshoff water-jacket furnaces with some novel details of construction.

Michigan.—The copper production of Michigan mines increased a little more than 14,000,000 lbs. in 1896, against an increase of something over 15,000,000 lbs. in 1894. The statistics are given in detail in the accompanying table.

The Calumet & Hecla Company turned out nearly 13,000,000 lbs. of fine copper more than in the previous year and has a capacity to greatly increase its present output. This company makes very incomplete statements to its shareholders, and it is consequently not easy to analyze its costs. It paid dividends during the year to the amount of $2,500,000. But since we may assume that it sold its entire output at 10.88c. per lb., the average price of Lake copper during the year, we would get gross receipts of $9,830,812 (neglecting the income from rents, profits on supplies, interest, etc., and assuming the cash on hand to have remained constant). Deducting the dividends paid and counting the balance to be the cost of production, as in fact it is, whether paid for machinery or for wages, it appears that the average cost of producing Calumet & Hecla copper was 8.11c. per lb. No further sinking was done in the Red Jacket shaft, but its equipment with machinery was proceeded with.

The production of the Quincy mine in 1896 was 20,370,725 lbs. of mineral, yielding 16,863,477 lbs. of refined copper. The total amount of rock mined was 637,113 tons; amount hoisted, 577,024; amount stamped, 555,543. The

receipts from sales of copper and silver were $1,864,197.61. The expenses at the mine were $893,696.52; for smelting, transportation, etc., $214,872.62. The mining profit was consequently $755,628.47. Adding to this receipts on interest and real-estate account, the total income for the year was $770,063.64. Dividends to the amount of $1,000,000 were paid during the year, and a balance of $927,564.32 was left on hand at the end of the year. A good deal of development work was done in the mine. The depths of inclined shafts Nos. 2, 4, and 6 are now respectively 4085, 4073, and 3920 ft., measured on the incline. The amount of rock crushed by the five steam stamps of the mill was 385.25 tons per head per 24 hours of actual running time.

The production of the Kearsarge mine in 1896 was 1,570,305 lbs. of mineral, yielding 1,337,226 lbs. of refined copper. The amount of rock stamped was 34,587 tons. The percentage of copper in the stamp rock was 1.99. The cost of production per ton of rock hoisted was $1.97; the cost per lb. of refined copper was 9.76c. The total receipts of the company from sales of copper (1,377,226 lbs.), silver and interest were $162,297.38; the total expenses were $134,368.02; the profit for the year added to the balance of assets December 31, 1895, came to $191,497.09. A good deal of development work was done in the mine, and according to the report of the superintendent there was more in sight at the end of the year than ever before in its history. The smallness of the production in 1896 was due to lack of milling capacity, which has now been obviated by the starting of the new works.

The production of the Osceola mine in 1896 was 7,365,739 lbs. of mineral yielding 6,251,304 lbs. of refined coper. The percentage of refined copper in the stamp rock was 1.26. The cost of production per ton of rock hoisted was $1.86, and the total cost per lb. of refined copper was 9.64c. During the year 278,517 tons of rock were hoisted, of which 30,455 tons were discarded and the remainder dressed. The cost of dressing was 24.788c. per ton. The total receipts of the company from the sales of refined copper (6,251,304 lbs.), sales of silver, sales of land, and interest were $687,017.85; the total expenses were $602,908.74; the profits of the year added to the balance of assets December 31, 1895, came to $357,171.76, out of which dividends amounting to $100,000 were paid. A good deal of new ground was opened but much of it proved lean.

The production of the Atlantic mine in 1896 was 6,449,600 lbs. of mineral, yielding 75.88%, or 4,894,228 lbs. of refined copper, which realized 10.84c. per lb. The receipts from sales of copper, together with $493.71 interest, amounted to $531,160.85. The working expenses at the mine were $394,378.80; smelting, freight, and other expenses, $64,830.34; total, $459,209.14—showing a gross profit of $71,951.71, to which was added $3185 for stumpage credited to real estate account. The sum of $46,726.30 was expended on construction account at mine and mill, on the railroad, and in payment for timber-land; consequently the surplus of receipts over expenditures in 1896 was $28,410.41; the surplus from 1895 having been $150,874.97, the net amount on hand December 31, 1896, was $179,285.38. Out of this a dividend of $40,000 was paid in February. The quantity of rock stamped was 371,128 tons against 331,058 in the previous year, the total cost per ton of rock treated being increased about 5c., mainly by an advance of 10% in wages; the yield of rock was only 13.19 lbs. of

copper per ton against 14.6 lbs. in the previous year, consequently the cost per pound of copper was materially increased. The gross value of the product in 1896 per ton of rock treated was $1.43; the cost of mining per ton was 76.43c.; transportation to mill, 4.96c.; cost of dressing, 24.87c.; making the total working expenses at the mine only $1.0626. The cost of smelting and marketing the product, including New York office expenses, was 17.47c. per ton of rock.

The production of the Tamarack mine in 1896 was 575,960 tons of rock, of which 490,625 tons were stamped at a cost of 29.95c. per ton. The total cost of production per ton of rock stamped was $2.28; per ton of rock mined, $1.94. The yield of mineral was 23,628,899 lbs. The production of refined copper is not stated in the report of the directors, but is reported to THE MINERAL INDUSTRY as 16,495,000 lbs. The total receipts from sales of copper and from interest were $1,746,188.36. The expenses at the mine were $1,116,787; smelting, transportation, etc., $260,704.61. The mining profit was consequently $368,696.75. Dividends amounting to $360,000 were paid during the year; $256,720.63 were charged to construction account and sinking of No. 5 shaft, leaving a balance of assets December 31, 1896, of $783,304.67. About July 1 it was decided to build a new mill, which is now nearly completed. The building is constructed of steel, and is equipped with two 20-in. improved Ball stamps, with the necessary accompaniment of jigs and slime-tables. It is thought that 600 tons of rock per day can be treated in this mill. In the old mill with 5 stamps the average work in 1896 was 290 tons per stamp per day for the entire year, although the average running time per stamp was only 25 days 7 hours and 32 minutes per month.

The report of the Central Mining Company for the year ending December 31, 1896, gave the total production of copper at 469,243 lbs., an increase of 90,223 lbs. over 1895. The gross receipts were $53,223 and the expenses $68,104, showing a deficit of $14,881 for the year. The surplus at the opening of the year was $65,140; deducting this deficit there remained a balance of $50,259 on December 31, 1896. The assets on that date were: Cash and loans, $34,727; cash and supplies at mine, $7967; copper on hand, $21,366; total, $64,060. Accounts payable amounted to $13,801. The directors said that it became necessary early in the year to discontinue work in the bottom of the mine, the explorations there demonstrating that the ore body worked so many years does not extend below the fault in the formation encountered at the 30th level. The work of the past year was mainly directed toward developing the southern end of the mine. The extension of the 21st level south continued to open a fairly productive vein, and the product for the year was principally derived from this point. The 20th level was also reopened and extended into this ground. Three of the levels above are being reopened and will be extended into the same run of ground. As all of these levels (in which work was discontinued many years since) are crushed in by the pressure of the rock and decaying of the original timbering, a considerable expenditure of time and money has been necessary to reach the required point.

The Franklin Mining Company produced 3,367,218 lbs. of mineral in 1896, which yielded 81.55%, or 2,746,076 lbs. fine copper. There were 135,156 tons rock hoisted and 122,079 tons rock stamped, the yield being 1.38%, or 27.58 lbs.

mineral per ton stamped. The amount of mass and barrel work was 961,930 lbs. The receipts were for copper, $307,356; interest, $92; total, $307,448. Expenses at mine were $246,893; smelting, freight, insurance, etc., $39,897; total, $286,790, leaving a balance of $20,658. The average receipt per pound of copper was 11.19c.; mining expenses, 8.99c.; other expenses, 1.45c.; total, 10.44c. In addition the sum of $64,835 was expended in exploration in the Franklin, Jr. At the close of the year the company had a surplus of $153,487. A large amount of work was done on the new property known as the Franklin, Jr., and while some of the explorations proved worthless, others were very encouraging, particularly in the Pewabic shaft, which is now below the sixth level. A lot of 600 tons of rock worked showed 1.7% mineral, or 1.4% fine copper. According to the superintendent's report "if the future openings of the Pewabic lode are as good as they have been during the last few months, there will be a large and profitable mine at this place." It is now expected that regular shipments of rock will be made to the Franklin mill early in the summer of 1897.

COPPER PRODUCTION IN MICHIGAN.

(Pounds of fine copper.)

Mines.	1891.	1892.	1893.	1894.	1895.	1896.
Allouez	1,241,423	546,530	.....	.....	.....	.....
Atlantic	3,653,671	3,703,875	4,221,933	4,437,609	4,832,497	4,894,228
Calumet & Hecla	63,294,634	58,292,576	62,825,674	61,842,722	77,439,907	90,356,733
Centennial	435,784	.....	.....	.....	.....	.....
Central	1,321,417	1,625,982	1,177,500	584,950	370,381	469,243
Copper Falls	1,440,000	1,400,000	1,000,000	.....	.....	.....
Franklin	4,319,840	3,767,000	3,504,244	3,602,608	3,086,933	2,746,076
Huron	1,215,734	461,499	562,776	.....	.....	.....
Kearsarge	1,727,390	1,580,192	1,546,318	1,998,710	1,946,163	1,337,226
Mass	30,114	17,450	22,737	41,805	18,372	38,684
National	103,888	36,358	63,433	36,390	50,128	53,762
Osceola	6,543,358	6,894,256	6,216,975	6,879,000	6,270,373	6,251,304
Peninsular	1,599,670	973,217	.....	.....	.....	.....
Quincy	10,542,519	11,015,686	14,398,477	15,484,014	16,304,721	16,863,477
Ridge	43,049	41,462	25,988	52,481	64,363	.....
Tamarack	16,161,312	16,426,643	15,085,113	15,375,000	14,840,000	16,495,000
Tamarack Junior	.....	796,769	1,610,259	2,350,000	2,605,000	2,330,000
Wolverine	312,112	500,074	1,025,062	1,665,255	1,817,806	2,195,791
All other mines	121,509	11,750	251,304	176,011	94,121	27,000
Total pounds	114,107,424	108,091,336	113,537,793	114,526,555	129,740,765	144,058,524

Montana.—The increase in the copper production of Montana in 1896 was enormous, chiefly due to the Anaconda Company. This huge concern, which is now largely owned in Europe, made its first public report in 1896, covering the year ending June 30. According to this its shipments from Anaconda during the year were 107,036,697 lbs. of copper in the form of anodes, electrolytic and converter bars. Its sales during the same period were 85,476,795 lbs. of fine copper, 4,498,560 ozs. of fine silver, and 14,384 ozs. of fine gold. The unsold product was valued at $4,888,020.26, which actually realized a larger amount. The total expenses at the mines were $5,071,677.61, which included $355,434.54 in equipping and developing unproducing mines, and $360,981 for new machinery, etc., for producing mines. The expenses at the reduction works were $5,616,516.78, which included $519,000 laid out on plant. The profits from subsidiary departments were $829,485.89, which amount was deducted from the operating expenses at mines and works. The profits for

the year were in round figures $4,000,000. The amount of ore received at the reduction works was 1,276,156 tons, all except 20,282 tons coming from the Anaconda mines. The shipments from the works were copper 107,036,697 lbs., silver 5,308,955 ozs., gold 18,300 ozs. Many improvements were made in the reduction works during the year. The converter plant was enlarged, and the capacity of the refining works was doubled, so that now it can turn out 6,000,000 lbs. of fine copper per month. Some attention has been devoted to improvement in the method of roasting the ore. The mines continue to show large reserves of ore. The St. Lawrence shaft is now 1200 ft. deep, the Bell, 1300 ft., and the Mountain Consolidated, 1100 ft.

The Boston & Montana Company made somewhat more copper in 1896 than in 1895. During a portion of the year it operated the Butte & Boston smeltery at Butte, leaving an equivalent part of the Great Falls works idle, for reasons which are not clear. The Butte & Boston Company was reorganized in 1896.

The Montana Ore Purchasing Company increased a good deal in importance as a copper producer in 1896. It made large extensions to its works, and others yet are contemplated. Among various improvements an installation of 36 Herreshoff improved MacDougall roasting-furnaces was made, which are giving excellent results as to economy in labor and in maintenance.

The Montana Ore Purchasing Company, Butte Reduction Works, and some of the other smelters of Butte receive large quantities of ore by purchase from the small mines, the production of which is now between 20,000,000 and 25,000,000 lbs. of fine copper per annum.

There is prospect that the Washoe Company of Butte will soon erect works. This concern, in which Messrs. Haggin and Daly are interested, owns several mines in Butte, which were developed under lease by the Anaconda Company and are already important producers.

COPPER PRODUCTION IN MONTANA.

(Pounds of fine copper.)

Mines.	1891.	1892.	1893.	1894.	1895.	1896.
Anaconda	46,500,000	100,000,000	75,256,657	95,578,000	99,775,294	125,350,693
Boston & Montana	26,507,929	30,386,595	31,800,000	57,937,633	60,746,000	60,250,000
Butte & Boston	18,392,054	10,641,269	20,457,928			4,500,000
Butte Reduction Works	2,915,000	2,864,000	2,985,485	2,282,000	3,390,000	4,225,647
Colorado Smelting and Mining Co	3,641,384	4,560,972	6,703,488	5,158,730	7,750,000	9,090,680
Hecla Consolidated Mining Co	91,090	159,859	77,565	362,897	239,631	312,445
Parrot	14,108,382	12,438,782	7,791,167	7,469,908	7,257,000	8,045,648
Montana Ore Purchasing Co			8,400,000	12,293,686	14,861,000	15,049,066
Others	204,000		827,810	2,011,901	750,000	4,242,583
Totals	112,359,839	161,051,477	154,300,100	183,094,755	194,768,925	a228,958,164

(a) In addition to 818,531 lbs of Canadian copper smelted by the Montana Ore Purchasing Co. and deducted in the above table, there has been also deducted 1,000,000 lbs. more of foreign copper estimated as having been included in the aggregate of the above returns, leaving the net amount 228,958,164 lbs.

Tennessee.—The Ducktown Copper, Sulphur and Iron Company carried on work in the same systematic manner and produced nearly the whole Tennessee output. Although the ore yields only an average of 3.5% copper, yet the mining and smelting are done at a fair profit. The company keeps about 300 men employed.

Vermont.—The Vermont mines were idle during the year.

THE WORLD'S COPPER PRODUCTION, 1893–96. (a)

Countries.	1893.		1894.		1895.		1896.	
	Tons of 2240 Lbs.	Metric Tons.	Tons of 2240 Lbs.	Metric Tons.	Tons of 2240 Lbs.	Metric Tons.	Tons of 2240 Lbs.	Metric Tons.
Australasia	7,500	7,621	9,000	9,144	10,000	10,160	11,000	11,176
Austria-Hungary	1,400	1,423	2,120	2,154	1,310	1,331	1,285	1,306
Bolivia	2,500	2,540	2,300	2,337	2,250	2,296	2,000	2,032
Canada	3,630	3,679	3,786	3,847	3,924	3,987	4,190	4,225
Cape of Good Hope:								
Cape Company	5,225	5,309	5,150	5,233	5,350	5,436	5,470	5,558
Namaqua	890	904	1,500	1,524	1,730	1,758	1,980	2,012
Chile	21,350	21,694	21,340	21,681	22,075	22,428	23,500	23,876
Germany:								
Mansfeld	13,454	13,669	14,963	15,202	18,860	18,098	18,265	18,557
Other mines	2,951	3,000	2,210	2,260	1,695	1,722	1,800	1,829
Italy	2,460	2,500	2,500	2,540	2,500	2,540	3,400	3,454
Japan	18,000	18,288	20,050	20,371	18,430	18,725	21,000	21,336
Mexico:								
Boleo	7,979	8,107	10,371	10,537	10,600	10,769	9,940	10,099
Other mines	1,480	1,500	1,400	1,422	1,170	1,189	4,032	4,098
Newfoundland	2,000	2,032	1,900	1,930	1,800	1,829	1,800	1,829
Norway	1,740	1,768	1,785	1,813	2,685	2,728	2,500	2,540
Russia	4,920	5,000	5,000	5,080	5,000	5,080	5,000	5,080
Spain and Portugal:								
Rio Tinto	31,000	31,499	32,689	33,215	32,985	33,513	34,501	35,055
Tharsis	11,000	11,177	11,000	11,177	12,380	12,638	11,040	11,217
Mason & Barry	4,400	4,470	4,200	4,267	4,100	4,166	3,381	3,435
Sevilla	1,010	1,026	1,170	1,188	1,050	1,069	1,025	1,041
Other mines	6,500	6,605	4,805	4,882	4,300	4,369	3,400	3,455
Sweden	544	552	500	508	515	523	500	508
United Kingdom	426	433	400	406	400	406	500	508
United States	149,874	152,271	161,510	164,194	172,524	175,294	208,760	212,112
Venezuela	2,861	2,907	2,500	2,540	.....	.....	*Nil.*	*Nil.*
All other countries	620	630	670	680	635	645	840	853
Totals	306,714	310,704	324,859	330,132	334,268	339,699	381,109	387,207

(a) The figures in this table are taken from the annual metal circular of Henry R. Merton & Co., except where returns have been received by THE MINERAL INDUSTRY direct from producers.

Australia.—The largest copper producer of Australia is the Wallaroo & Moonta Mining Company, which works a low-grade ore. In 1895 the production of the mines was about 200,000 tons, which was dressed to 28,579 tons, nearly one-half assaying 20% Cu and the remainder 14%. The cost of mining was $153 per ton of copper produced; smelting, $52.50; and total, $212.88.

Canada.—The production of copper in Canada in 1896, according to the report of the geological survey, was 9,385,556 lbs. against 7,987,000 lbs. in 1895; of the production in 1896, British Columbia contributed 3,818,556 lbs. against 952,840 lbs. in 1895. The remainder of the Canadian copper production is derived chiefly from the Sudbury copper-nickel mines, concerning which reference should be made to the article on "Nickel and Cobalt" elsewhere in this book.

The British Columbia Smelting and Refining Company, which began the erection of works at Trail Creek in October, 1895, blew in the first furnace in February, 1896, and before the end of the year had five in operation. The plant consists of a 150-ton sampling mill, a 9 by 90-ft. double two-hearth O'Hara roasting furnace, and 6 Herreshoff calciners, four 14 by 22-ft. reverberatory smelting furnaces, one 40-in. circular water-jacket furnace, and one 120 by 38 in. water-jacket furnace, with movable crucible and forehearth. By the middle of the year the works were smelting from 140 to 160 tons of ore per day, effecting a concentration of 20 : 1. The matte is shipped to Butte for further treatment.

The chief copper producer of the Nelson district is the Silver King mine of the Hall Mines Company, Limited, which is located high up on Toad Mountain, 8 miles by a steep wagon road above Nelson. The ore is bornite with some tetrahedrite, copper and iron pyrites, and small quantities of galena and blende. Owing to the inaccessibility of the property, an aerial tramway of 23,205 ft. length had to be constructed, connecting with the smeltery erected in the outskirts of Nelson. These works were blown in on January 14, 1896, and up to the end of the year had been in operation 255½ days, during which about 30,000 tons of ore were treated. The average yield was 21 ozs. silver and 3.7% copper. This return is largely from low-grade ore, other shipments having averaged 190 ozs. silver and 18.17% of copper.

Chile.—Alberto Herrmann, in the *Boletin de la Sociedad Nacional de Mineria*, Series 2, No. 98, December 31, 1896, reviews the condition of the mining industry in Chile, from which the following notes relating to copper in that country are abstracted.

The Chilean copper mining industry has suffered heavily from the decline in the value of copper, the exhaustion of the principal old deposits, and the general belief that these deposits become poor or play out with depth. Thus it is said that the principal veins of Tamaya have been worked out; also the great vein of Carrizal Alto, and the mines of Higuera; the vein of Las Animas in Chañaral is poor, and the Paposo mines are unproductive. These districts are situated near the coast, where primitive crystalline rocks predominate, in what has been considered the Chilean copper zone *par excellence*. They are near the coast, wherefore they have enjoyed moderate freights, and their smelteries have been supplied with coal much cheaper than the districts of the interior.

However, it is not certain that all the principal veins of the coast have become poor in depth, and the unfavorable opinion of their downward continuance is erroneous, as there is proof in Carrizal Alto. This famous district contains two principal veins: the eastern, known as Los Lachos, is almost vertical; its mean width is less than a meter; it has been worked in numerous claims (*pertenencias*), its longitudinal extent being great. The house of Gonzalez, Izaga & Co. has worked the Armonia mine on this vein to a vertical depth of more than 400 meters, winning from it in the year April 1, 1895, to March 31, 1896, 37,990 metric quintals of ore averaging 11.454% Cu, a tenor which has been more or less uniform from the surface down. The principal vein of Carrizal Alto, that of La Mondaca or Portezuelo, has a dip of 45°, and in its upper workings attained a width of 15 to 20 meters. Its levels are now generally in poor iron pyrites; the deepest opening is in the Santa Margarita mine, which cuts the vein with a vertical shaft at a depth of 268 meters, and then following the vein on its slope for 30 meters more shows a thickness of 1.5 meters of very ferruginous ore.

The same house, Gonzalez, Izaga & Co., has worked for many years in the Astillas district, also near the coast. Its vein is very narrow, but it has nevertheless been followed to a vertical depth of 300 meters. The grade of the ore near the surface did not exceed 2% Cu, but it has been mined on account of its excellent fluxing qualities with other ores. As depth has been gained, however, the grade has improved somewhat, so that in the two years from

April 1, 1894, to March 31, 1896, the total production, 93,000 metric quintals, showed an average of 4.663% Cu. In the lower levels this vein presented the rare phenomenon of dividing into three parts, each narrow, but with a grade of ore averaging 10 to 11%, of which class the mine is now producing more than 4000 metric quintals monthly.

Another proof that the opinions of the general impoverishment of the mines in depth is erroneous is presented in the Cerro Blanco district, where Gonzalez, Izaga & Co. have resuscitated the mines which had become dead in the hands of the house of Gibbs from which Señor Izaga purchased them. The Cerro Blanco district does not have the crystalline rocks of the coast, its formation being the andesites and trachytes of the Cordilleras. The principal mine reopened there is the famous Agua Amarilla, which has a depth of 500 meters, and in its exploitation during the two years April 1, 1894, to March 31, 1896, produced 68,361 metric quintals, of an average grade of 14.575% Cu. Cerro Blanco is a vast mineral district. Between the two principal veins, Agua Amarilla and Coquimbana, there are twelve parallel veins, and in the quebrada of Agua Amarilla toward the plain are many others.

The house of Gonzalez, Izaga & Co. is working, moreover, in another district of much importance, namely, Los Bronces in La Jarilla. These mines exceed 400 meters in depth and show rich reserves. In the two years from April 1, 1894, to March 31, 1896, they produced 88,462 metric quintals of an average grade of 20.207% Cu.

Don Gregorio Donoso V. has re-established the Volcan district in the cajon of Maipo by exploitation of the vein of Los Bronces, whose ores are of low grade, rarely reaching 8% copper. The mines are worked systematically, however, and the ores are smelted in blast furnaces at the foot of the mountain, saving the cost of transportation; a new converter plant is being installed.

The Tamaya district is no longer very productive, work having been abandoned below the Lecaros adit since the Señores Errázuriz have lost faith in the mines which produced so many millions for their family. In the Tamaya district, however, there are also other veins, like those of Llanos Blancos and that of Los Patos, which are now being worked by Enrique Concha T.

The Higuera district, whose ores have generally been of low grade, has not been explored to any considerable depth, the mining of these low-grade ores being affected much by the low price of copper.

The Province of Coquimbo also possesses many copper mines, some of which are now being worked, and others are awaiting lower freight rates, which depends partly upon the prolongation of the line from Ovalle to San Marcos, and partly upon the reconstruction of the line from La Serena to Rivadavia. These lines will open the copper, copper-silver, silver, and silver-gold districts of the Cordillera, which here, as in all the eastern part of Chile, are at present intact, though their number is important.

Without going at length into the copper resources of the north of Chile, it may be said that in Atacama there is a mine, La Dulcinea, belonging to an English company which has been exploited to a vertical depth of almost 600 meters, showing ore of an average grade of 18% Cu. In the bastions of the pass of Paposo there are the great deposits of the Reventon and La Abun-

dancia, from which Don Jose Antonio Moreno took great quantities of rich ore, and though recent exploitations have been unprofitable they doubtless still have a future.

Thus the opinion that the copper deposits in the crystalline rocks lose their richness with depth is evidently false, and in the region of the Cordilleras are rich deposits of copper now almost untouched. If the mines could have cheap explosives and were exempt from customs imports on machinery and firebrick; if they had the advantage of improved metallurgical methods, and the construction of railways to give them cheap freights, without doubt the copper mining industry of Chile would take a fresh start.

At present the industry suffers from lack of capital, high price of coal, scarcity or absolute lack of water, exceedingly high cost of explosives, necessity of selling the ores to the nearest smelter, who converts them to matte and on account of the excessive price of coal and coke cannot pay high prices for them; moreover the matte smelter is in most cases obliged to sell his product to the few great establishments which convert matte into ingots and pay the matte smelter but little more than the price which he could obtain by exporting the matte.

The exportation of copper ingots from Chile in 1895, according to the *Estadística Comercial,* was as follows: From Coquimbo by the Guayacan smeltery (including the Lambert smeltery), 11,849.6 metric tons; from Coronel by the Lota smeltery, 5441.8; from Valparaiso by the smelteries of Catemu, Maitenes and E. Concha, 1497; from Caldera by the Compañia Industrial de Atacama, 1225; from Antofagasta by the Bellavista smeltery, 28.5—total 20,041.9 metric tons. It should be mentioned that a part of the copper exported from Valparaiso, and all that from Antofagasta, is argentiferous and auriferous. The amount of copper matte exported was as follows: From Caldera, 231.67 tons; from Antofagasta, 154; from Valparaiso, 23.7; from Coquimbo, 7.36—total 416.73 tons. Although these mattes are designated as copper mattes, it is probable that they are argentiferous and auriferous. The exportation of argentiferous copper matte from Valparaiso was 258.07 tons; from Antofagasta, 389.13; from Caldera, 16.62—total 663.82 tons. The exportation of copper ores from Caldera was 5153.69 tons; from Taltal, 996.90; from Antofagasta, 728.04; from Valparaiso, 79.15; from Carrizal Bajo, 5.40—total, 6963.18 tons. Of silver-copper ore there were exported 86.43 tons from Valparaiso and Caldera. The copper contents of the above five classes amounted to 21,797.5 tons. There are no exact statistics to show how much ore was smelted to produce this amount of copper, but it will not be far out of the way to assume that the average yield of the ore was 11% Cu, wherefore after deducting the 7049.6 tons shipped crude it may be calculated that 191,109.5 tons were reduced in the country.

In smelting these ores reverberatory furnaces are employed for the more part. Blast furnaces are used only, or practically only, in Panulcillo, La Ligua, and Cabildo, in the Volcan district, and in the Cajon de Maipo, in the Maitenes smeltery in the Cajon del Mapocho, and to some extent in Catemu, in the establishment of the Compañia Industrial de Atacama. Of the total production of matte about 5000 tons are from blast furnaces, and

15,500 from reverberatories. The Chilean copper ores are generally basic, and Señor Herrmann is of the opinion that all the reverberatory furnaces might well be supplanted by blast furnaces.

In the Charñarcitos smeltery of Gonzalez, Izaga & Co., using reverberatory furnaces, in the two years April 1, 1894, to March 31, 1896, the amount of ore smelted was 53,625.5 metric tons, averaging 12.361% Cu, for which the consumption of coal was 21,401.4 tons, or ore to coal in the ratio of 2.5056 : 1. The total cost of the coal was $503,712.06, or $23.53 per metric ton. No deduction is made here for profit on coal sold, which amounted to $34,658.03. The cost of coke in Chañarcitos would be $29.74 (it is worth $26 of 17½d. on board ship at Carrizal Bajo). The consumption of firebrick at the Charñarcitos works in the two years was 140,942.

The smelters who produce only matte sell it to the establishments which convert it into ingot, chiefly to those of Guayacan and Lota, whose combined output in 1895 amounted to 17,291.40 tons, or about 80% of the total for the country. These works are large buyers of ores as well as of matte. In purchasing 50% matte, delivered, they pay the market price for its copper contents less $3.75 per Spanish quintal. Some large concerns are able to make more favorable contracts.

In converting matte into ingots two methods are used in Chile: reverberatory smelting at Guayacan, and bessemerizing which was first introduced at Lota, and then at the small works of Maitenes in the Cajon del Mapocho. Later Don Enrique Concha put in a plant in his smeltery, and recently Don Gregorio Donoso made a similar installation in the Volcan smeltery. Of these Bessemer plants the first only uses steam, the others having water-power.

The method employed at Guayacan, which is also in use at the Totoralillo works, consists in grinding the matte very fine, roasting in reverberatory furnaces, and smelting it with a mixture of silicious, oxidized ore and rich slag to metal of 96% Cu and upward, a certain amount of regulus with 80% Cu being formed at the same time. The latter is reduced separately in reverberatory furnaces, the resulting metal being very pure.

Señor Herrmann estimates the cost of production at these works as follows: Crushing 12.3 tons of matte per diem at $1 per ton, $12.30; roasting at $4.50 per ton, $55.35; smelting, together with 4.6 tons of 16% ore and 2.6 tons of 8% slag, $79.60—total, $147.25. The amount of copper produced is 7.058 tons, 36 kgms. being estimated as lost in slag. The copper contained in the 4.6 tons of 16% ore is valued at $400, which with freight will have cost $320. The estimate of the cost of production does not include the resmelting of the 80% regulus. Coal is estimated as costing $10 of 18d. per ton.*

The works at Guayacan and Lota do not work under precisely equal conditions, but the freights that the former works have to pay on coal and part of the matte purchased on the coast are offset by the freights of the latter on all ores and mattes to the coast. The cost of bessemerizing, based on experience at the Maitenes works, which have water-power, is estimated at $9 per ton of copper produced.

Germany.—The consumption of copper in Germany last year showed a large

* $1.00 United States = $2.74 Chilean, when exchange is 18d.

increase. According to Messrs. Aaron Hirsch & Son the imports were 73,123 metric tons, of which 59,993 tons were in metallic form, 5000 tons in copper ores from Austria, and 8130 tons in pyrites from Spain and Portugal. The exports in all forms amounted to 12,452 tons. The statement of copper supply for four years past is given as follows:

	1893.	1894.	1895.	1896.
Total imports in all forms	51,806	52,504	59,742	73,123
Production of German smelteries	24,011	25,857	26,013	29,700
Totals	75,817	78,361	85,755	102,823
Less exports	11,304	10,406	10,893	12,452
Less metal counted twice	4,000	5,000	4,500	5,000
Total deductions	15,304	15,406	15,393	17,452
Total supply	60,513	62,953	70,362	85,371

The increase last year, as compared with 1895, in the total supply was 15,009 tons, or 16.3%. The Mansfeld mines are the chief producers in Germany.

Portugal.—The report of the Mason and Barry Company for 1896 stated that the profit, including the balance, £3579, brought forward, amounted to £40,720, less sundries, £547, and the directors having written the sum of £4500 off La Sabina shares, there remained a balance for appropriation of £35,673. From this a dividend of 3s. 6d. per share was paid and £3268 was carried forward. The company's claim against the assets of La Société Industrielle et Commerciale des Metaux was adjudicated upon by the French court, and, in October, the liquidator paid to the company the sum of £17,111.

Spain.—The copper mining industry in the Province of Huelva was prosperous in 1896. The Rio Tinto Company produced 1,440,000 tons of pyrites (60,000 more than in 1895), the Sotiel Coronada, 74,000 tons, Cabezas del Pasto, 5273, and the Herrerias, 9672. The total production of the province was 2,800,000 tons. The Aznalcollar mines in the Province of Seville produced 25,000 tons of pyrites, while the Profunda mine of Villamanin (León) put out 1050 tons. The Carracedo mines (Palencia) have been sinking a shaft of 700 meters. The production of copper precipitate with 75% Cu was 35,000 tons, and that of matte, with 45% Cu, 23,000 tons.*

The report of the Rio Tinto Company for the year ending December 31, 1896, shows that the profit on sales of produce amounted to £828,103; rents, railroads, etc., £21,091; adjustment of exchange account, £94,386; total, £943,580. To this is to be added £28,677 brought forward from previous year, making a total available of £972,257. The payments included interest on 4% debentures, £143,466; taxes of all kinds, £39,069; general expenses in Spain, £47,023; general expenses, etc., in London, £21,021; plant and outlay written off, £28,677; debentures drawn and redeemed, £53,980; total, £343,225, leaving a net balance for the year of £639,032. From this a total of 38s. per share, or 19% on the par value, was paid in dividends, requiring £617,500, and leaving a balance of £21,532 to be carried forward to the current year.

On the North Lode the quantity of overburden removed during the year was

*Roman Oriol, *Revista Min. Met. y Ingenieria*, Feb. 8, 1897.

786,418 cu. meters, the outlay on the work being charged to revenue. The overburden account now stands at £73,254. On the South Lode it has been decided to extend the open-cast in order to extract the valuable decomposed mineral which it would not be prudent longer to work by underground galleries. The extension and development account has been reduced from £137,465 to £121,075, and as the charge which is made on every ton of mineral raised to liquidate both this and the overburden account is proving more than sufficient, it does not seem necessary to set aside special amounts to this end. The following table shows the production from the mines for six years:

Year.	Pyrites Extracted.				Pyrites Consumed.		Copper Produced at Mines. Tons. (*a*)
	For Shipment. Tons. (*a*)	For Local Treatment. Tons. (*a*)	Total Tons. (*a*)	Average Copper Contents. Per Cent.	Tons. (*a*)	Average Copper Contents. Per Cent.	
1891.....	464,027	972,060	1,436,087	2.649	432,532	2.651 1.309	21,227
1892.....	406,912	995,151	1,402,063	2.819	435,758	2.569 1.465	20,017
1893.....	477,656	854,346	1,332,002	2.996	469,339	2.659 1.544	20,887
1894.....	498,540	888,555	1,387,095	3.027	485,441	2.594 0.988	20,606
1895.....	525,195	847,181	1,372,376	2.821	518,560	2.595 0.986	20,762
1896.....	591,752	845,580	1,437,332	2.931	549,586	2.529 1.068	20,817

(*a*) Tons of 2240 lbs.

The quantity of pyrites invoiced in 1896 to consumers in England, Germany, and the United States was 549,585 tons, against 518,560 tons in 1895, 485,441 tons in 1894, and 469,339 tons in 1893. The copper produced during the year by treatment at the mines was 20,817 tons, and the copper in pyrites shipped amounted to 13,684 tons, making a total of 34,501 tons. The following quantities were brought to market: Refined copper, 17,777 tons; copper in sulphate of copper, 980 tons; copper in pyrites, 11,928 tons; total, 30,685 tons.

	1892.	1893.	1894.	1895.	1896.
Capital and Liabilities:	£	£	£	£	£
Share capital	3,250,000	3,250,000	3,250,000	3,250,000	3,260,000
Bonded debt	3,627,000	3,534,360	3,437,060	3,600,000	3,546,020
Creditors' bills, etc	282,887	343,239	252,842	212,655	247,451
Revenue:					
Balance from previous year	66,161	73,881	71,174	25,087	28,677
Profit on sale of produce	496,849	528,295	459,405	626,287	828,103
Other credits	102,823	105,082	97,692	77,307	115,477
Totals	665,833	707,258	628,272	728,681	972,257
Charges:					
Bonds redeemed	75,900	92,640	97,300	50,500	53,980
Interest, discount, etc	184,397	180,452	179,429	159,613	143,465
Income, etc., taxes	27,189	31,490	29,147	22,041	39,069
Administration, London	18,596	19,208	18,864	18,771	18,236
Administration, Spain	51,558	52,268	50,521	47 048	47,623
Written off	43,494	46,072	44,595	19,531	23,667
Available for dividend, etc	301,381	298,674	230,087	411,177	639,032
Reserve	.....	.....	75,000	100,000	100,000
Dividend per share	7%	7%	4%	11%	19%
Amount of dividend	227,500	227,500	130,000	357,500	617,500
Balance carried forward	73,881	71,174	25,087	28,677	21,532
Ores and produce on hand at end of year	862,161	882,156	852,439	728,782	736,925
Overburden, extension, and development accounts	279,342	282,436	250,700	220,665	194,330
Range of prices of shares	(*a*) (*b*) 19½–14⅜	(*a*) (*b*) 16 1/16–12 1/16	(*a*) (*b*) 16½–12¾	(*a*) (*b*) 20⅛–12½	(*a*) (*b*) 26–14 9/16

(*a*) Highest. (*b*) Lowest.

The stocks of copper at the company's works at Cwmavon, consisting of refined copper, copper in process of treatment, precipitate, matte and sulphate of copper (in all 5129 tons) were carried over at cost price. An improved demand at remunerative prices induced the directors to resume the manufacture of sulphate of copper, and the production at the mines and Cwmavon in 1896 was 5427 tons, of which 3917 tons were sold.

The reserve heaps contain 109,651 tons of fine copper, which stand in the company's books at £4 6s. 10d. per ton, and this figure being considerably less than their value, it is not proposed to write them down further. The rainfall during 1896 registered only a little over 16 in., but has, since January 1, 1897, been so abundant that the company's reservoirs at the mines are now full. As it was found that by concentrating the company's work in one place considerable economy could be effected, the Grange Works have been closed and the lease disposed of. All refining will now be done at Cwmavon.

The report of the Tharsis Sulphur and Copper Company for the year ending December 31, 1896, showed that the net profit from the mines during the year was £277,470, and interest received $5539, making a total of £283,009. Charges were, for taxes and fees, $11,546; management, legal expenses, etc., £13,799; depreciation accounts, £38,395; reserve insurance fund, £500; total, £64,240, leaving a balance of £218,769. To this is to be added £15,689 brought over from the preceding year, making a total of £234,458. Since the close of the year a dividend of 7s. per share, or 17½% on the stock, has been paid, requiring £218,750, and leaving a balance of £15,708 to the current year.

The total quantity of mineral raised was 557,577 tons, a decrease of 54,906 tons from 1895. Shipments of pyrites amounted to 239,725 tons, an increase of 12,896 tons. The quantity of copper precipitate shipped was 7538 tons, a decrease of 1905 tons from 1895. This was due to the extremely dry year and short supply of water, which limited the quantity of ore treated. In consequence the production of refined copper at the company's Hebbum works showed a decrease of 1875 tons.

At the Lagunazo mine there were 94,423 tons of mineral taken out, against 35,618 tons in 1895. There were 86,333 cu. meters of overburden removed. At the Calanas mines 209,950 tons of mineral were extracted, a decrease of 88,474 tons. There were also 48,357 tons of stuff removed to facilitate working; much of this carries copper enough to pay for working on the spot. The overburden stripped was 427,681 cu. meters. At the Tharsis mines 253,204 tons of mineral were extracted, a decrease of 25,237 tons. The Esperanza deposit gave 154,696 tons of schist, a decrease of 71,553 tons.

Tasmania.—The great Mt. Lyell mine became a producer in 1896. Smelting was begun with one furnace in July; in October a second was blown in. Up to the end of October, with 3¼ months of operation, the amount of ore smelted was 9741 tons, producing 992 tons of matte, which contained 481 tons of copper, 33,772 ozs. silver, and 1795 ozs. gold, an average of 4.94% Cu, 3.47 ozs. Ag, and 0.18 oz. Au. The profit in October was $10.80 per ton. In the first 8 months of operation the amount of ore smelted was 33,610 tons, yielding about 3000 tons of matte with 1540 tons Cu, 97,839 ozs. Ag, and 6159 ozs. Au.

The New York Copper Market in 1896.

The business community were not sorry when the year 1896 finally came to an end. It could hardly be foreseen that the year would bring with it so many vicissitudes and such vast uncertainty, which again brought us on the verge of a panic almost as serious as the one of 1893, the consequences of which are still more or less felt.

During all the turmoil of the year copper stood the test better than could have been expected, and this is to be attributed to the great prosperity to be recorded of the industries of Europe. The reports received from there throughout the year can practically be compared with the most prosperous times which the United States experienced during the seventies and eighties, and while we were hampered here right and left, pending the political and financial issues, Europe could hardly find hands and machinery and workshops enough to execute the orders on the books. Never in the history of the world have European shipbuilders been so busy as during the past year; never before was there observable such an expansion in trade in the direction of the transmission of electric power, and never before since the introduction of the electric light, telephone, etc., was the demand for copper wire larger than during that period. This was the more fortunate for the copper industry, as not only did consumption in the United States during the year show a continual falling off, which, toward the fall, resembled almost an utter stagnation, but on the other hand there was such an enormous increase in the production from the beginning of the year that the total output of copper in the United States amounts to more than 15% above the very large total of 1895. Were it not, therefore, for the unprecedented demand for copper from Europe, it is difficult to realize how far the otherwise unavoidable depression in prices would have gone. As it was, copper not only showed an astonishing stability, but almost throughout the year ruled at such high figures as brought the average up to about that of 1892 and 1893, and far above that of 1894. For the last four months shipments were so large, and, in consequence of a great movement in grain, steamer room so scarce that exporters had the greatest difficulty in securing freight room, although freights had gone up from 100 to 200%, and in several instances even more. The total exports for the year 1896 amounted to 125,605 tons of 2240 lbs. against 64,000 tons in 1895, or an increase of 61,605 tons.

In the meantime home manufacturers who were very cautious kept their stocks as low as possible, and most of them were able to work only half time or not much more, and though they may have to chronicle a rather bad business for themselves, they were certainly not exposed to such fluctuations in the cost of raw material as to inflict heavy losses on them in this direction.

Consumption of sulphate of copper again showed a very large increase, and though production has, at least to a certain extent, been enlarged, there was such a very large demand for shipment over the first six months of 1897 that the stocks which usually accumulate before winter are sure to be exhausted before February. Consumption in this direction is becoming more important from year to year. The exports in 1896 reached a total of 7220 long tons.

The larger production already mentioned was about equally divided among

the main producers. The Anaconda continues to stand at the head of the list, and will average above 10,000,000 lbs. of fine copper monthly. On Lake Superior it is the Calumet & Hecla which showed the material increase, and this company may be set down as producing 7,000,000 to 8,000,000 lbs. of fine copper monthly.

It is not expected that the Old World will show any material increase in production, but it must not be assumed that this could not easily be done if desired; and some of the larger producers are in excellent position to materially increase their output at comparatively short notice.

As already pointed out, prices for copper ruled fairly high, and the fluctuations may be figured at 1½ to 1¾c. at the outside. Late in December, 1895, the Calumet Company made a large sale to manufacturers, extending over several months, at 10c., but early in January, under the general depression contingent upon the President's Venezuela message, this price was shaded, and by the end of that month prices had sagged off to 9¾c., while a large business had been doing in electrolytic copper at 9½c. These low prices brought large orders, not only from home consumers, but principally from abroad, and in February prices were marked up to 10½c. for Lake, and close to this figure was obtainable for the prime brands of electrolytic copper. Orders from consumers were coming in to such a large extent and the demand, especially for wirebars, was so large that prices quickly advanced to 11c., at which figure the Calumet & Hecla Company sold the bulk of its output for several months ahead, and later on this proved an exceedingly good sale.

Manufacturers being now well covered, had no need for additional copper, which was held at even higher prices by producers. Then a period of comparative quietude set in, which lasted pretty well until May, in which month the business revival was quite general, and particularly so as far as copper was concerned. Some transactions took place, Lake copper being paid for at prices ranging up to 11½c., but for electrolytic copper proportionate prices could not be obtained, and the margin between the two grades—the quality of which is almost equal—became wider. The highest prices obtainable for electrolytic copper did not go above 11⅛@11¼c., while all along casting copper remained within ¼c. of the quotation for electrolytic.

In the month of June market values did not change much, but one transaction of the greatest magnitude took place, and that was a sale of Anaconda stock for account of the Hearst estate to the Exploration Company of London, which was followed in October by a further important sale, and through these about one-half of the company's stock was transferred to European capitalists. This is of the greatest interest to the trade, as it is well known that the Exploration Company of London, either directly or through its main shareholders, is in close touch with the main producers of copper in Europe.

In July the political contest made itself felt. The copper market was stagnant; orders for home trade fell off more and more, and the quantities available were all thrown on the foreign markets. While support from there was all that could be desired, values had nevertheless to suffer, and slowly but surely prices declined during that and the two following months, reaching about the lowest level in the second half of September, when, influenced by the panicky

condition of the stock exchanges and money markets, copper came down to 10⅝@10¾c. for Lake and 10¼@10⅜c. for electrolytic. These values ruled for some little time, with only slight fluctuations, but in the meantime home manufacturers had repeatedly to ask producers for extension of their contracts, a great many even preferring to cancel the purchases previously made. Several of the larger producing companies were considerably affected by this, and in the meantime Europe had to bear the brunt of the situation. It was then that the Calumet Company made a sale at 10½c. for both export and home trade, but the quantities taken up at this low figure were not as large as was anticipated.

Very shortly after this event confidence began to gain ground, and when the elections were determined, prices advanced very rapidly to 10¾@11c., at both of which figures very large quantities were contracted for, and very soon Lake copper reached 11½c., with sales of electrolytic at 11⅛@11¼c., and casting copper at 11@11⅛c. At that time orders which had evidently been kept back all came in at once, but after the first appetite was satisfied, the demand became very much quieter, and especially Europe declined to pay the high prices then established over here. Under these circumstances values gave way somewhat, and at the beginning of December Lake copper had to be quoted at 11⅛@11¼c., electrolytic, 10⅞@11c., and casting copper the same price.

This condition continued during December; in fact, before the holidays set in values eased off in some instances still more, but during the last few days of the year a sudden and large demand sprang up, partly for export and partly for home trade, and the market closed very strong and with an upward tendency, the closing prices being: Lake copper, 11⅜@11½c.; electrolytic copper, 11@11⅛c; casting copper, 10⅞@11c.

It will be noticed that values for casting copper throughout the year ruled close to those for electrolytic, which was owing to the small production of this description of copper, as nearly all the raw material available is now being treated electrolytically.

The following table shows the average prices in New York for each month of the last five years, the figures being taken from the weekly reports of the *Engineering and Mining Journal:*

AVERAGE PRICE OF LAKE COPPER PER POUND AT NEW YORK.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.	Cents
1892..	11.00	10.63	10.38	11.50	11.63	11.88	11.50	11.50	11.13	11.50	11.88	12.38	11.55
1893..	12.13	12.00	11.88	11.38	11.00	11.00	10.88	10.00	9.88	9.75	10.00	10.25	10.75
1894..	10.13	9.63	9.81	9.50	9.80	8.94	9.00	9.13	9.40	9.88	9.60	9.80	9.56
1895..	10.00	10.00	9.75	9.75	10.25	10.63	11.25	12.00	12.25	12.00	11.00	10.50	10.76
1896..	9.87	10.64	11.03	10.98	11.15	11.67	11.40	10.98	10.66	10.66	11.23	11.28	10.88

For the years previous to 1892 the quotations may be found in THE MINERAL INDUSTRY, Vols. I.–IV.

The London Copper Market in 1896.

The tendency of the copper market at the opening of the year was downward, partly owing to the unsatisfactory political outlook, and partly to the dullness characteristic of the season, with its pause in active operations at most manufacturing centers. The weakness of the financial situation in New York assisted the decline which ensued on our metal market, and which continued until near the middle of the month, when £40 10s. was accepted for cash g. m. b.'s., a fall of 30s. from December 30. Meanwhile, Lake in America had declined to 9⅞c., and the shipments thence were continuing on a very large scale, those for the first week in January alone amounting to 3000 tons. Despite the lack of new business, however, the consumption of copper was undoubtedly very active, and a reaction soon took place in the trade generally, the fears of a premium on gold in the States having been removed, and, with it, the weakness on the Stock Exchange here, as well as the disinclination to speculation which had depressed the g. m. b. market. The change was marked by active purchases of g. m. b.'s, and the large transactions in consumers' copper, the English admiralty alone taking about 1500 tons of fine copper, while the higher values which resulted scarcely checked the general buying of rough and best selected qualities. In America there was more disposition to sell and, by way of exception to this rule, values there were well below the English parities. The stimulus to g. m. b. values was reinforced by active buying in connection with the manipulation of copper shares, and the last week of this month was marked by much excitement, the turnover amounting to 10,000 tons g. m. b.'s for the week, and an advance which reached £43 17s. 6d., ere the month closed. In the States fears had at last arisen as to a possible scarcity over there of consumers' copper, in consequence of the excessively heavy shipments, and sellers becoming more reserved, a rally took place, Lake recovering to 10¼c., while American consumers began to buy freely.

Early in February a dearth of best selected copper became apparent, supplies having been absorbed by the continued buying, which had been prolonged into the first week of this month. High conductivity bars had also been in very brisk demand and to obtain prompt delivery of this sort was a matter of considerable difficulty. One result of the heavy sales of refined copper was that smelters, whose stocks of raw material had become much depleted, turned their attention to Chile bars, large quantities of which were taken out of stock to be converted into best selected. During the remainder of the month, with an occasional lull, the demand continued good in England, on the continent, and in America, Lake advancing to 11c., at which price a large sale was made by the leading Lake company to home consumers, while g. m. b. s rose, with a few comparatively unimportant relapses, to £46 11s. 3d. "Strong" sheets were in fair request, but Indian demand had fallen away almost entirely. The values at the close of this month were as follows: £50 for tough, £51 5s. for best selected, £57 for strong sheets and £54 for India sheets, with yellow metal squares (for India) at 4¾d. Sulphate of copper was bought at £16 15s. per ton f. o. b. Liverpool for shipment to Boston.

The statistics for the month of February revealed an improvement of over

2600 tons for the entire month, but only about 500 tons for the second fortnight, and as a much larger decrease had been counted upon, the effect was a general disappointment, and with Lake quoted lower from America there was no speculative interest displayed, but, on the contrary, considerable pressure of sales, partly realizations, "bear" sales; and the market gave way to the extent of £2 per ton during the first fortnight of March.

The activity of consumption, however, continued, although the lower g. m. b. market deterred people from making any fresh purchases of importance; on the other hand there was a big government inquiry (about 800 tons refined copper), and the scarcity of finest sorts becoming very pronounced, the margins over g. m. b.'s grew higher than usual, and there was some disposition to abstain from purchases of refined sorts, in favor of g. m. b.'s, and when the mid-monthly figures were published, showing a decrease of nearly 2500 tons for the first fortnight, the market improved quickly, £45 17s. 6d. for cash g. m. b.'s being realized. This improvement was, however, not of long duration, the good effect of the statistics being neutralized by the advices of heavy shipments from America, while the political outlook in connection with Egypt and South Africa acted as a damper on speculation generally, and we witnessed the consequence in the shape of a relapse to £44 11s. 3d. The producers' returns also showed that production in the States was proceeding on a very heavy scale, the figures for the month of February being at the rate of 116,000 tons per annum, while the state of trade and finance over there gave rise to serious apprehensions.

The fact that, notwithstanding the large shipments above alluded to, the month of March witnessed a total decline in the visible supplies of 3500 tons, naturally gave a certain amount of stimulus to the market, so that April opened in better tendency, cash g. m. b.'s rising to £45 12s. 6d. An interesting feature may be noted here in connection with the stock of Chile bars, which had been undergoing such constant depletion that in the eight months previous it had dwindled from 400,000 tons to 30,000 tons, while the London stock of Japanese bars had decreased from 3500 tons to 890 tons. A consequence of this was a gradual lessening of the contango on three months' owing to the relative enhancement of the value of spot bars. A continuance of the large American shipments, greater accessibility on the part of producers on both sides to proposals for fresh business, together with the restriction in England at this juncture of consumptive demand by the closing of works for the holidays—these were the leading factors responsible for the fall which ensued from the point named above to £44 2s. 6d. toward the middle of the month; but the fact that despite America's huge production and heavy exports the figures of supply and delivery for the first half of April gave a decline, though a small one, was looked upon as a further proof of the very satisfactory scale upon which consumption was proceeding, and this produced a good effect upon the market, which recovered to £45 11s. 3d., easing off again to the extent of about 7s. 6d. before the close.

The month of May opened with limited activity in speculative operations, and values remained very steady during the first ten days without any feature of interest developing itself. The Board of Trade returns for the previous

month clearly reflected the satisfactory state of consumption, both of raw and refined copper, but fresh purchases were only made very sparingly, while apathy continued to characterize the Indian market for copper and yellow metal sheets. About the middle of the month a fresh impulse was imparted to our market by the brighter news from the States, where large quantities of Lake were purchased at the advanced price of 11c. by American consumers, while Europe also bought freely, the rise which took place in the g. m. b. market rendering the values of American fine sorts less disproportionate than they had been of late to values on this side. Large purchases of g. m. b.'s were made, the bulk, perhaps, being for American account, and from £45 8s. 9d. on the 11th of the month the value rose quickly to £47 10s. This higher level bringing out some realization sales, we receded to £47 3s. 9d. at the close. Meanwhile, the price of Lake had again advanced, some of the producers obtaining 11½c. with buyers over thereat.

Speculative buying continued in full swing during the early part of the following month (June), the purchase being partly in cover of "bear" sales and partly with a view to higher values. This resulted in a rise to £47 18s. 9., for spot, and after a temporary setback to £47 12s. 6d. there was a renewal of the strong upward movement, which culminated in the payment of £50 7s. 6d. for sharp cash on the 15th. As usually happens, the high prices induced realization, resulting in a drop to £48 12s. 6d. in two days, but the remainder of the month was steady at from £50 1s. 3d. to £49 7s. 6d. Consumptive demand continued quiet, though the various copper-consuming industries were very busy. In America, Lake, after rising to 11¾c., had receded to 11½c. again, and shipments thence continued on a large scale, the figures for the month of June amounting to nearly 10,000 tons.

July was a rather dull month as far as speculation went, and after a comparatively firm opening under the influence of the good statistics, there came a decline from £49 13s. 9d. cash g. m. b.'s to £48 2s. 6d. on the 30th, the month closing about 7s. 6d. per ton better. The main causes of the decline were the continuance of the retrograde movement in America (brought about by the unsatisfactory state of finance over there, and resulting in a fall in the price of Lake to 11c.) and the persistently heavy shipments to Europe, where there was very limited demand on the part of consumers, notwithstanding that most works had very full order-books. One cause of the small amount of buying was to be found in the fact that consumers had already laid in very considerable stocks under apprehension of a possible "corner," which had for some time been threatened by the action of a group of outside speculators who had been buying up large quantities of g. m. b.'s, but this eventually was by the well-informed regarded as a bugbear, for whereas, during the Secretan period, only Chile bars were deliverable as g. m. b.'s, at the present moment a whole host of recognized brands of tough, best selected, electrolytic, and other sorts of fine copper, are included under the category of g. m. b.'s, and the list was added to at this time with the view of still further providing against even the off-chance of such manipulations succeeding. Accordingly, after repeated trotting-out of the said bugbear, the thing died a natural death, without any palpable effect, beyond a transitory substitution of a backwardation of about

2s. 6d. upon three months' g. m. b.'s, for the customary contango. With respect to consumers' copper, we should here allude to the increasing competition of various makes of electrolytic copper with best selected, the former sort being produced in ever-increasing quantities, and being sold, especially by America, at low prices. The consequence of this was a cheapening of best selected copper, which was also suffering from over-production.

Early in August there was rather better buying on the part of consumers, but the fears entertained in some quarters of a possible victory of the silver party in the United States led to very low prices being taken by Americans, who made, especially to the Continent, extensive sales during this month, and this reacted upon the g. m. b. market, a decline from £49 1s. 3d. early in the month to £46 6s. 3d. almost at its close being the result. Lake had meanwhile come down, the quotation at the beginning of September being 10⅝c. The demand in England for manufactured copper was active, but India's apathy showed small sign of giving place to activity. Perhaps the most lively demand of all was experienced for high conductivity wire, which led to largish purchases by wirebar makers of the requisite copper.

September opened with a rise in g. m. b.'s, from £47 to £48 8s. 9d. in sympathy with the advance which took place in Rio Tinto shares, the speculative purchases made being of considerable extent. The American market remained steady with Lake at about 10¾c. and with no great quantities of copper offered thence to Europe. In the States the attitude was one of expectancy, but the general trend of opinion was in the direction of confidence as to the election resulting in a victory for the "sound-money" party, in which event a great improvement in trade was expected, the fact being that the great uncertainty prevailing as to the issue of the momentous struggle between the conflicting interests of the silverites and the goldites had very seriously checked the progress of commercial and also industrial activity, leading to the closing of many works and to the exporting of huge quantities of copper which had for some time been flooding the European markets. The genuine activity of consumption on this side found its most striking proof in the fact that, notwithstanding these huge imports from America, the visible supplies had been steadily decreasing, until at the end of September they had touched the lowest point known for very many years past, 34,000 tons. As to the speculative market, there only remains to point out the reinstatement of the contango on three months' g. m. b.'s, the value of which had, during the greater part of August and September, been either level for the two positions (cash and three months) or with a slight backwardation on the latter. September closed at £47 15s. for cash and £47 18s. 9d. three months.

October opened with a setback to £47 for cash g. m. b.'s in sympathy with the decline in tin and weakness on the stock exchanges, especially in Paris. The decline made further progress toward the middle of the month, and cash was depressed to £46 13s. 9d., while the contango on three months rose to 10s. per ton, the relative depression of the former position being chiefly due to heavy sales of the same, in connection with the carrying forward for three months of large quantities of copper, against the purchases made by the outside speculators alluded to earlier in this report. As the date of the United

States presidential election drew near, numerous transactions in g. m. b.'s were entered into with the option for buyers of cancelling in the event of Bryan being elected. A few sales were also made with sellers' option to cancel should McKinley be elected. Before the month closed g. m. b.'s rallied to £48 5s., the end being about 5s. lower than that. As to consumers' copper, right at the end of the month, American producers were entirely out of the market, waiting the result of the election, and the tendency here was rather firmer, with more demand, best selected especially being in request, and correspondingly scarce, the consequence being that the margin had now recovered a good portion of the ground lost earlier in the year.

Early in November, on the defeat of the silver party becoming known, the g. m. b. market at once rose, and business in cash took place at rapidly rising prices, £48 13s. 9d. being done in the first week, and in the second as high as £49 7s. 6d. In the following week the highest points were reached, when cash changed hands at £49 15s. and three months at £50 7s. 6d. Lake meanwhile had advanced to 11½c.; European consumers had been buying steadily at improving prices, and English tough and best selected copper was even bought by one of the large American export houses, presumably for shipment to the States, though possibly purely as a speculation. Manufactured copper continued in good request, but India sheets and yellow metal squares for India languished still, owing to the famine and strike in India.

American shipments continued very large, those for the third week in November being, in fact, about the largest on record for a single week, 4207 tons. The month closed with g. m. b.'s at £49 cash and £49 11s. 3d. three months.

The statistical position did not recover to the point at which it stood in September, when the visible supply fell to about 34,000 tons as already mentioned, but the large increase of October (1352 tons) was partly wiped off by the result of the November stock-taking, the total visible supplies at the close amounting to 34,800 tons.

As to the general position of the copper trade, we have to report phenomenal activity in the various copper-consuming industries in Europe; the important engineering and shipbuilding branches, the government arsenals, etc., being very busy indeed, while the brass and ammunition makers are also full of work; and as to the requirements of copper for electrical purposes, these seem to be increasing on a large scale from month to month.

The following table shows the fluctuation of prices of g. m. b.'s in London during 1896, the prices being in sterling per ton of 2240 lbs.:

RANGE OF PRICES OF COPPER (G. M. B.) IN LONDON, 1896.

Month.	Highest	Lowest	Average	Month.	Highest	Lowest	Average	Month.	Highest	Lowest	Average
	£ s. d.	£ s. d.	£ s. d.		£ s. d.	£ s. d.	£ s. d.		£ s. d.	£ s. d.	£ s. d.
January..	43 17 6	40 10 0	41 11 3	May....	47 11 3	45 6 3	46 3 11	September.	48 8 9	47 2 6	47 16 6
February.	46 12 6	43 7 6	44 14 2	June....	50 8 9	47 0 0	48 16 2	October. ..	48 5 0	46 13 9	47 9 3
March	46 7 6	44 3 9	44 5 9	July....	49 13 9	48 2 6	49 1 0	November.	49 15 0	48 1 3	49 1 9
April.....	45 12 6	44 1 3	45 0 7	August.	49 2 6	46 13 9	47 14 8	December..	49 10 0	48 7 6	48 14 4

The highest quotation for the year was £50 8s. 9d., in June, and the lowest £40 10s., in January; the average for the whole year was £46 14s. 1d. per ton.

Copper Sulphate Manufacture at Freiberg, Saxony.

By Albert Doerr.

Copper sulphate is produced in Freiberg (Muldenhütten and Halsbrücke) from two sources: 1. Copper matte. 2. The cupriferous solution resulting from the precipitation of silver from the solution obtained in parting bullion.

1. The copper matte is originally a product of the combined base bullion and matte smelting, and undergoes, before it is sufficiently freed from impurities and rich enough in copper to be employed in the production of copper sulphate, a long series of concentrating operations.

The products of the lead blast furnaces in Freiberg are: Lead bullion; lead and copper matte; speiss; slag. As is customary all over Germany, all slag from furnaces running on ore is resmelted twice. These two operations, called *Schlacken-Verändern* (slag changing) and *Schlacken armmachen* (slag impoverishing), serve to cleanse the slag and to concentrate the copper into a matte, and the nickel, cobalt, and arsenic into a speiss.

The copper-lead matte after a kiln-roasting is added to the charge in the "slag-changing" operation. Such a charge consists of roasted matte, raw matte, slag (from ore furnace), and impure litharge from cupellation. The products are again lead bullion, copper matte, speiss, and slag. The matte is reroasted either in kilns or stalls, and is then added, with slag from the first operation, to the charge in the second slag impoverishing operation. This charge consists of roasted matte low in copper, crusts from the liquating furnace, slag from the reverberatory concentrating furnace, slag from first operation and impure litharge. The products are copper matte, lead bullion, and slag. The slag from this operation goes to the dump; the matte, which now contains about 30 to 35% copper, is concentrated in a small Welsh reverberatory furnace, where it is freed, as far as possible, from impurities. The charge consists of 2750 lbs. roasted matte, 220 lbs. raw matte, 110 lbs. copper sweepings and cement copper, 550 lbs. quartz, and 440 lbs. heavy spar. The operation is as follows: The furnace, after having been charged, is tightly closed and all doors and openings luted with clay. After three hours' high temperature the charge is melted, when the rabbling doors are opened, and the contents of the furnace given a thorough stirring, after which the doors are again closed and luted and the temperature considerably raised, with the employment of forced draught toward the close of the operation. On the charge becoming thoroughly liquid the doors are opened, the slag drawn off, and another charge introduced, without, however, tapping out the matte already formed. The same operation is repeated and at its completion the slag is drawn off and the matte tapped, which is then done after every second charge. Four charges are run through in 24 hours. Owing to the high temperature in the furnace SO_3 is set free ($BaSO_4 + heat = BaO + SO_3$), but rapidly decomposes into O and SO_2. The oxygen oxidizes the iron present and this with the BaO and the quartz forms a very fusible singulo-silicate slag. Barium sulphide is also formed, but decomposes, and any Cu_2O, or metallic copper, which is formed is again reduced to a sulphide. The resulting matte is of a bluish color, a columnar structure, and the following composition: Au, trace; Ag, 0.31%; Cu, 73.95%; Pb, 4.85%; As, 0.18%; Ni+Co, 0.21%; Bi, 0.02%; Sb, 0.06%; Fe, 0.13%; Zn,

trace; S, 18.98%. The slag from this operation is added to the charge in the second (slag impoverishing) smelting in the blast furnace.

In concentrating the matte in the reverberatory a small quantity of black copper is always formed; this is returned to the furnace.

The concentrated matte, carrying from 70 to 75% copper and never more than 0.20% Fe, is dead roasted in reverberatory calciners and shelf-burners, a charge of from 1100 to 1600 lbs. being thoroughly roasted in 16 to 20 hours. Toward the close of the roasting tests are made to ascertain if all the Cu_2S has been oxidized to Cu_2O. For this purpose a small portion of the matte is treated in a beaker with HNO_3; so long as the brown fumes are given off the matte is not thoroughly roasted. After the roasting is finished the matte is ground in a ball-mill and passed through sieves, the fines going to the leaching vats and the coarse back to the roasting furnace.

The leaching vats, which are lined with hard (antimonial) lead, are 1.5 meters high and 1.2 meters wide. They are first charged with 440 lbs. chamber acid (about 50° B.), water, and about 880 lbs. of solution from previous work. The liquor is heated with steam, after which 220 lbs. of the dead-roasted matte are slowly added, the liquid being constantly stirred meanwhile. In from 45 minutes to an hour the Cu_2O is completely dissolved. After being allowed to settle the solution is drawn off into filtering vats and thence to the crystallization vats. The latter are made of wood, lined with hard lead, and will hold about 2¼ cu. meters of solution. Strips of hard lead are suspended in the solution from cross pieces on top of the vats to give more surface for the crystals of $CuSO_4+5H_2O$ to deposit on. After 8 to 10 days the crystallization is considered finished, and the strips of lead, now quite covered with copper sulphate, are lifted out, the crystals are removed, and after being washed and dried are ready for the market. The crystals from the bottoms and sides of the vats are impure, and are consequently dissolved in water heated by steam, and allowed to recrystallize. The solution for the second crystallization must possess a strength of from 32° to 34° B. Old solutions which have become too poor in copper are treated with scraps of metallic iron to precipitate cement copper, the iron going into solution as $FeSO_4$, which is crystallized out and sold. The cement copper is added to the charge in the reverberatory concentrating operation. In the dissolving vats there remains a slime which contains lead, copper, and antimony, and is rich in silver. This slime is collected, washed, and added to the charge of the blast furnace running on rich ore.

2. A small amount of copper sulphate is also produced from the parting of silver and gold. The granulated silver from the cupellation furnace is dissolved in H_2SO_4 of 66°, from 900 to 1100 lbs. being dissolved at a charge, the time required for its solution being 8 to 10 hours. After being allowed to settle for 10 to 12 hours the $AgSO_4$ solution is drawn off into filtering vats, and from there to precipitation tanks, where it is diluted to about 20° B., and the silver precipitated with metallic copper, the solution being meanwhile heated with steam to about 70° C. The resulting $CuSO_4$ solution and the wash-water are then treated in the same manner as the matte solutions.

NOTE.—The slag from the reverberatory, concentrating furnace has this composition: SiO_2, 30.60%; PbO, 14.51%; CuO, 4.40%; FeO, 8.10%; NiO+CoO, 0.38%; ZnO, 1.95%; Al_2O_3, 1.80%; BaO, 28.63%; CaO, 7.70%; MgO, 0.72%; S, 0.18%; total, 98.97%. The matte assays: Ag, 0.31%; Cu, 73.95%; Pb, 4.85%; Bi, 0.02%; Sb, 0.06%; As, 0.18%; Fe, 0.13%; Ni (Co), 0.21%; S, 18.98%; total, 98.69%.

Progress in the Electrolytic Refining of Copper.

By Titus Ulke.

With the exception of the small electrolytic separating plant operated in 1879–1880 at the Phœnixville Works with two of Weston's nickel-plating dynamos, the Balbachs of Newark, who commenced electrolytic refining early in the eighties, were the pioneers in this branch of industry. They were followed later by the Baltimore Copper Company, which in 1887 added to its works an electrolytic plant for the purification of copper by electrolysis. According to Mr. Brent Keyser, this process was not a success at Baltimore until 1890, but the profits then became so large as to lead to the erection of the Hayden plant by the Baltimore Electric Refining Company in 1891. The next year its capacity was doubled. Thus developed the great Baltimore Copper Works, which now refine two thirds of the entire Anaconda output, or between 90 and 108 tons daily.

In 1893 the electrolytic works of the Boston & Montana Company, at Great Falls, were started, while the Anaconda refinery, although turning out considerable copper, was still regarded as "in the experimental stage." The latter plant to-day, however, is able to handle at least 100 tons daily, or quite as much as the Baltimore Works.

Finally, towards the close of 1895, refining was begun at the modern copper-works of Guggenheim's Sons at Perth Amboy, with anodes obtained from arseniferous black copper from Pueblo, Colorado, and bessemer copper high in silver from Aguas Calientes, Mexico.

In 1892 there were about thirty refineries in the world, producing a total of 32,000 tons of electrolytic copper annually. Yet in 1893 the production of electrolytic copper in the United States alone was 37,500 tons or about one-fourth of the total copper production of the United States; in 1894 it had risen to 57,500 tons or almost one-third of the entire copper output; and in 1895 and 1896 it reached the enormous totals of 87,000 and 124,000 tons respectively, which equal one-half to six-tenths of the entire American copper production or nearly one-third of the production of the world. The estimated daily output of the eleven refineries in this country now in operation is 342 tons. From the blister copper treated are obtained gold and silver slimes containing between 37,000 and 39,000 ozs. of silver per diem, or about 14,000,000 ozs. per annum, besides over 68,000 ozs. of gold. The slimes taken from the copper tanks are generally screened from scrap copper, and boiled in dilute sulphuric acid, dried, cupelled with lead and parted by acid, although Guggenheim's Sons employ successfully the Moebius process. The current densities employed in the various works range between 7 and 20 amperes per sq. ft. of cathode surface, according to the quantity of arsenic, iron, and silver contained in the blister copper treated.

It is interesting to note that the cost of refining copper has decreased from about $19.60 per ton in 1892 to $8.00 in 1896. In the West the average cost, exclusive of office expenses, is from $12 to $14, and in the East from $8 to $10 per ton of blister. This compares very favorably with the cost of electrolytic refining in Europe, which lies between $13 and $18.

The great advance in electrolytic copper refining during 1896, especially in the United States, is shown by the subjoined table. No claim is made for absolute accuracy, since data relating to several of the companies had to be estimated.

ELECTROLYTIC COPPER REFINERIES IN THE UNITED STATES (March 1, 1897).

No.	Name of Company.	Location.	Number of Generators. Kw = Kilowatts.	No. of Tanks.
1	Baltimore Copper Smelting & Rolling Co...	Baltimore, Md.	8 Edison @ 80 Kw. 3 Westinghouse @ 80 Kw.	540
2	Anaconda Mining Company.....	Anaconda, Mont...	5 Westinghouse @ 270 Kw. 2 " @ 220 Kw.	1200
3	Nichols Chemical Company.	Laurel Hill, N. Y..	3 General Electric	90 large tanks
4	New England Electrolytic Copper Company.	Central Falls, R. I.	2 General Electric @ 50 Kw. 6 Edison @ 65 Kw.	560
5	Bridgeport Copper Company.....	Bridgeport, Conn..	3 General Electric @ 60 Kw.....	432
6	Balbach Smelting and Refining Company...	Newark, N. J.....	8 Excelsior @ 30 Kw.....	432
7	Guggenheim Smelting Company.....	Perth Amboy, N. J.	2 General Electric @ 200 Kw.....	360
8	Boston & Montana Copper and Silver Co....	Great Falls, Mont..	4 Thomson-Houston @ 180 Kw..	288
9	Irvington Smelting and Refining Company..	Irvington, N.J.....	2 Hochhausen @ 60 Kw.....	96
10	Chicago Copper Refining Company.....	Blue Island, Ill....	2 Edison @ 64 Kw.....	165
11	Omaha & Grant Smelting Company.....	Omaha, Neb.....	1 Excelsior @ 15 Kw.....	48

No.	Arrangement of Electrodes.	Estimated Daily Output of Electrolytic Copper in Tons. (2000 lbs.)	Kind of Material chiefly treated.	Estimated Current Density in Amperes per Sq. Ft. of Cathode Surface.	Estimated Cost of Refining per Ton (exclusive of Office Expenses).	Disposition of Doré Bars obtained from Slimes.	Estimated Daily Output of Gold and Silver from Slimes.
1	Series and Multiple.	90–108	Anaconda anodes.	8–12	$8–9	Parted by acid.	11000 oz. Ag 50 " Au
2	Multiple. tanks in series.	50 (capacity 120)	Anaconda anodes.	8–9	12–14	Parted by acid.	5500 " Ag 25 " Au
3	Series.	40 (capacity 50)	Cinders from acid works.	10–15	10	Sold.	3000 " Ag 17 " Au
4	Multiple. tanks in series.	40	Boston and Montana anodes.	13–20	8–9	Sold.	2400 " Ag 8 " Au
5	Multiple.	30	Parrot copper.	13–20	8–9	Parted by acid.	1800 " Ag 6 " Au
6	Multiple. tanks in series.	30	Orford copper.	7–10	8–9	Parted by acid.	2300 " Ag 13 " Au
7	do.	27	Mexican and Colorado copper.	7–10	9	Parted electrolytically.	10000 " Ag 60 " Au
8	do.	20 (capacity 40)	B. & M. copper.	15	12	Slimes sold.	1200 " Ag 4 " Au
9	do.	9	B. & M. copper.	15	8–9	Parted by acid.	500 " Ag 1 " Au
10	do.	5	By-products.	7–10	10	Slimes sold.	400 " Ag 1 " Au
11	do.	1	By-products.	7–10	12	Parted by acid.	300 " Ag 2 " Au

Estimated total daily production of electrolytic copper, 342 tons, or over 124,000 tons annually.

Estimated total gold and silver obtained from blister-copper annually: 14,016,000 oz. Ag.; 68,255 " Au.

ELECTROLYTIC COPPER REFINERIES IN EUROPE.

Country.	Name of Company.	Location.	Arrangement of Electrodes.	Estimated Annual Production of Electrolytic Copper in Tons. 1886.	1895.
Germany	North German Refinery	Hamburg	Multiple.	775	1728
	Communion Hüttenwerke	Oker	"	115	550
	Mansfeld Copper Company	Eisleben	"	734	(?)
	Borchers Bros.	Goslar	"	....	360
	Messrs. André	Frankfurt	(?)	(?)	(?)
France	Eschger-Chesquière et Cie	Biache	Multiple.	146	781
	Société de Cuivre de France	Éguille	"	....	1100
	H. Roux et Cie	Marseilles	"	81	(?)
	Grammont Affinerie	Pont de Chéruy	"	....	400
England	Bolton & Sons	Widnes & Froghall..	"	2340	7000
	McKechnie Bros.	Widnes.	"	....	3340
	Vivian & Co	Swansea	"	2080	2600
	Elliott's Metal Extraction Co.	Pembrey	"	462	3120
Russia	Kalakent Copper Works..	Kedabeg	"	78	270
	Bogoslousk Copper Works	Ural Mts	"	(?)	(?)

The absence from this list of the Baltimore Electric Refining Company is due to the fact that it was incorporated with the Baltimore Copper Smelting and Rolling Company on Jan. 1, 1897. The Electrolytic Copper Company of Ansonia, Conn., which operated 75 tanks by the Smith process and produced between 3 and 4 tons daily, failed in 1896. Lewissohn Bros. have taken charge of this concern's affairs and, as the Ansonia Refining Company, will operate the plant in the future. The electrolytic works of the Pennsylvania Salt Manufacturing Company of Natrona, Pa., burned down last year and have not been rebuilt, while the copper refineries of the St. Louis Smelting and Refining Company, at Cheltenham, Mo., and of the American Nickel Company, at Camden, N. J., both of which contain 48 tanks with electrodes in multiple, are still idle. Washburn, Moen & Co., who formerly refined 3 tons daily by the series process, gave it up two or three years ago, and finally the Salt Lake Copper Refining Company, which is frequently mentioned in technical papers as operating a refinery, has not yet built one.

THE COLORIMETRIC ASSAY FOR COPPER.

Geo. L. Heath in the *Journal of the American Chemical Society*, XIX. 1, January, 1897, describes improvements in the colorimetric assay for copper. He asserts that standard colors permanent for one year or more may be prepared if H_2SO_4 be used to dissolve the copper instead of HNO_3, if ammonia is in sufficient excess to produce the clear neutral blue, and if the standards are kept in colorless, glass, very tightly stoppered bottles.

In preparing standards weigh out 0.3 g. of pure copper, dissolve in 5 cc. of HNO_3 of 1.40 sp. gr. and 5 cc. of H_2SO_4 of 1.84 sp. gr. Evaporate carefully until white fumes begin to come off, then cool and add 25 cc. distilled water and strong NH_4OH in sufficient excess to produce a clear solution again. Prepare at least 3 liters of 1 part $NH_4OH + 6$ parts H_2O, and dilute the copper solution until 1 cc. = 0.0025 g. Cu, since 2.5 g. of substance will be taken for analysis, and the degree of dilution then permits the whole range of colors to be made and diluted to a standard volume of 200 cc. Measure accurately the portions of standard solution required for a set of colors from 0.10 to 1.30% Cu, and make up each one to 200 cc. with the dilute ammonia.

If 0.5 g. be taken substances containing as much as 6% Cu may be determined by this method, but it is best adapted to material with less than 2%, of which it is best to take 1.25 or 2.5 grams.

Transfer the weighed sample to a $4\frac{1}{4}''$ casserole, barely moisten with water, add 15 cc HNO_3 (sp. gr. 1.42), and warm gently until the covered dish is filled with red fumes, but not long enough to evaporate the acid. When the decomposition appears to be complete add 5 cc. H_2SO_4 (sp. gr. 1.84) and boil until the HNO_3 is nearly driven off and the mass has become a thick but rather soft paste. This insures dehydration of silica, and change of copper to sulphate. From this point the procedure may follow one of three courses.

1. Cool the pasty mass in the casserole, add 120 cc. H_2O, stir well, warm, and precipitate iron and alumina with NH_4OH (sp. gr. 0.9), using enough to make

the liquid smell strongly. Filter into one of the bottles prepared for color-comparison, wash with dil. NH_4OH (1 : 10) until filtrate is colorless. Add water to mark on bottle and compare with standards. If the slag or tailings contain less than 1% Cu this method is sufficiently accurate.

2. In this course the copper is separated by precipitation with zinc or aluminum in the ordinary manner.

3. To the contents of the casserole after boiling with HNO_3 and H_2SO_4 add only 70 cc. H_2O, stir well, add rapidly 30 cc. NH_4OH, filter, wash twice with 10 cc. dil. NH_4OH (1 : 10). Then by means of a rod and 50 cc. H_2O measured in a small wash-bottle transfer the precipitate as completely as possible to the casserole without damaging the filter-paper. Add drop by drop enough H_2SO_4 to dissolve the precipitate, and follow with 20–25 cc. NH_4OH (sp. gr. 0.9). Filter and wash with dilute NH_4OH, make up to mark in test-bottles, and compare with standards.

The author considers the third method better than the second, since the reprecipitation of iron and alumina frees all but a trace of copper, and there is always a danger of incomplete reduction of copper by aluminum or retention by the silica of a little copper (which would be extracted by ammonia), and more time is required, especially if the silica be removed by filtration before reduction with aluminum.

FELDSPAR.

THIS mineral is produced in the United States entirely for consumption in the pottery trade, in which it is known commonly as "spar." It serves as a constituent in the body of white earthenware or porcelain, playing the part of a flux to bind the mass together, and is also one of the elements of porcelain glaze.

The chief sources of feldspar in the United States are Brandywine Summit, Pa., Branchville and Glastonbury, Conn., Georgetown and Topsham, Me., and Bedford, N. Y. Some is also obtained from Delaware, Virginia, and North Carolina, the two Southern States being recent sources of supply.

The mineral occurs in veins often of large size and usually contains quartz and mica intermixed, which have to be separated by hand-picking. There is much spar, however, that contains fine quartz, or quartz so interwoven with it that it cannot be separated by breaking and hand-sorting. The clean spar is shipped to Trenton, N. J., or East Liverpool, Ohio, where it is ground fine with heavy stone rollers and screened. The pottery trade requires that the feldspar shall be practically free from iron oxide, mica, and quartz, and that when subjected to the heat of the pottery biscuit kiln it shall fuse into a white opaque glass.

The Maine feldspar is considered the best; that of Connecticut is of good quality, but it is not easy to obtain much of it free from quartz. The Brandywine Summit spar, which is produced by a number of small quarries, is inferior to the Maine and Connecticut, and requires a much closer sorting. The more part of it goes to East Liverpool, Ohio. An analysis of this material was as follows: Silica, 65.607%; alumina, 16.916%; lime, 0.164%; magnesia, 0.978%; soda, 2.114%; potash, 12.915%; loss on ignition, 0.496%—total, 99.190%.

The production of feldspar in 1895 was 22,195 long tons, valued at $104,082; in 1896 it was 22,199 long tons, valued at $112,829. About 60% of the annual production is derived from Brandywine Summit, the Maine and Connecticut quarries furnishing a little less than 20% each. The value of feldspar delivered at Trenton, N. J., is $5@$5.50 per long ton, while at East Liverpool, Ohio, it is worth $6.50@$7. The cost of milling is about $4 per ton.

A good deal of rock known as "St. Austell stone" is imported from England and used at Trenton, N. J., as a substitute for feldspar, over which it has advantages for certain purposes. It is also a good deal cheaper than feldspar. This "St. Austell stone" is a partially decomposed feldspathic granite, free from mica.

FLUORSPAR.

THIS mineral is used as flux in iron-founding, as a base of hydrofluoric acid, and in the manufacture of opalescent glass. The American market is supplied by the mines at Rosiclare, Ill., and by importations from England, chiefly from Newcastle. The latter mineral can be brought hither so cheaply that the Rosiclare cannot compete with it on the Atlantic seaboard for those demands which do not need mineral of exceptional purity. The American mineral is, however, of higher quality than any other in the market, and is brought to the coast for the manufacture of hydrofluoric acid, and at times even has been exported. American hydrofluoric acid is regularly exported to Europe, where it is in demand on account of its superior quality.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton	Year.	Tons.	Value.	Per Ton
1885....	5,000	$22,500	$4.50	1889.....	9,500	$45,835	$4.82	1893.....	9,700	$63,050	$6.50
1886....	5,000	22,000	4.40	1890.....	8,250	55,328	6.70	1894.....	6,400	38,400	6.00
1887....	5,000	20,000	4.00	1891.....	6,320	38,000	6.00	1895	4,000	24,000	6.00
1888....	6,000	30,000	5.00	1892.....	9,000	54,000	6.00	1896.....	4,000	24,000	6.00

The statistics of fluorspar in the United States are very unsatisfactory. In the customs records no separate entry is made of the importations, and it is consequently impossible to learn precisely their amount.

The domestic production of the mineral is made entirely by one company, the Argyle Fluorspar and Lead Company, which last year declined to give us its figures. From the best information obtainable we believe that they did not differ materially from those of the previous year, and consequently we estimate the production in 1896 as 4000 short tons. Recently a new concern, The Fluorspar Company of St. Louis, has become a producer.

The geology of the fluorspar deposits of Rosiclare, Ill., was described by S. F. Emmons in a paper in the *Transactions of the American Institute of Mining Engineeers*, February, 1892, Vol. XXI., p. 31, and the conditions of mining there in an article by P. L. Fearn in THE MINERAL INDUSTRY, Vol. II. The latter volume also contained a description of the fluorspar deposits of Kentucky, by C. J. Norwood.

GEMS AND PRECIOUS STONES.

THE mining of gems and precious stones has not yet become an important industry in the United States, the production in 1896 amounting to only about $200,000 in value. The more part of this was in sapphire and ruby from Montana, and turquoise from New Mexico. In the former State several claims in the vicinity of Phillipsburg were worked in 1896, and a considerable output is reported from them. Sapphire and ruby mining is also being carried on in North Carolina, mention of which will be found under the caption "Sapphire."

In New Mexico the American Turquoise Company has been exploiting mines 15 miles south of Santa Fé, and some fine specimens of turquoise are reported to have been found at Hatchita in the southern part of Grant County, while the mines in the Burro Mountains, a short distance south of Silver City, have been exploited somewhat extensively.

AMBER.

THE production of amber in Germany in 1895 was about 440 tons, or nearly 100 tons more than in the previous year. By far the larger part of this is put out by the two mines of Palmnicken and Kraxtepellen, belonging to the firm of Stantien & Becker, while the smaller part is obtained by dredging and searching the shore of the Baltic Sea. The industry employs about 1200 persons.

The occurrence and production of amber in Burma is described in the article on that country in a subsequent part of this book.

DIAMONDS.

DIAMONDS are obtained from Kimberley, Jagersfontein, and the Bloemhof district, in South Africa; from Brazil, and from India. By far the more part come from Kimberley, where the mines are owned by the De Beers Consolidated Company, which controls prices by regulating its output to the demand of the market.

The production of Jagersfontein, which is situated in the Orange Free State about 80 miles from Kimberley, in 1896 was 218,886 carats, valued at £382,780.

In the Bloemhof district, South African Republic, diamonds are obtained from the gravel of the Vaal River. The production in 1895 was 2469 carats, valued at £4938. The production of diamonds in Cape Colony, practically the whole being from Kimberley, was 2,507,408 carats, valued at £3,013,578, in 1894, and 3,622,344 carats, valued at £4,775,016, in 1895.

De Beers Consolidated Mines, Limited, continues to be one of the most remarkable enterprises in the world. The dividend paid for the year ending June 30,

1896, was at the rate of 40% per annum, as compared with 25% per annum distributed for 1894–5, a balance of £329,357 being carried forward to the credit of 1896–7. The diamonds sold in the year ending June 30, 1896, realized £3,165,382, while the working expenses were £1,452,528. The working expenses included £60,225 for depreciation of machinery and plant, and £206,754 for interest upon the company's debentures. The profit of £1,712,854 was increased to £2,019,087 by £100,817 received from rents and dividends on sundry investments, £82,231 representing profits on the realization of investments, £7183 derived from miscellaneous revenue, and £116,001 from the balance brought forward from 1894–5. The dividend declared for 1895–6 absorbed £1,579,582, £92,539 was invested in consols on reserve-funds account, £17,609 was applied in repayment of expenses incurred in certain debenture-conversion operations, and £329,357 was carried forward, as already indicated. The average yield of diamonds per load of "blue" handled by the company in 1895–6 from De Beers and Kimberley was 0.91 carat. The average value of each carat of diamonds raised was 26s. 9d., so that the average value of each load of "blue" was 24s. 4½d. An increased price was obtained for diamonds in 1895–6, and the directors anticipate that this rise will be maintained. The quantity of "blue" and lumps on the floors at the close of June, 1896, was 3,674,357 loads, as compared with 3,452,020 loads at the close of June, 1895. The company commenced working the premier mine (Wesselton) in 1896–7, the lessee's contract having expired. Including its reserves, its debenture capital, and its share capital, the company at the close of June, 1896, possessed resources to the aggregate amount of £10,207,704.

India, which was once famous for its diamonds, now produces very few. According to the official statistics the output was 210¾ carats, valued at 16,251 rupees, in 1894, and 215 carats, valued at 19,506 rupees, in 1895.

There are no statistics of the production of diamonds in Brazil, and there is much difference of opinion as to the future of the industry in that country among those who have had experience there. Some hold that it is still in its infancy and will become important when more scientific methods are introduced, while others assert that the best ground has been worked over already. The most important fields that have been exploited are in the Serra das Lavras Diamantinas and the Serra da Sincora at the headwaters of the Paraguassu River, in the deep pools of which diamantiferous gravel is raised by divers. Recently an American imported diving-suits for carrying on this work. On the Serra Itaraça, near Salobre in the southern part of Bahia, there are important deposits which furnish very fine stones and are now being worked. Associated with the Brazilian gems is a good deal of boart and carbonado, reference to which is made under the caption "Abrasives," in the first part of this volume.

Diamonds have recently been found in British Guiana on the Himaracka-Mazaruni and Potaro rivers. Some of these deposits may be profitably worked.*

The Kimberley mines were described by William Crookes in the *Imperial Institute Journal*, January, 1897. See also "Diamonds, Where they Occur, and How to Search for Them," by Melville Atwood, *Engineering and Mining Journal*, August 15, 1896.

* D. E. Headley, *Engineering and Mining Journal*, Aug. 22, 1896.

Sapphire.

By Charles N. Jenks.

Sapphire, sometimes called telesia, is the hardest of gems next to the diamond; it crystallizes in the rhombohedral division of the hexagonal system, the six-sided doubly terminated prism being the most common form. Its sp. gr. varies from 3.907 to 4.16, with a mean of about 4, and it stands at 9 in Mohs' scale of hardness. The composition of the mineral varies somewhat as well as the hardness and specific gravity. The question of specific gravity will be referred to later.

The sapphire is unacted upon by acids and is infusible before the blowpipe, but if finely pulverized is soluble in fused borax and partially soluble in potash. With steel it strikes fire, but not so readily as some of the silicates. By collision it becomes luminous, the light varying with the color of the stone, in some instances purple, in others a deep cherry-red. Its diaphaneity varies from transparent to translucent. Sapphire exhibits a great variety of colors, and nine gems are designated according to this difference, namely: Oriental sapphire, ruby, emerald, asteria, topaz, amethyst, chatoyant, girasol, and white or colorless sapphire.* The index of refraction of sapphire is 1.76, and this high figure accounts for its great brilliancy.

The blue or Oriental sapphire is the true jeweler's sapphire; its color is generally attributed to protoxide of iron, or possibly to uranium. When of a carmine-red or pigeon-blood color the gem is called Oriental ruby, and when deep green (not too dark), Oriental emerald. The coloring material for both the ruby and emerald is thought (almost certainly) to be chromium oxide. If the gem is violet, it is called Oriental amethyst; if yellow or reddish yellow, Oriental topaz. These and other shades are thought to be produced by different proportions of oxide of iron.

The Oriental emerald and ruby are the most valuable of all gems, the emerald taking precedence on account of its extreme rarity. The emerald of commerce is not, however, to be confounded with the corundum or true Oriental emerald. Stones bearing this name obtained from Peru and Bogota are largely silica, and are comparatively soft and lacking in the brilliancy which is the distinguishing feature of the corundum emerald. This is also true of the hiddenite or lithia emerald. Both these so-called emeralds stand only at 7 in the scale of hardness and are of a lower and variable specific gravity. Of the true Oriental emeralds there are only a few in existence and these are of enormous value. A ruby of the finest pigeon-blood color, free from flaws, is worth per carat, when over one carat in size, several times more than a diamond of the first water. But the true ruby should not be confounded with the spinel ruby, which is an inferior and entirely different gem.

Among the foreign localities where the Oriental corundum gems are found may be mentioned India, Ceylon, China, Armenia, Siberia and Bohemia. Burma, Siam, and Ceylon yield the finest rubies. The Mogok district, which covers an area of about 100 sq. miles in Upper Burma, about 70 miles from Mandalay, has for

* The use of the terms topaz and amethyst to designate varieties of the sapphire or ruby is licensed by Dana, but on this account they should not be confounded with the minerals topaz and amethyst proper, and when applied to the ruby should be qualified by the adjective "Oriental." In any case it is bad nomenclature.—Ed.

ages supplied the finest blue and red sapphires. The Ceylon stones are next in value. The Siam ruby is usually light red in color and dull in luster; while the sapphire is of a dark, dull blue without the luminous silken gloss which is the distinctive mark of the Burma and Ceylon stones.

According to Streeter the original matrix of these stones was probably crystalline limestone, the disintegration of which has distributed them along the hillsides and valleys where they are found. The credit of the first discovery of corundum gems *in situ* belongs to Col. Charles W. Jenks of Boston, who early in 1872, while mining for commercial corundum at Corundum Hill, in Macon County, N. C., at a depth of 10 to 75 ft., found Oriental sapphire and ruby crystals imbedded in the chlorites and vermiculites between two walls of olivine rock. This discovery was followed by a further find of the corundum gems in the alluvial gravel at one or more points below the veins; and later at Laurel Creek, Raybun County, Ga., sapphire and ruby crystals were also found by Col. Jenks *in situ* and in the placer below.

About 100 true gems of Oriental sapphire, ruby, and emerald were taken out by Col. Jenks at this time and by the writer, who, under the direction of his father, superintended the washing. A few of these crystals brought good prices as gems, being pronounced by experts equal in color and brilliancy to any Oriental stones of the same size. Some of the finest specimens were never cut, but were eagerly purchased by the British Museum and foreign collectors; also by the leading American colleges and collectors, including Tiffany & Co. of New York. The writer remembers one specimen which he saw picked from the wash-boxes: a mass of blue sapphire semi-transparent (a fragment of a large crystal) and about the size of a turkey's egg. This had one perfect termination, of a pure pigeon-blood color, which at the extreme end was without flaw or fracture, perfectly transparent. This specimen was sold in London to an English collector for $2500.

Mr. George F. Kunz, the distinguished gem expert, says of one of these crystals discovered at this time: "At the Jenks mine was found probably one of the first known specimens of emerald-green sapphire (Oriental emerald). It is the transparent part of a crystal 4 by 2 by 1½ in., from which several gems could be cut that would together furnish from 80 to 100 carats of very fine almost emerald-green gems (not too dark as the Siamese are), the largest possibly fully 20 carats in weight. As this gem is one of the rarest known, it makes this specimen a very valuable one."

Only about three months was spent by Col. Jenks in actual washing for gems. The financial panic of 1872–3 suddenly closed the mines, and in consequence of the death of his partner in the enterprise the property went into litigation, and after being tied up for a number of years it passed out of his control. Since this time the corundum gems have been picked up at intervals at the same and at different points within the gem-bearing area of North Carolina and Georgia, but no more systematic washing was done until early in 1896, when a New York syndicate, The American Prospecting and Mining Company, began placer mining at one or more points in Macon County, N. C., a short distance from Corundum Hill. This work is still in progress, but the success in finding gems cannot be learned. At three other points in Macon and Jackson counties small crystals of good color and brilliancy have been found *in situ* within the past year.

Among the other localities where sapphires are found in considerable number may be mentioned the deposits near Helena, Mont., where a large number of small "gem gravel" sapphires are found in the bars on the bed of the Missouri River and in some of its tributaries. These stones when cut possess a peculiar and beautiful metallic luster, but on account of their uniformly pale color do not rank with the gems from the Orient. The Sapphire and Ruby Company of Montana, Limited, which was organized in London some years ago to work these deposits and had a paid-in capital of $2,500,000, has never been financially successful for this reason. Sapphires are also found in Southern Colorado, Arizona, and near Santa Fé, where they are associated with garnets and peridotes, and are often found in the sands on and near the countless anthills. Those that the writer has seen in and from these localities are generally of a better color than the Montana stones, having also a wider range of color, with some remarkably fine specimens of red and blue. A few sapphires have been found at Vernon, N. J., but they were generally opaque and unfit for gems.

The corundum gem deposits of North Carolina and Georgia are of much greater extent than any similar single deposit of the Eastern Hemisphere. They are known to occur in Macon, Jackson, and Transylvania Counties in North Carolina, and in Raybun County, Ga., at intervals through a territory 40 miles in length by 15 in extreme width and comprising about 500 sq. miles. The deposits or sources of supply are at an average altitude of 3500 ft., or only a little less than the Burma fields, which average about 4000 ft. The placer occurrences are almost precisely identical with some of the most famous of the foreign washings, the geological as well as the topographical conditions being much the same.

A series of exhaustive tests made by the writer show an average of the crystals of both localities to be structurally alike, and the mean hardness (9) and specific gravity (4) are the same. An analysis of both the sapphire and the ruby from three different localities in North Carolina and Georgia also gave almost identical results with those from Burma, Siam, and Ceylon. An average of three analyses of the blue sapphire of India and one of North Carolina compare as follows:

	Hardness.	Alumina.	Magnetic Oxide of Iron.	Lime.	Silica.	Loss in Decomposition.
Blue sapphire of India.....	9	97.87	1.29	0.2	0.28	0.54
Blue sapphire of North Carolina.	9	98.06	1.38	.....	0.23	0.33

Acids do not act upon the sapphire from either localities, and both remain unaltered before the blowpipe. The fracture of the sapphire of India, Montana, North Carolina, and Georgia is identical. Their hardness varies somewhat in different localities, that of the North Carolina blue being apparently a trifle more (Corundum Hill, N. C., Cullasegee, N. C., Sapphire, N. C.) than the blue sapphire of Siam and Ceylon and about the same as that of Burma, Bactria, and Montana crystals. These are all a little harder than the red (ruby) and other colors from the same and different localities. This may be due to a greater proportion of silica in the ruby, an average of six analyses showing about 2.67% more than the average of the six analyses of the blue sapphire before referred to. Mr. Chenevix's analysis of the ruby of India, which has been widely quoted,

is: Alumina 90%, magnetic oxide of iron 1.2%, silica 5.25%; loss of 1.8% in decomposition. Prof. J. Lawrence Smith's analysis of the ruby of India was: Alumina 97.32%, magnetic oxide of iron 1.09%, silica 1.21%.

A peculiarity of the North Carolina crystals that does not seem to occur elsewhere is that in the same specimen may occur all the colors characteristic of each of the nine corundum gems. The writer has seen in one large semi-transparent crystal the Oriental sapphire, ruby, emerald, topaz, amethyst, and colorless sapphire. No locality in the world has thus far produced such magnificent cabinet specimens of corundum as the fields of North Carolina.

The North Carolina deposits have the most favorable gangue for a perfect gem formation, consisting in many instances almost entirely of the chlorites and soft vermiculites, and when they are systematically washed there is no reason why they may not prove of equal importance with those of the Orient. It is a fact, although it will be disputed by the trade, that several hundreds of carats of corundum gems, mostly of an inferior quality (the reason for this being that they are mostly surface specimens), are annually obtained by the natives of North Carolina, by strolling prospectors and others within this gem area, but without systematic mining. These are handled principally through local dealers in the rough or sold to curio collectors, and their identity is in most cases lost.

In the opinion of the writer, who has carefully studied the several American localities, the reasons why sapphires have not been found in Montana in greater abundance, of better color, and no more than three or four carats in weight, are that in the eruptive dike carrying the crystals they are very scattering and very small in size. Occasional fine specimens have been and will be found, but not in commercial quantity. These same reasons apply to all other localities I have visited in the United States, with the single exception of the few deposits and limited area in Georgia and of the large gem area in North Carolina. The largest sapphire crystal ever found in either hemisphere was taken from the Jenks mine (Corundum Hill) in 1872 at the time of the first work. It weighed 312 lbs., and was both red and blue, ruby and sapphire, in color. It is now in the Shepard collection at Amherst College, Mass.

Sapphires, rubies, and emeralds, highly crystallized and of brilliant luster, have been artificially produced by Deville, Feil, Frémy, and others, and are described at length by Kunz, Miers, and other authorities on gems. These are, however, small in size, rarely exceeding $\frac{1}{3}$ carat in weight, and are consequently of no especial value, as genuine stones of this size may be purchased by the pound. The artificial product is readily detected by the expert, as it is more brittle and not so brilliant, and under a magnifying glass the cavities (which occur in both natural and artificial stones) in the artificial stone are of a cloudy appearance and globular in shape, while in the true Oriental stone they are angular or crystalline. These are only a few of the more common tests, the spectroscope, polariscope, and dichroscope revealing other differences. In detecting readily any spurious or so-called corundum gems from the true Oriental stone several methods are employed by experts. One is by determination of the specific gravity of the specimen. Another is by use of the Roentgen rays; the true sapphire is, like flesh, transparent to the rays, but their imitations are, like bones, opaque to them.

The Sapphire (Ruby) Mines of Burma.*

By T. Trafford Wynne.

The principal ruby-producing district of Burma is the country near Mogok, which is situated 90 miles north of Mandalay. Kyatpyin is the present headquarters of the Ruby Mines Company, and a 15-mile circle around this place would contain nearly all the mines now being worked.

Geological.—The country rock is usually a very hard gneiss, varying to granite, or a soft micaceous schist. Large hill masses of calc-spar are met with which have evidently been forced by some upheaval through the gneissic formation. These masses, of which the hill called Pingudoung, near Kyatpyin, is the most prominent example, contain fissures and caves usually filled with "byon." This "byon," in which the rubies are usually found, varies from a somewhat tenacious clay with small, rounded quartz and other pebbles to a fine gravel almost like river sand. In color it may be from dark red to light yellow.

The great bulk of the "byon" is found as an alluvial deposit in valleys which show these characteristics, viz., a more or less shallow basin, closed at the lower end by a barrier of hard rock, and with a stream flowing through it. The "byon" having presumably been brought down by the denudation of the hills above into these basins, the rock barriers have acted as riffles, and, while allowing the lighter mud to flow over, have held up the heavier ruby-bearing ground. Those valleys also contain a varying quantity of calc-spar in huge boulders to small pebbles, and the greater the quantity of this the better seems to be the quality of the "byon."

Rubies are also found in some places with the calc-spar matrix, but the stones are imperfectly formed and full of flaws; and even if they were of any value, no method of extracting them from the matrix without injury has yet been discovered; hence no attempt is now being made to work these mines.

Exploitation.—After various failures the English company finally decided to sink a pumping pit at the lower end of the Tagoungnandine valley, and to use the available water-power to drive part of the machinery. A 4-ft. Pelton wheel was erected at a point about three-quarters of a mile below the rock-barrier, at the lower end of the valley, where 100 ft. of effective head of water was available, and the power was transmitted by means of wire ropes to the pumps. A 4-in. centrifugal was found sufficient to drain the excavation to a depth of 50 ft. during the dry weather, a 6-in. being required during the rains.

A third valley was opened up in a different manner. In order to save the heavy pumping costs, the valley being quite waterlogged and under water for several months in the year, it was decided to drive a tunnel through the rock barrier at the lower end, and so drain the valley to a depth of 40 ft. A tunnel 10 ft. by 10 ft. was driven for a length of 500 ft., the entrance being in a ravine, where a good tip for the spoil, as well as a site for the washing plant, was obtained. The washer erected there is driven by water-power, and the present cost of excavating, hauling, washing, and sorting is only 0.55 Rs. per load of 15 cu. ft. Unfortunately this plan is not feasible at Mogok, at least not for the large valley, as the length of tunnel required would make the cost prohibitive.

All these valley deposits are worked in the same manner. When the ground

* From a paper read before the Institution of Mining and Metallurgy (London), Jan. 20, 1897.

has been drained, either by pumps or drainage tunnel, the surface soil is stripped off, and tipped to waste, the byon being loaded into trucks, and either run through the tunnel, or wound up an incline and run to the washer. On arrival at the washer, it is tipped on a grizzly to remove all large stones, and is then fed into a revolving trommel, covered with 1-in.-square wire mesh, all the byon passing through this mesh going directly to the pans, while what is discharged through the end of the trommel, principally stones and lumps of barren clay, is removed for further treatment. The washing pans used are either of the Whitmore & Binyon or Davy Paxman type, both giving equally satisfactory results, the quantity passing through each pan before it is necessary to wash out varying from 3000 to 7000 cu. ft., according to the nature of the byon, as when much clay is found with the byon the washing is very much more difficult than when it is sandy nature.

The concentrated byon left in the pan, about 70 to 80 cu. ft., is run into a locked receptacle, from which it is fed to classifying trommels, in which it is washed in a strong stream of water, to remove all the sand. The different sizes are then picked over by specially selected sorters, and all valuable stones are removed; or the deposit may be again treated in a pulsator, which still further reduces the quantity, and then it is passed to the sorters.

Turquoise.

Turquoise, which is a hydrous phosphate of aluminum, is derived chiefly from Persia, where it occurs in seams as wide as 6 mm., or in irregular patches in the brecciated portions of a porphyritic trachyte and the surrounding clay-slate, near Nishâpûr in the northern part of the kingdom. The mines are situated on the southern slope of Mt. Ali-Mirsa, northwest of the village of Madèn, at an elevation of 5000 to 6000 ft. The Reish mine is the only one worked vigorously at present. This is opened to a depth of about 90 ft. The turquoise-bearing rock is hoisted in sheepskin bags, and the waste sorted out at the surface. The good stone is then sent to Meshed, where it is dressed, cut, and packed for export.

Since the Abdur Rezai mine caved in, turquoise of perfect color is very rarely found, though in all probability the hill contains considerable good stone. The Reish mine furnishes stones of good shape, but their color soon fades. According to a recent report of the British vice-consul at Meshed, although good stones are rare, there is an abundance of imperfect ones, which are eagerly bought, since all Orientals prize them, and the poorest persons like to possess one in a ring, even if it is green and spotted. Some of the stones now found look excellent at first, but the color soon fades, a green tinge develops, or white spots appear. Some of the spots can only be detected at first with the aid of a magnifying-glass, but gradually become larger, and finally spread across the stone. The color of most faded turquoise can be temporarily revived by dampness. In Meshed no one would dream of buying a turquoise without having it in his possession for some days.

Besides the Nishâpûr mines turquoise is found in the Megara Valley, Sinai; from the Karkaralinsk (Kirghiz Steppes), Semipalatinsk, Siberia; in the Kara-Tube Mountains in Turkestan, 50 versts from Samarkand, where it occurs in limonite, etc., in seams in a silicious clay-slate, which were worked at some unknown time in the past.

GOLD AND SILVER.

THERE was a large increase in the world's production of gold in 1896, which was due almost entirely to the United States. Australasia made a small increase proportionately, and Mexico and Canada showed large gains, though the total gold production of these two countries is not yet very important. Russia and South Africa both made smaller outputs than in the previous year. The world's production of silver increased. The details of the production are given in the following tables, the returns where made in crude metal being carefully reduced to the common basis of fine metal:

GOLD PRODUCTION OF THE WORLD.

Countries.	1895.				1896.			
	Reported Ounces.	Fine Ounces.	Kilograms.	Value.	Reported Ounces.	Fine Ounces.	Kilograms.	Value.
North America:								
United States	.....	2,265,612	70,468	$46,830,200	.....	2,558,433	79,576	$52,886,209
Canada	110,881	92,449	2,876	1,910,921	.....	135,956	4,229	2,810,206
Mexico	.....	270,924	8,427	5,600,000	.....	293,892	9,141	6,075,108
Central American States	.....	23,222	722	480,000	.....	24,127	750	498,450
South America:								
Argentine Republic	.....	4,500	140	93,015	.....	4,500	140	93,015
Bolivia	.....	3,144	98	65,000	.....	3,300	103	68,211
Brazil	.....	108,000	3,359	2,232,360	.....	120,000	3,732	2,480,400
Chile	.....	32,150	1,000	664,600	.....	33,758	1,050	697,830
Colombia	.....	154,000	4,890	3,183,180	.....	150,000	4,665	3,100,590
Ecuador	.....	3,800	118	78,546	.....	3,800	118	78,546
Guiana (British)	.....	122,936	3,824	2,541,077	.....	123,759	3,849	2,558,099
Guiana (Dutch)	.....	28,228	878	583,473	.....	26,878	836	555,568
Guiana (French)	.....	90,261	2,807	1,865,695	.....	101,938	3,171	2,107,058
Peru	.....	5,658	176	116,951	.....	5,948	185	122,945
Uruguay	.....	6,884	213	144,600	.....	6,880	213	144,600
Venezuela	.....	41,200	1,281	859,604	.....	39,384	1,225	814,067
Europe:								
Austria-Hungary	.....	88,500	2,753	1,829,300	.....	88,750	2,760	1,834,463
France	.....	11,640	362	240,600	.....	11,285	351	233,261
Germany	.....	114,037	3,547	2,357,144	.....	79,957	2,487	1,652,860
Italy	.....	9,002	280	186,071	.....	8,840	275	182,765
Norway	.....	500	16	10,335	.....	500	16	10,335
Russia	.....	1,537,584	47,825	31,781,851	.....	1,499,897	46,653	31,002,870
Sweden	.....	30,225	940	624,750	.....	30,382	945	627,996
Turkey	.....	392	12	8,000	.....	392	12	8,000
United Kingdom	.....	6,600	205	136,422	.....	6,150	191	127,125
Asia:								
China	.....	225,000	6,998	4,650,750	.....	225,000	6,998	4,650,750
India (British)	249,355	218,186	6,786	4,509,905	321,878	281,643	8,760	5,821,600
Japan	.....	21,000	653	434,070	.....	25,000	778	516,750
Korea	.....	10,000	311	206,700	.....	10,000	311	206,700
Malay Peninsula	.....	20,000	622	413,400	.....	25,000	778	516,750
Africa:								
Transvaal	2,505,407	2,044,412	63,589	42,257,996	2,497,946	2,023,337	62,934	41,822,376
Other countries	.....	75,000	2,232	1,550,250	.....	100,000	3,110	2,067,000
Australasia	2,350,567	2,070,335	64,396	42,793,824	2,375,950	2,114,608	65,773	43,696,653
Indian Archipelago	.....	2,500	78	51,675	.....	4,500	139	92,715
Totals	.....	9,737,881	302,882	201,292,265	.....	10,167,817	316,254	220,181,781

SILVER PRODUCTION OF THE WORLD. (a)

Countries.	1895.			1896.		
	Fine Ounces.	Kilograms.	Commercial Value.	Fine Ounces.	Kilograms.	Commercial Value.
North America:						
United States	46,331,235	1,441,051	$30,254,296	58,488,810	1,819,208	$39,245,991
Canada	1,775,683	55,230	1,159,278	3,205,343	99,699	2,150,785
Mexico	50,890,267	1,582,901	33,225,091	41,373,835	1,286,842	27,761,843
Central American States	1,607,500	50,000	1,049,500	1,623,575	50,500	1,089,285
South America:						
Argentine Republic	1,212,625	37,500	787,125	1,197,588	37,250	803,483
Bolivia	20,767,852	642,857	13,500,000	20,511,700	638,000	13,762,660
Chile	4,822,500	150,000	3,148,500	4,870,725	151,500	3,267,855
Colombia	1,720,025	53,500	1,122,965	1,646,080	51,200	1,104,384
Ecuador	7,716	240	5,037	7,716	240	5,177
Peru	3,697,250	115,000	2,513,850	3,381,569	105,181	2,268,654
Europe:						
Austria-Hungary	1,816,475	56,500	1,185,935	1,840,588	57,250	1,234,882
France	3,102,475	96,500	2,025,535	3,078,363	95,750	2,065,328
Germany	12,602,125	391,979	8,227,689	13,774,992	428,429	9,241,213
Italy	1,768,250	55,000	1,154,450	1,808,438	56,250	1,213,312
Norway	146,216	4,859	101,990	151,748	4,720	101,810
Russia	389,272	12,108	254,147	336,128	10,455	225,514
Spain	7,426,650	231,000	4,848,690	7,166,235	222,900	4,807,953
Sweden	93,238	2,900	60,871	94,048	2,925	63,090
Turkey	49,059	1,516	33,821	49,029	1,525	32,894
United Kingdom	253,985	7,900	165,821	155,593	7,950	171,482
Asia:						
Japan	1,768,250	55,000	1,154,250	1,816,475	56,500	1,218,705
Australasia	19,971,580	621,200	13,038,988	19,463,610	605,400	13,058,473
Totals	182,220,228	5,667,691	119,027,829	186,042,188	5,786,567	124,894,780

(a) The average commercial value of silver was 65.3c. per Troy oz., or $21.00 per kgm., in 1895, and 67.1c. per Troy oz., or $21.58 per kgm., in 1896.

UNITED STATES.

The production of gold in the United States increased from $46,830,200 to $52,886,209, the gain being attributable especially to Alaska, California, and Colorado, though nearly every State showed an increase. The production of silver increased from 46,331,235 to 58,488,810 ozs., a gain which was foreshadowed by the great increase in the production of lead and copper. Only a small part of the silver production of the United States is now derived from ores beneficiated by lixiviation and amalgamation. In Colorado there is not a silver-mill in operation; in Utah there are only the Ontario and Daly mills at Park City, and a small works in the Silver Reef district; in Montana, where the great Alice, Lexington, Bluebird, Granite Mountain, Bimetallic and Elkhorn amalgamating mills were in operation only a few years ago, the entire production of silver in bullion in 1896 was only 676,451 ozs.; in Idaho the Delamar mill is the only one of importance now running and the ore it grinds contains as much gold as silver; in Arizona, where the Silver King mill was formerly a great producer, the output of silver in mill-bullion is now only about 125,000 ozs. per annum; in Nevada, mills in the Tuscarora district and on the Comstock Lode are still in operation, but the Comstock ore is nearly as important for its gold contents as for its silver. It is safe to say that in 1896 only about 4,000,000 ozs. of silver were obtained in the United States by pan-amalgamation and tank-lixiviation; nearly 1,000,000 ozs. was obtained from placer gold, and gold recovered by plate amalgamation and the cyanide process. The remainder of the silver produced in the United States in 1896 was derived from copper and lead ores, or from dry ores smelted with them, being recovered

finally from silver-bearing copper ingot, or silver-bearing base lead bullion. The copper-silver mines of Butte alone produced 10,614,003 ozs. of silver in 1896. The proportion of gold obtained by smelting in the United States is a good deal less; it is doubtful if more than $20,000,000 were recovered in this manner in 1896.

PRODUCTION OF GOLD IN THE UNITED STATES.

State or Territory.	1893.		1894.		1895.		1896.	
	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)	Fine Ounces.	Value. (a)
Alaska	48,863	$1,010,100	53,868	$1,113,550	78,140	$1,615,300	99,444	$2,055,700
Arizona	57,286	1,184,200	96,313	1,990,966	95,072	1,965,300	124,770	2,579,000
California	584,370	12,080,000	656,468	13,570,397	722,171	14,928,600	737,036	15,235,900
Colorado	364,119	7,527,000	461,969	9,549,731	648,074	13,525,300	719,264	14,867,971
Idaho	79,669	1,646,900	100,682	2,081,281	86,088	1,779,600	104,263	2,155,300
Michigan	2,032	42,000	2,150	44,444	2,075	42,900	1,800	37,200
Montana	172,989	3,576,000	176,637	3,651,410	198,405	4,101,400	209,207	4,324,700
Nevada	46,367	958,500	55,042	1,137,819	75,088	1,552,200	116,620	2,410,538
New Mexico	44,171	913,100	27,465	567,751	23,810	492,200	23,017	475,800
Oregon	79,592	1,645,300	68,792	1,422,056	42,972	888,300	59,313	1,226,000
South Dakota	193,809	4,006,400	159,594	3,299,100	187,187	3,869,500	237,978	4,919,000
Southern States (b)	13,293	274,800	11,715	242,171	15,026	310,600	12,785	264,300
Utah	41,283	853,600	41,991	868,031	66,419	1,373,000	91,908	1,899,900
Washington	10,744	222,100	9,438	195,100	16,980	351,000	19,626	405,700
Other States	726	15,000	1,495	30,903	1,693	35,000	1,413	29,200
Total domestic	1,739,313	$35,955,000	1,923,619	$39,761,205	2,265,612	$46,830,200	2,558,433	$52,886,209
Foreign	103,878	.....	117,765	2,434,202	217,334	4,490,227	409,315	8,461,023
Grand total	1,843,191	.....	2,041,384	$42,195,407	2,482,937	$51,320,427	2,967,737	61,347,232
Total domestic—kgms	.....	.....	59,824	.....	70,468	.....	79,576	.....
Total foreign—kgms	.....	.....	3,653	.....	6,761	.....	12,731	.....
Grand total—kgms	.....	.....	63,487	.....	77,229	.....	92,307	.....

(a) 1 oz. gold = $20.87; 1 kgm. = $664.60. (b) South Carolina, North Carolina, Georgia and Alabama.

The production of gold in the United States in 1896, as reported by the Director of the Mint, was 2,568,132 ozs. ($53,088,000), which does not differ much from our revised figures. In both cases these figures are compiled by adding together (1) the amount of fine gold in bullion of domestic origin refined by private works, (2) the gold contents of unrefined domestic bullion deposited at the United States assay offices, and (3) the contents of ore, matte, and doré bullion exported. The distribution of the production by States in the above table is based on that of the Director of the Mint.

PRODUCTION OF SILVER IN THE UNITED STATES. (a)

Year.	1895.			1896.		
	Fine Ounces.	Kilograms.	Value.	Fine Ounces.	Kilograms.	Value.
Ores mined in United States	46,331,235	1,441,051	$30,254,296	58,488,810	1,819,208	$39,245,991
Ores or bullion imported	30,105,836	936,388	19,659,111	33,133,529	1,030,569	22,232,598
Totals	76,437,071	2,377,439	$49,913,407	91,622,339	2,849,777	$61,478,589

(a) The average commercial value of silver was 65.3c. per Troy oz. in 1895, and 67.1c. in 1896.

Alaska.—There was a large increase in the gold production, both from Treadwell Island and from the Yukon placers. At the end of the year there were 549 stamps dropping in the Territory, the Alaska-Treadwell Company having 240 and the Alaska-Mexican 120. Alaska also produced in 1896 about 150,000 ozs. of silver. This was smelted at Tacoma and Puget Sound.

Arizona.—Mr. John F. Blandy contributes the following review of gold and silver mining in Arizona in 1896:

"There was an improvement in the yield of all the gold camps and some new ones have been added to the list, notably the Fortuna mine in Yuma County, which is now producing at the rate of $100,000 per month. The Fortuna vein is from 12 to 18 ft. wide, and the ore averages $40 per ton. The Mohawk, 60 miles north of Tucson, was also added to the list of producing gold mines during the year. Its well-known neighbor, the Mammoth, which in former years produced over a quarter of a million per year, was idle, though rumor says it is soon to be provided with a new mill, since it still has an abundance of ore. The Congress, Crowned King, McCabe, and Little Jessie in Yavapai County, all increased their yield over the previous year. The Placeritas, a new mine about 30 miles southwest of Prescott, made a very excellent run during the last half of 1896. This is in a new district and is of great promise.

"Several mines will be added to the list in 1897, notably the Providence, in the Big Bug district, and some in the Cherry Creek district; also the Mescal mill on the Hassayampa, which is running on ore from the old Sterling mine. At this mine a chlorination plant was built in the 'Indian days' and has lain idle ever since, but the mine is now proving quite rich.

"Much interest is felt in the work going on at the Planet-Saturn, which has been opened very extensively and rapidly during the past eighteen months. The new cyanide works for treating the ores were started January 1, 1897. Active work has been going on in gold mining in Gila County, both in Tonto basin and near Globe. But for the inaccessibility of Tonto basin it would be a very lively district, as the gold veins of that section have been long known. Much was hoped from the Oro Blanco and Arivaca districts in southern Pima County, but expectations for various reasons have not been realized.

"The most noteworthy event of the year in Arizona was the opening of the Pearce mine in Sulphur Spring Valley, Cochise County. The ores are gold and silver, about one-fourth of the value being in gold. The vein is very large, being variously reported from 10 to 20 ft. in width and all ore, so that they are able to ship 80 to 100 tons per day of ore averaging $60 per ton. Near this mine a second one is now being opened—the Gold Cliff—and reports have it that it will prove equal in value to the Pearce."

Colorado.—The production of gold in Colorado did not increase so much as there was reason to expect at the beginning of the year. This was chiefly due to the strike at Leadville which interfered with the operations of its great gold producer, the Little Jonny, which turned out only about two-thirds of its output of the previous year. The production of Leadville in 1896 was $1,233,350 gold and 6,687,849 ozs. silver.

The Cripple Creek mines, however, made an increased output, notwithstanding the fire which destroyed the principal town, the accident in the Anna Lee (Portland) mine, and the closing down of the Independence for several months. The exact production of Cripple Creek is not easy to arrive at, since only a small part of its gold is recovered directly as bullion, and the smelters cannot conveniently report the gold contents of the ore they buy from a single district. There are now in the district seven plate-amalgamating mills with 175 stamps.

Five mills were in operation in 1896, crushing 11,800 tons of ore, valued at $144,500; it is doubtful if they saved over 55% of this; the two cyanide works crushed 48,400 tons, valued at $864,000; and the two chlorination works 15,600 tons, valued at $476,000. About 117,000 tons of ore were sold to smelters. The total production of the district was $9,099,300. The production in 1895 was $7,225,000. The total dividends paid by Cripple Creek companies in 1896 were $1,097,000, of which $105,000 was paid from sale of property. This amount does not include the profits of several important private concerns, among which is the rich Independence mine. The Portland company was the largest producer of the year, and the Victor the largest dividend payer.

In San Miguel County the great gold producer is the Tomboy mine, which is opened on a vein of wonderful continuity and richness. The ore is beneficiated by amalgamation in Huntington mills, but the present practice is wasteful, a large amount of gold being lost in tailings. These are of such slimy character that ordinary cyanide lixiviation is not feasible. The mine, which paid $480,000 in dividends during the year, was sold at the close to an English syndicate.

The production of gold in Gilpin County increased a good deal in 1896, the total number of stamps dropping having been 795, which was 170 more than in the previous year. Of the total 125 are rapid-drop.

Montana.—The production of gold in Montana in 1896 was 209,207 ozs., which was an increase of 10,802 ozs. over the previous year. This was due to the larger production of the copper mines of Butte, whose ores carry some gold. Otherwise there would have been a decrease in the gold production of the State, since the Drumlummon mine closed down temporarily, and other important mines were idle for one reason or another. The water-supply was abundant during the summer, and consequently the placers had a good year, producing 32,271 ozs. of fine gold. The large dredging plant erected on Grasshopper Creek, near Bannock, in 1895 was operated successfully throughout the season of 1896, recovering a good deal of gold that could not otherwise have been won, and it is probable that one will be put on Alder Creek during the current year. Further promise for an increased yield of gold in 1897 lies in the erection of a 400-ton cyanide works to treat the vast accumulation of tailings from the Montana Mining Company's mills (Drumlummon mine), experiments conducted in 1896 having demonstrated that the cyanide process was well suited to them. This will be the first large cyanide works in Montana.

The silver production of Montana decreased from 17,701,658 ozs. in 1895 to 15,720,022 ozs. in 1896, notwithstanding the increased output of the copper mines, especially the Anaconda. The copper-silver mines of Butte produced in 1896, 10,614,003 ozs. of silver, which is nearly 20% of the total silver production of the United States. In 1895 the production of silver in copper ores mined in Butte was 8,828,146 ozs. The production of silver from ores treated by amalgamation and lixiviation is now very small, the Elkhorn mill being the only important one in operation in 1896. The silver-lead mines, however, were active, especially those at Castle, to which district a branch of the Northern Pacific was built from Townsend, 57 miles. These mines have been idle for several years for lack of transportation facilities. New silver-lead mines were

worked near Libby Creek, in Flathead County, their output being upward of 2,000,000 lbs. of lead, and 200,000 ozs. of silver.

PRODUCTION OF GOLD AND SILVER IN MONTANA ACCORDING TO ORIGIN. (c)

Origin.	Gold.		Silver.	
	1895.	1896.	1895.	1896.
Alluvial	a23,451.465	32,270.952	a3,127.68	3,868.34
Mill bullion	a73,584.133	82,823.593	a33,913.08	676,450.66
In copper ores	33,366.000	37,473.719	8,828,146.00	10,614,002.62
In lead ores	24,291.000	6,139,869	5,311,538.00	4,014,454.58
In dry ores and concentrates classed as smelting ores.	(b)	53,206.828	(b)	411,246.24
Totals	209,320.586	211,914,901	17,701,658.11	15,720,022.44

(a) Deposits at Helena Assay Office only; in 1895, 19,755.976 ozs. of Montana gold were deposited at the New York Assay Office and at the Denver, Philadelphia, and San Francisco mints. (b) No statistics. (c) These statistics were compiled by Mr. E. B. Braden, assayer in charge of the United States Assay Office at Helena, Mont., and differ somewhat from the figures credited to Montana by the Mint.

Nevada.—There was a large increase in the production of gold in Nevada owing to the activity in the Delamar district, Lincoln County. The Comstock mines made about the same output as in the previous year. Explorations in the Brunswick Lode have not yet been successful.

Oregon.—The large increase in the gold production of this State was due chiefly to the quartz mines of the Baker City district, and to the greater activity in quartz and gravel mining in Southern Oregon.

South Dakota.—The production of gold in the Black Hills increased from $3,869,500 in 1895 to $4,919,800 in 1896. This is mostly the production of the amalgamating mills and chlorination works.

The Southern States.—Mr. W. M. Brewer contributes the following: "The Creighton and Walker gold mines in Georgia were the sources of most of the bullion sent to the mints during the year. The reopening of the old Camille mine in Georgia, which was rechristened the "Royal," was accomplished, and some bullion was produced before the close of the year, but results are not up to expectations. A large investment was made in new machinery, a chlorination plant being added to the stamp mill already on the property. The Dr. Charles mine in Forsyth County changed hands and work of developing was commenced, a new mill being erected. The industry in other sections of Georgia showed few changes from previous years. The Villa Rica district was the subject of a good many boom articles in the press, but no appreciable increase in the bullion returns was apparent.

"In Clay County, Ala., the work of reopening the Idaho mine and installing some new machinery was successfully accomplished, and the first regular clean-up in December showed a profit for the month. Aside from this there is little new to put on record relative to gold mining in this State. The ore bodies in the belt in which the Idaho mine is located are of greater extent than in any other portion of the Southern gold fields. So far as can be determined at present they promise permanency, and although the ore is low grade it can be mined and treated at a fair profit. At the Idaho mine the quantity of ore in sight above water level is estimated at nearly 300,000 tons.

"In summarizing the conditions in the gold fields in the South I feel per-

fectly warranted in saying that, if the systematic work commenced in a few locations is carried on vigorously during the coming year, the value of a large section of these fields will be fully determined, and in all probability a few more productive mines will be added to the list, in which at present I feel justified in including only the Haile mine in South Carolina, the Creighton and Walker mines and the dredgeboats in Georgia, and the Idaho mine in Alabama.

"The success of the dredgeboats on the Chestatee River is worthy of record. This method of work is applicable, however, only to the beds of the Chestatee River, and portions of the two Tallapoosa Rivers."

Utah.—There was much activity in gold and silver mining in Utah in 1896, especially in the Camp Floyd district. The production of ore was as follows:

District.	Tons.	Tons.	District.	Tons.	Tons.	District.	Tons.	Tons.
Tintic { Milled	155,500	.....	Bingham	*b*46,016	*b*46,016	Fish Springs	*b*1,315	*b*1,315
Tintic { Shipped	98,085	253,585	Horn Silver	*b*10,141	*b*10,041	Stockton	*b*1,000	*b*1,000
Mercur	*a*132,000	*a*132,000	Ophir	*b*5,379	*b*5,379	American Fork	*b*1,000	*b*1,000
Park City { Milled	49,612	.....	Silver Reef	*a*2,500	*a*2,500			
Park City { Shipped	33,153	82,765	Little Cottonwood	*b*1,300	*b*1,300	Totals	536,901	536,901

(*a*) All milled. (*b*) All shipped.

The Camp Floyd district produced 36,798 ozs. of gold, of which 25,283 were from the Mercur Company; the production of this district in 1895 was 21,430 ozs., all of which was from the Mercur Company. The amount of ore treated in this district in 1896, all by the cyanide process, was as follows: Mercur, 70,000 tons; Marion, 18,000; Geyser, 16,000; Sunshine, 15,000; Sacramento, 13,000; total, 132,000 tons. The Mercur paid $225,000 in dividends; the Sacramento, $12,000; while the Marion reported net earnings of $3168; total, $240,168, which is about 33% of the gross output. The Sunshine mill, which is rated at 200 tons per day capacity, was not started until late in the spring, and was closed down in December; hence its comparatively small crushing. The ore reserves in the district are reputed to be vast; the ore is exceedingly docile with the cyanide process; the conditions are favorable to cheap mining and milling, and consequently an increasing output is to be looked for.

The shipments from the Tintic district amounted to 4876 carloads, or say 120,000 tons, against 3837 carloads in 1895. The Bullion-Beck & Champion was the shipper of 1852 carloads, and 874 were iron flux from the Dragon mine. The four dressing works crushed 155,500 tons of ore, producing 23,915 tons of concentrates. The total amount of ore raised from the mines was about 250,000 tons. The Centennial-Eureka paid $450,000 in dividends; the Bullion-Beck & Champion, $290,000; the Mammoth, $100,000; while Eureka Hill reported $167,010 net earnings; Gemini, $37,915, and Northern Spy, $1178.

The Ontario and Daly mills at Park City crushed 49,612 tons of ore in 1896, while 35,200 tons were shipped to smelters. The product of the Ontario mine was 1,114,000 ozs. silver; of the Daly, 590,565 ozs. The Silver King paid $412,500 in dividends; the Ontario, $180,000; and the Daly, $37,500.

The ore shipments from Bingham, the silver-lead district, amounted to $50,016 tons, the Dalton & Lark, Old Telegraph, and Jordan & Galena being the heaviest shippers. The Dalton & Lark paid $75,000 in dividends, and the Old Telegraph reported $15,000 net earnings.

OTHER NORTH AMERICAN COUNTRIES.

Canada.—The production of gold in Canada is very closely reported by the Geological Survey of the Dominion and the Mines Departments of several of the provinces. The total output in 1896 was 135,956 fine ozs., or $2,810,206, which was an increase of 5.5% over 1895. Nova Scotia, which does not vary much in its output, in 1896 produced 23,501 fine ozs., or $485,683, a small increase over 1895. A new gold field was opened in Queens County, which is believed to have a large deposit of low-grade ores.

In British Columbia mining was very active. The development of the new fields in the Slocan, Kootenay and other districts proceeded rapidly. The total production of gold reported by the Mines Department of British Columbia in 1896 was 86,512 fine ozs., or $1,788,206. Of the total $544,026 was from the placer workings on the Fraser River and its tributaries, and the remaining $1,244,180 from quartz mining. The placer output shows a gain of $62,343, or 11.8%, over 1894, but is still very far below the figures of 20 to 25 years ago. The rapid growth in quartz mining in recent years is shown by the following table from the *Provincial Mines Report:*

GOLD AND SILVER PRODUCTION OF BRITISH COLUMBIA.

Year.	Gold.			Silver. Ounces.	Year.	Gold.			Silver. Ounces.
	Placer. Value.	Quartz. Value.	Total Value.			Placer. Value.	Quartz. Value.	Total Value.	
1891....	$429,811	.....	$429,811	4,500	1894....	$405,516	$125,014	$530,530	746,379
1892....	399,526	.....	399,526	77,160	1895....	481,683	785,271	1,266,954	1,496,522
1893....	356,131	$23,404	379,535	227,000	1896....	544,026	1,244,180	1,788,206	3,135,341

In Western Ontario there was considerable activity, though the only new mines opened were in the Rainy River country, and their production was not large. Some gold from the Yukon placers was worked on Canadian soil, but was reported in the Alaska production.

The total production of silver in Canada in 1896 was 3,205,143 fine ozs., an increase of 1,429,660 ozs., or 80.5%, over 1895. British Columbia produced 3,135,343 fine ozs., which was 96% of the total.

Mexico.—The production of gold in the republic is still increasing. This is stimulated by the low price of silver, which relatively reduces the cost of producing gold, Mexican finances being on a silver basis. In 1895 the production of gold was 270,924 fine ozs., or $5,600,000; in 1896 it rose to 293,892 fine ozs., or $6,075,108, a gain of 8.5%. The silver production of Mexico decreased from 1,582,901 kgms., or 50,890,267 ozs., in 1895 to 1,286,842 kgms., or 41,373,835 ozs., in 1896.

Central America.—The statistics of the precious metals in these countries are very incomplete. Costa Rica and Nicaragua are the chief producers of gold. From the best available information there was an increase in the gold production in 1896, and it may be set down at 750 kgms., equal to 24,127 fine ozs., or $498,705. The production of silver does not increase; for 1896 we estimate it at 50,500 kgms., or 1,623,575 ozs.

SOUTH AMERICA.

Argentina.—The gold and silver production of this republic is chiefly from the Andes. A small quantity of gold comes from Patagonia.

Bolivia.—Bolivia ranks next after Mexico as a silver producer. The Huanchaca Company is still in difficulties, but showed some improvement over 1895. The gold production of Bolivia is not large, and shows little change, though attention has been recently called to some of the old placers which yielded largely under the Inca and Spanish workings.

Brazil.—The best-known gold mines in this country are the Ouro Preto and St. John del Rey, worked by English companies, and the Morro de Fogo, Momondo and Jacobinas, owned by Brazilians. A French company has begun to work a large property in Minas Geraes, but the results are not yet considerable. Brazil has immense placer deposits still unworked.

Chile.—Exact statistics cannot be obtained, but we have a careful estimate, made by the Sociedad Nacional de Mineria, which puts the gold production of the country in 1895 at 1000 kgms., these figures being based on the exports and the Mint returns. There was a slight gain in 1896, bringing the total up to 1050 kgms. At the present time there is a good deal of activity in mining in Chile and new fields are being opened up. Several promising mines have been located in the Araucanian desert. The same authority estimates the production of silver in 1895 at 150,000 kgms., and in 1896 at 151,500 kgms.

Ecuador.—No gain in gold or silver is to be noted. The Playa de Oro Company, from which so much had been expected by the uninitiated, continues to show a yield of less than 8c. per cu. yd.

The Guianas.—The gold production of the Guianas in 1896 showed an increase over the preceding year in the British and French colonies and a slight decrease in Dutch Guiana. From the reports furnished us by Mr. W. Van Esveld, of Surinam, which are based upon the official records, we find the totals for two years past to have been as follows:

	1895.		1896.		Changes.	
	Ozs.	Kgms.	Ozs.	Kgms.	Ozs.	Kgms.
British Guiana	122,936	3,824	123,759	3,849	Inc. 823	25
Dutch Guiana	28,228	878	26,878	836	Dec. 1,350	42
French Guiana	90,261	2,807	101,938	3,171	Inc. 11,677	364
Totals	241,425	7,509	252,575	7,856	Inc. 11,150	347

Nearly all the gold produced in the Guianas is still from placer workings on the rivers, though in British Guiana a single company, the Barima, has been for some time exploiting a quartz mine, and has had its mill at work for several months. The production from this source is not yet large enough to make any considerable difference in the output. In Dutch Guiana two or three new companies have lately begun work on the placers on a considerable scale. Of the production of French Guiana in 1896, 1831 kgms., or 57.8% of the total, came from the territory which is in dispute between France and Brazil. Several French companies are now at work there. It was recently announced that the title to the disputed territory is to be settled by arbitration.

Peru.—Estimates based on the Mint reports of this country show a considerable increase in gold production, the total in 1896 being 176 kgms., a gain of 55.7% over 1895. The same authority indicates a decrease in the production of silver of about 8.5%, the estimate for 1896 being 105,181 kgms.

EUROPE.

The gold and silver industry of Europe experienced no material change in 1896. Reference to the gold and silver mines of Italy and Norway is made under those captions, respectively, in a subsequent part of this volume. The production of silver in Spain is mentioned in the article "Lead in Spain."

France.—A company has been formed to work placers in the Ariege.

Germany.—The returns of gold production in Germany, as furnished by the Imperial Statistical Department, have recently been corrected and the amount considerably reduced, probably by excluding the bullion obtained in German works from foreign ores, although this is not clearly stated in the report. The production as officially corrected has been as follows:

Year.	Kgms.	Value.	Year.	Kgms.	Value.	Year.	Kgms.	Value.	Year.	Kgms.	Value.
1890....	2,281	$1,586,500	1892....	2,553	$1,776,250	1894....	3,204	$2,232,750	1896....	2,487	$1,652,680
1891....	2,432	1,693,000	1893....	2,551	1,774,150	1895....	3,547	2,357,144			

Russia.—For the gold production of Russia we have the figures reported by the Imperial Mint. In theory these figures should represent the entire production, since the law requires all the gold produced to be deposited with the mint or one of its branches. In practice this law is evaded to a certain extent, varying with locality. In the placers of the Trans-Baikal, especially in those near the Manchurian-Chinese frontier, the amount of gold stolen by miners or withheld by the finders is considered by the best authorities to be fully 20% of the total. In the districts which present fewer facilities to illicit gold buyers the proportion is lower, but an allowance of 10% on all gold deposited is a conservative one and is probably below the fact.

The total amount of gold returned to the Imperial Mint as won during the year 1896 was 2589¼ pouds. Reducing from Russian measure and adding 10% the output in 1896 is reckoned at 46,653 kgms., equal to 1,499,897 ozs., or $31,002,870. The production, with the same addition of 10% for unreported gold, for six years past has been as follows:

Year.	Kgms.	Value.	Year.	Kgms.	Value.	Year.	Kgms.	Value.
1891.....	39,183	$26,041,000	1893.....	35,619	$23,672,400	1895.....	47,725	$31,787,500
1892.....	39,161	26,026,400	1894.....	39,913	26,525,980	1896.....	46,653	31,002,870

The decrease in 1896 was due chiefly to the closing down of one of the great Siberian placers, which in 1895 yielded some 700 pouds of gold, but was hardly worked at all in 1896. No material progress in vein-mining is yet reported. In all the Trans-Baikal and the Amour region—which is just now the most productive—there is only one mill (20 stamps), and this mill, at the mine of Baian-Zourga, was not in use, from recent accounts.

The center of greatest activity in Siberian gold mining at present is on the upper tributaries of the Amour—the Onon, the Amgoun and the Bielaya—but more or less work has been done down the great river as far as Khabarovsk and the Oussouri, while the rich grounds in the upper valleys of the Lena have been opened up to some extent.

The limitations of Russian production are found in the crude methods employed and the scarcity of labor. At present only the richer placers are worked, and those in a desultory and imperfect fashion. A further limitation is found in the rigorous climate, which restricts work to a few months in each year. In the Trans-Baikal the alluvial ground is perpetually frozen, the summer heats thawing it out only a few feet below the surface; beyond that depth the ground must be mined like rock.

The rapid extension of the Siberian Railroad which is now in progress, and the settlement which follows the railroad, will permit the introduction of machinery and furnish more and better labor; while the extension of Russian influence over Northern Manchuria, in which all the southern affluents of the Amour rise, will largely extend the area open to working. Production cannot increase rapidly, however, until better methods are introduced.

The silver production of Russia varies but little. By the returns of the Imperial Mint it was 8578 kgms. in 1894, rose to 12,108 kgms. in 1895, and dropped to 10,455 kgms. in 1896. These returns are close to the actual production, as there is not the same opportunity nor the same inducement for concealment as in the case of gold. The greater part of the silver also is produced in works owned by the government. A little comes from Alagyr in the Caucasus, but the greater part is from Siberia, where it is obtained in the Nertchinsk, Verkhne-Oudinsk and Semipalatinsk districts. Other deposits are known to exist, but the silver-lead mines of the Altai will not be opened extensively until fuel and supplies can be carried to them by the railroad.

ASIA.

The chief production of gold in Asia is from Siberia, but the output of that country is given with that of Russia.

China.—The production of this country is more or less a matter of conjecture. The estimates are based on a comparison of the exports and imports officially reported; but these reports do not give the correct amount, since a large amount of gold is carried clandestinely into China by natives returning from other countries. Gold is mined in China, but no one knows how much.

India.—The greater part of the production of this country is from the Colar gold field in Mysore. In 1896 its total production reached $5,821,600, and in six years has shown a gain of 147%. In 1896 the gain was 29% as compared with 1895. The output of this field for six years past has been as follows:

Year.	Crude Ozs.	Fine Ozs.	Value.	Year.	Crude Ozs.	Fine Ozs.	Value.
1891.....	130,137	113,870	$2,353,693	1894.....	209,729	183,513	$3,793,214
1892.....	163,140	142,748	2,950,601	1895.....	249,355	218,186	4,509,905
1893.....	207,135	181,243	3,746,293	1896.....	321,878	281,643	5,821,561

Operations in the district have been chiefly in the hands of four companies, which have furnished 95% of the entire production. Three other companies, the Balaghat-Mysore, the Gold Fields of Mysore, and the Mysore West & Wynaad, have operated with less success, owing to faults in location and management. In 1896 a new company, the Coromandel, made its first appearance as a producer, and its property is said to promise well. The reports of the four leading companies for 1896 show the following results:

Company.	Ore Worked.	From Mill.	From Tailings.	Total Gold.	Per Ton.
	Tons.	Crude Ozs.	Crude Ozs.	Crude Ozs.	Ozs.
Mysore	64,297	89,053	18,728	107,731	1.68
Champion Reef	70,972	78,796	7,799	86,595	1.22
Ooregum	63,788	55,162	10,513	65,675	1.03
Nundydroog	39,490	43,299	1,626	44,925	1.14

From this statement it appears that the return varied from $18.63 per ton in the Ooregum to $30.38 in the Mysore. The largest mill in operation is the Champion Reef, which has 140 stamps, 40 having been added in July, 1896.

Outside of the Colar field the production of India is not large. A little is obtained from the east coast from placer workings, and some from the mines of Burma, where several companies are operating. A new company has lately begun work at the Wondalli field in the Deccan, which is said to show promise, but has not yet made any returns.

Japan.—This country has a small and fairly steady production of gold and silver, chiefly won in connection with lead and copper. It is understood that the Japanese government is making arrangements to work the gold fields of Formosa, which yielded under Chinese rule a varying amount, never ascertained.

Korea.—The gold production is ascertained approximately by comparison of exports and imports. No results have been reported from the concessions granted to foreigners in 1894 and 1895; probably nothing has been done owing to the unsettled state of the country.

The Malay Peninsula.—A small quantity of gold is obtained from the mines of the Straits and the adjoining native states. Several companies are operating in the Peninsula, the chief mines being those of the Raub A'lian Company and the Punjon Mining Company. The production, including some gold obtained by Chinese miners, is estimated at 25,000 ozs. in 1896.

Other Countries.—The existence of gold in Tonkin has been ascertained, but very little has actually been obtained. A small amount also is said to be obtained in Afghanistan and Tibet.

AFRICA.

Transvaal.—The political agitation which began about the close of 1895 continued all through the year 1896; and the "boom" period having passed, the year 1896 was consequently a period of transition, of enforced economy, and of consolidation. Some mines were compelled to close down, and others must follow, unless the cost of working can be reduced.

The returns made to the Witwatersrand Chamber of Mines, reduced to fine ounces, at the valuation given, which would make the gold average .810 fine in 1896, against .816 fine in 1895, are as follows:

Districts.	1895.		1896.		Changes.
	Crude Ozs.	Fine Ozs.	Crude Ozs.	Fine Ozs.	Fine Ozs.
De Kaap	63,046	51,446	121,390	98,326	Inc. 46,880
Lydenberg	63,506	51,821	50,387	40,813	Dec. 11,008
Klerksdorp	90,841	74,126	38,818	31,443	Dec. 42,683
Zoutpansberg	8,726	7,120	5,602	4,538	Dec. 2,582
Others	1,653	1,349	857	694	Dec. 655
Total outside districts	227,772	185,862	217,054	175,814	Dec. 10,048
Witwatersrand	2,277,635	1,858,550	2,280,892	1,847,523	Dec. 11,027
Total Transvaal	2,505,407	2,044,412	2,497,946	2,023,337	Dec. 21,075
Total values	.....	$42,257,996	.....	$41,822,376	Dec. $435,620

The total decrease in 1896 was, therefore, 0.8%, made up by a loss of 10,048 ozs., or 4.4%, in the outside districts, and one of 11,027 fine ozs., or 0.6%, in the Witwatersrand. The crude ounces reported for the last-named district showed a slight increase, 3257, but owing to the decrease in values this was converted into a loss in fine ounces. The lower value resulted from the greater proportion obtained from the tailings, the cyanide bullion being less fine than that obtained from the retorts. The most notable feature of the year was a decrease in the average tenor of the ore worked on the Witwatersrand, the average yield per ton in 1896 having been only 0.56 oz. The estimated number of men at work in the district at the close of the year was 7430 whites and 70,000 natives.

The following table, giving the value of Witwatersrand ore milled from 1890 to 1896, and the accompanying tables on pages 254, 255, and 256 were compiled for THE MINERAL INDUSTRY by Mr. H. Y. Campbell, of Johannesburg. From these statistics it appears that the 48 companies paid in dividends during the seven years, 1890–96, £7,783,198, which was a trifle more than 21.5% of the total output of the Witwatersrand during that period. In studying these figures, however, it must be borne in mind that this list includes all of the successful concerns, and none of the unsuccessful. That is to say it is a list of the dividend payers only. A vast amount of money has been spent at Johannesburg in fruitless explorations, and in enterprises which have returned only a part of the outlay in them, and a statement which records profits without taking into account the losses is, of course, misleading. The proportion of dividends paid by the Witwatersrand mines to gross output is nevertheless remarkably high.

VALUE OF WITWATERSRAND ORE MILLED FROM 1890 TO 1896.

Year.	Tons. (c)		Range of Value Per Ton Milled.		Range of Value Per Ton Cyanided.		Range of Total Value Per Ton Ore. (d)		Cash Value Won.
	Milled.	Sands Cyanided	Highest.	Lowest.	Highest.	Lowest.	Highest.	Lowest.	
			s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	£ s. d.
1890	711,877	.....	a 46 8	.....	.....	.....	a 46 8	.....	1,661,046 6 8
1891	1,154,145	.....	a 40 6	.....	.....	.....	a 40 6	.....	2,337,143 12 6
1892	1,979,354	.....	a 35 3	.....	.....	.....	a 44 2	.....	4,371,589 11 4
1893	2,200,704	b620,348	37 1	32 11	b 16 6	b 13 7	49 10	43 9	5,116,423 16 4
1894	2,826,330	2,674,673	35 3	31 9	14 3	12 11	51 3	47 5	6,974,181 5 1
1895	3,456,575	2,759,066	33 8	28 2	14 8	12 2	49 0	40 4	7,859,191 8 1
1896	4,011,695	2,807,963	28 7	25 10	15 8	13 11	41 0	38 0	7,842,446 18 4
Totals	16,340,676	8,862,050	.....	.....	.....	.....	.....	.....	36,192,022 17 4

(a) Average. (b) The first report of amount of sands cyanided is for the month of August, when 107,586 tons were so treated. (c) The ton of ore is supposed to be 2000 lbs., but in practice is probably not more than 1750 lbs.; the ton of sand varies from 24 to 30 cubic feet. (d) Includes value of gold won from concentrates.

DIVIDENDS PAID BY WITWATERSRAND

Number.	Name of Company.	1886–89.		1890.		1891.		1892.	
		Amount Paid.	Rate. %	Amount Paid.	Rate. %	Amount Paid.	Rate. %	Amount Paid.	Rate. %
1	Aurora	£2,500	5	.....	.....	.....	.....	£3,250	5
2	African Gold Rec	.....	.....	.....	.....	.....	.....	17,400	10
3	Champ d'Or	.....	.....	.....	.....	£8,010	10	.....	.....
4	City & Suburban	34,911	55	£58,750	75	42,500	50	.....	.....
5	Crown Reef	24,000	24	15,900	15	53,000	50	63,000	55
6	Durban-Roodepoort	18,000	20	30,000	30	30,000	30	43,333	41
7	Evelyn	2,600	10	.....	.....	.....	.....	.....	.....
8	Ferreira	.....	.....	.....	.....	22,000	50	56,250	125
9	Geldenhuis Estate	4,000	5	.....	.....	18,000	15	17,500	10
10	Geldenhuis Main Reef	.....	.....	.....	.....	.....	.....	.....	.....
11	Glencairn Main Reef	.....	.....	.....	.....	.....	.....	.....	.....
12	Grahamstown	1,650	10	.....	.....	.....	.....	.....	.....
13	Golden Kopje	5,625	7½	.....	.....	.....	.....	.....	.....
14	Henry Nourse	.....	.....	.....	.....	.....	.....	.....	.....
15	Johannesburg Pioneer	.....	.....	2,100	10	7,350	35	10,500	50
16	Jubilee	17,575	95	14,300	55	9,100	35	17,502	60
17	Jumpers	11,500	25	.....	.....	.....	.....	.....	.....
18	Langlaagte Block "B"	.....	.....	.....	.....	.....	.....	.....	.....
19	Langlaagte Estate	129,630	29	67,050	15	44,700	10	85,400	20
20	Langlaagte Royal	.....	.....	.....	.....	2,500	5	19,500	35
21	May Consolidated	.....	.....	.....	.....	.....	.....	.....	.....
22	Meyer & Charlton	8,300	20	12,648	20	31,620	50	29,350	45
23	Meyer & Leeb	.....	.....	.....	.....	.....	.....	1,825	12½
24	Moss Rose	.....	.....	7,400	20	.....	.....	.....	.....
25	Moss Rose Extension	4,000	4	.....	.....	.....	.....	.....	.....
26	New Aurora West	.....	.....	.....	.....	.....	.....	6,000	10
27	New Chimes	4,000	10	.....	.....	.....	.....	20,048	40
28	New Crœsus	.....	.....	.....	.....	.....	.....	7,500	10
29	New Kleinfontein	.....	.....	.....	.....	.....	.....	.....	.....
30	New Heriot	8,995	15	.....	.....	.....	.....	.....	.....
31	New Primrose	.....	.....	.....	.....	26,044	15	47,731	27½
32	Nigel	.....	.....	.....	.....	.....	.....	75,989	47½
33	Orion	.....	.....	.....	.....	.....	.....	1,725	7½
34	Paarl Ophir	.....	.....	.....	.....	.....	.....	.....	.....
35	Paarl Central	18,750	25	.....	.....	.....	.....	.....	.....
36	Porges Randfontein	.....	.....	.....	.....	.....	.....	.....	.....
37	Rand Central Ore Reduction	.....	.....	.....	.....	.....	.....	.....	.....
38	Rietfontein	.....	.....	.....	.....	.....	.....	.....	.....
39	Robinson	135,937	5	.....	.....	108,750	4	190,312	7
40	Royal Treasury	3,500	17½	.....	.....	.....	.....	5,250	35
41	Salisbury	43,175	200	.....	.....	.....	.....	27,900	30
42	Simmer & Jack	7,500	10	.....	.....	24,900	30	33,400	40
43	Stanhope	13,550	42½	.....	.....	23,800	70	8,500	25
44	Van Ryn Mines	.....	.....	.....	.....	.....	.....	.....	.....
45	United Main Reef-Roodepoort	.....	.....	.....	.....	.....	.....	.....	.....
46	Wemmer	17,051	87	.....	.....	4,630	10	.....	.....
47	Worcester	13,580	75	13,108	15	9,071	10	15,855	17½
48	Wolhuter	.....	.....	.....	.....	.....	.....	.....	.....
	Total dividends	530,329	.....	221,256	.....	465,975	.....	804,990	.....

The large gain showed in the De Kaap district in 1896 came chiefly from the Sheba mine. The gain was partly offset by a decrease in the Klerksdorp district.

Other South African Countries.—A small quantity of gold is obtained in Natal, but the amount is not increasing. The gold production of the British South Africa Company's territory has not come up to expectations. The *Rhodesian Times,* which has every reason to make the best possible showing, gives the total production from 1890, when the country was first opened, to the end of 1896 at 7047 crude ozs. only, or probably about 6000 fine ozs. The returns of ore mined and crushed are not complete, but the average result seems to have been not far from 0.67 oz. per ton. The difficulties with the natives, which have occurred from time to time, notably the Matabele insurrection in 1896, interfered very much with work. The chief difficulty in Rhodesia is in transportation, which makes supplies and machinery very costly. A railroad from the coast has been under construction for several years, but its progress has been extremely slow and there is still a long interval between its terminus and any of the towns or trading posts

MINING COMPANIES, 1886 TO 1896.

1893.		1894.		1895.		1896.		1886-96.	Issued Capital Dec. 31, 1896.	Quotation of Shares Dec. 31, 1896.	Market Value on Issued Capital Dec. 31, 1896	Number.
Amount Paid.	Rate. %	Amount Paid.	Rate. %	Amount Paid.	Rate. %	Amount Paid.	Rate. %	Total Dividends				
.....	.....	.....	.....	.....	.....	.....	.....	£5,750	£65,000	£7/8	£56,777⅜	1
£35,000	20	£17,500	10	£17,500	10	.....	.....	87,400	200,000	£3/4	150,000	2
.....	.....	26,500	20	.....	.....	.....	.....	34,510	131,624	£1	131,624	3
65,000	100	21,500	25	63,750	75	£68,000	5	374,411	a1,360,000	£4⅝	1,488,000	4
60,000	50	60,000	50	90,000	75	132,000	110	497,900	120,000	£10⅜	1,245,000	5
68,750	55	75,000	60	75,000	60	56,750	45	398,833	125,000	£6½	812,500	6
.....	.....	.....	.....	.....	.....	.....	.....	2,600	.....	.....	.....	7
45,000	100	89,500	150	175,500	195	247,500	275	635,750	90,000	£18¾	1,687,500	8
43,750	25	57,500	30	60,000	30	25,000	12½	225,750	200,000	£3¼	650,000	9
.....	.....	.....	.....	30,000	20	.....	.....	30,000	150,000	16s. 6d.	123,750	10
.....	.....	28,125	12½	61,875	27½	56,250	25	90,000	225,000	£2 7/16	548,431¼	11
.....	.....	.....	.....	.....	.....	.....	.....	1,650	.....	.....	.....	12
.....	.....	.....	.....	.....	.....	.....	.....	5,625	.....	.....	.....	13
.....	.....	.....	.....	.....	.....	37,500	30	37,500	125,000	£6¾	843,750	14
7,875	37½	.....	.....	.....	.....	84,000	400	111,825	21,000	£8½	183,500	15
36,000	120	36,000	120	45,000	90	30,000	60	205,477	50,000	£7¼	362,500	16
25,000	25	55,000	55	55,000	55	30,000	30	176,500	100,000	£4¼	425,000	17
.....	.....	5,693	8	.....	.....	.....	.....	5,693	617,500	£1 1/16	649,843¾	18
140,100	30	211,500	45	235,000	50	141,000	30	1,054,380	470,000	£4⅜	2,056,000	19
7,000	5	.....	.....	.....	.....	.....	.....	29,000	180,000	£11 16s.	123,750	20
.....	.....	.....	.....	54,000	20	55,000	20	109,000	265,500	£2½	594,750	21
43,012	60	42,896	55	42,039	50	17,000	20	216,806	83,200	£5½	458,600	22
2,225	15	.....	.....	.....	.....	.....	.....	4,050	15,000	.....	.....	23
.....	.....	.....	.....	.....	.....	.....	.....	7,400	.....	.....	.....	24
.....	.....	.....	.....	.....	.....	.....	.....	4,000	.....	(c)	.....	25
4,000	5	.....	.....	.....	.....	.....	.....	10,000	100,000	£7/8	87,500	26
.....	.....	25,000	25	55,000	55	5,000	5	109,048	100,000	£11 16s.	68,750	27
.....	.....	.....	.....	.....	.....	.....	.....	7,500	500,000	£13 16s.	406,250	28
.....	.....	13,750	12½	13,750	12½	.....	.....	27,500	185,000	£2 5/16	427,812½	29
8,500	10	34,000	40	123,432	125	95,084	85	270,011	111,864	£7 7/8	880,929	30
78,567	40	112,000	40	140,000	50	.....	.....	404,342	300,000	£4⅛	1,237,500	31
80,000	50	80,000	50	32,000	20	.....	.....	267,989	200,000	£2 5/16	462,500	32
.....	.....	31,500	105	32,000	20	.....	.....	65,225	160,000	£5/8	100,000	33
.....	.....	.....	.....	.....	.....	3,000	25	3,000	12,000	.....	.....	34
.....	.....	.....	.....	.....	.....	.....	.....	18,750	400,000	£9 10s.	360,000	35
.....	.....	.....	.....	43,750	10	.....	.....	43,750	437,500	£1 5/16	574,218¾	36
.....	.....	18,500	25	36,000	30	14,875	10	69,375	149,749	£1	149,749	37
40,000	25	.....	.....	.....	.....	.....	.....	40,000	270,000	£2	540,000	38
217,500	8	275,000	10	385,000	14	330,000	12	1,642,499	b2,750,000	£8½	4,675,000	39
6,900	20	8,000	20	.....	.....	.....	.....	23,650	a540,000	£17/8	252,000	40
9,300	10	.....	.....	.....	.....	.....	.....	80,375	100,000	£2 7/8	287,500	41
34,000	40	17,000	20	47,160	20	.....	.....	163,960	b4,700,000	£4 7/8	4,582,000	42
17,000	50	34,000	100	13,600	40	1,700	5	112,150	34,000	£¾	25,500	43
.....	.....	.....	.....	.....	.....	32,000	20	321,000	176,886	£4 13/16	718,599⅜	44
.....	.....	26,000	20	70,000	50	15,000	10	133,500	135,000	£4	540,000	45
.....	.....	.....	.....	.....	.....	22,660	15	179,181	80,000	£7⅜	590,000	46
13,609	15	63,500	70	110,000	200	47,500	75	235,910	93,722	£4⅛	386,603	47
.....	.....	13,000	10	55,637	60	51,550	55	13,000	a86,000	£5⅛	1,116,185	48
1,108,088	.....	1,477,937	.....	2,161,993	.....	1,541,959	.....	8,313,527	16,504,545	.....	30,241,372	..

(a) £4 shares. (b) £5 shares. (c) *Vide* New Primrose.

in Mashonaland and Matabeleland, while there is a still greater distance to be traversed to reach the present terminus of the railroad running northward from the Cape of Good Hope. The activity in prospecting and locating claims was very great for a year or two, but recently very little has been done.

West and Central Africa.—There is a small production from the Gold Coast where one English company, the Wassau Gold Mining Company, has a plant at work, and gold is also obtained by trade with the natives. Some gold is obtained from Central Africa and the Soudan in the same way.

The French authorities in Madagascar, having completed the conquest of that island, have begun to explore its resources. Gold has been obtained in small quantities by the natives in Betsileo, Fianarantsoa and other districts.

The gold production of Western and Central Africa as measured by the amount imported into England from the West Coast since 1888 has been as follows: 1889, $844,262; 1890, $705,705; 1891, $856,730; 1892, $1,011,924; 1893, $649,695; 1894, $574,653; 1895, $661,630.

YIELD PER TON OF ORE MILLED BY WITWATERSRAND MINING COMPANIES.

Company.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	Highest for Period.	Lowest for Period.
	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.	s. d.
Aurora	31 9	31 2	24 9	20 9	21 6	35 7	*Nil.*	35 7	20 9
Banket	13 1	15 0	13 6	25 1	37 6	41 11	*Nil.*	41 11	13 1
Bonanza	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	100 11	100 11	100 11
Champ d'Or	*Nil.*	84 1	46 9	28 8	63 6	53 6	33 8	84 1	28 8
Champ d'Or Deep	*Nil.*	*Nil.*	*Nil.*	*Nil.*	24 6	28 2	*Nil.*	28 2	24 6
City & Suburban	69 1	27 6	45 8	65 7	47 1	39 7	37 7	69 1	27 6
Crown Reef	62 3	40 6	51 11	45 4	43 0	40 11	44 11	62 3	40 6
Durban-Roodepoort	45 8	40 5	37 5	52 9	53 0	46 2	41 7	53 0	37 5
Ferreira	67 0	69 2	69 2	81 0	93 7	84 4	74 3	93 7	59 2
Geldenhuis Deep	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	18 0	26 10	26 10	18 0
Geldenhuis Estate	*Nil.*	73 6	44 2	41 7	35 2	32 6	27 0	73 6	27 0
Geldenhuis Main Reef	61 4	34 10	24 9	27 7	34 4	40 11	35 4	61 4	24 9
George Goch	45 8	29 8	27 2	25 7	30 6	34 7	28 6	45 8	25 7
Glencairn	24 4	20 4	36 4	29 6	48 10	39 7	26 10	48 10	20 4
Ginsberg	*Nil.*	*Nil.*	*Nil.*	*Nil.*	38 8	52 10	45 3	52 10	38 8
Henry Nourse	42 0	50 2	52 11	61 6	79 11	62 8	57 3	79 11	42 0
Johannesburg Pioneer	70 6	57 6	59 11	40 8	39 4	55 10	73 9	73 9	39 4
Jumpers	29 9	26 0	31 6	31 3	40 10	42 4	33 4	42 4	26 0
Jubilee	79 4	54 2	67 1	47 0	48 0	35 10	32 1	79 4	32 1
Langlaagte Estate	56 11	50 7	34 0	31 11	37 6	38 5	30 1	56 11	30 1
Langlaagte Block "B"	*Nil.*	16 3	19 0	28 6	31 11	26 2	23 0	31 11	16 3
Langlaagte Royal	69 11	34 10	30 7	29 8	40 0	23 0	21 1	69 11	21 1
Langlaagte Star	21 6	19 8	17 2	16 9	*Nil.*	*Nil.*	20 8	21 6	16 9
Langlaagte United	.....	13 5	27 8	22 3	22 8	27 11	*Nil.*	27 4	13 5
Lancaster	25 1	*Nil.*	*Nil.*	*Nil.*	*Nil.*	35 2	*Nil.*	35 2	25 1
May Consolidated	48 1	28 8	31 6	32 11	37 0	38 8	30 0	48 1	28 8
Minerva Block Reef	*Nil.*	*Nil.*	*Nil.*	16 6	19 10	32 7	21 6	32 7	16 6
Meyer & Charlton	50 11	80 3	68 3	70 9	53 3	41 5	33 1	80 3	33 1
Metropolitan	28 4	18 0	18 5	15 10	28 0	31 3	28 6	28 6	15 10
New Heriot	35 9	16 3	48 3	71 7	56 6	52 3	51 6	71 7	16 3
New Chimes	66 10	51 3	44 10	44 7	41 10	42 4	26 7	66 10	26 7
New Midas	*Nil.*	*Nil.*	14 0	*Nil.*	*Nil.*	122 4	39 0	122 4	14 0
New Primrose	42 2	35 3	39 4	40 6	39 10	34 7	28 7	42 2	28 7
New Rietfontein	118 8	*Nil.*	79 10	104 0	48 11	44 8	26 6	104 0	36 6
New Kleinfontein	19 6	15 6	*Nil.*	34 11	37 1	32 6	.....	37 1	15 6
New Modderfontein	84 2	*Nil.*	52 3	43 6	49 4	*Nil.*	23 9	84 2	23 9
Nigel	143 0	141 9	193 5	142 0	137 8	93 9	59 0	193 5	59 0
New Comet	*Nil.*	*Nil.*	*Nil.*	*Nil.*	46 9	34 7	29 2	46 9	29 2
New Crœsus	62 3	35 9	31 2	31 7	*Nil.*	26 6	25 4	62 3	25 4
New Heidelberg Roodepoort	*Nil.*	*Nil.*	*Nil.*	30 0	*Nil.*	*Nil.*	16 6	30 0	16 6
Orion	*Nil.*	35 6	33 4	32 8	40 9	40 10	26 0	40 10	26 0
Princess	12 10	17 6	15 5	21 11	40 2	48 4	43 5	48 4	16 5
Paarl Central	*Nil.*	11 6	22 5	18 0	34 1	36 0	*Nil.*	36 0	11 6
Randfontein Porges	*Nil.*	*Nil.*	28 4	38 10	36 4	42 8	33 9	42 8	28 4
Robinson	113 0	108 11	96 2	104 0	96 10	80 6	76 2	113 0	76 2
Roodepoort (Kimberley)	*Nil.*	12 0	13 0	16 2	24 3	61 53	27 6	61 53	12 0
Roodepoort Deep	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	32 6	32 6	32 6
Salisbury	83 7	85 10	88 9	67 8	54 2	39 6	28 0	88 9	28 0
Simmer & Jack	38 4	31 7	28 8	27 8	38 0	46 8	42 8	46 8	27 8
Stanhope	60 0	43 4	50 6	43 8	52 7	39 9	32 1	60 0	32 0
Treasury	*Nil.*	48 1	47 4	65 10	38 5	*Nil.*	26 7	65 10	26 7
United Main Reef	30 7	20 0	25 2	27 10	50 7	48 2	37 10	50 7	20 0
Van Ryn	40 2	*Nil.*	28 3	24 7	35 7	46 1	30 7	46 1	24 7
Village Main Reef	*Nil.*	*Nil.*	56 10	49 5	53 10	111 8	*Nil.*	111 8	49 5
Wemmer	55 4	74 2	.....	66 3	54 3	64 2	51 3	74 2	51 3
Witwatersrand	18 5	15 9	.....	.....	.....	.....	31 2	31 2	15 9
Wolhuter	20 5	*Nil.*	31 6	34 10	51 5	39 10	33 7	51 5	20 5
Worcester	60 2	54 9	46 0	51 2	82 11	57 0	37 9	60 2	46 0
Vogelstruis	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	20 1	20 1	20 1

AUSTRALASIA.

Australasia made in 1896 a production of gold slightly exceeding that of the previous year. The figures for each colony are given in the following table:

	Crude Ozs.	Fine Ozs.	Value.	Per Cent.
New South Wales	296,072	259,063	$5,354,832	12.3
New Zealand	263,722	237,350	4,906,025	11.2
Queensland	638,212	526,525	10,883,272	24.9
South Australia	29,004	27,384	566,027	1.3
Tasmania	62,586	55,258	1,142,295	2.6
Victoria	805,089	756,782	15,642,684	35.8
Western Australia	281,265	251,646	5,201,518	11.9
Totals	2,375,950	2,114,008	$43,696,653	100.0
Totals, 1895	2,350,567	2,070,335	42,793,824	.....

The total increase, as compared with 1895, was 43,675 fine ozs., or 2.1%. Nearly all the colonies showed an unexpected falling off in the last quarter of 1896, partly the result of dry weather and scarcity of water. Australasia led the Transvaal in 1896 by a total of $1,894,277, and therefore remains the second gold producer in the world. The production of the quartz or vein mines in all the older colonies is very steadily kept up, but the production from placers, which increased largely in 1895, showed a considerable falling off in 1896.

New South Wales.—The production in 1896 of 296,072 crude ozs., equivalent to 259,063 fine ozs., or $5,354,832, as compared with 1895, shows a decrease of 56,081 fine ozs., or $1,159,194. This reduction of 17.6% was due chiefly to a falling off in placer mining and the return of many men who engaged in that work for a time to other occupations which might prove more remunerative. The chief losses were in the Bathurst and Adelong districts; the Colar fields showed a considerable gain, and some other districts also increased their production.

This colony furnishes nearly all of the silver output of Australasia. This showed a decrease, owing chiefly to the lower grade of the Broken Hill ore.

New Zealand.—The gold production of New Zealand in 1896 was 263,722 crude ozs., which was, at the values reported, equivalent to 237,350 fine ozs., or $4,906,025. In 1895 it was 293,491 crude ozs., equal to 264,142 fine ozs., or $5,459,815. The decrease was, therefore, 26,792 fine ozs., or 10.2%. There was a large increase, however, in the number of prospecting and mining licenses issued by the Mines Department during the year.

Two New Zealand mines distinguished themselves in 1896. The Waihi Company obtained $609,330 from 32,850 tons of ore, an average of $18.55 per ton, and paid dividends of 40% on its capital stock of $750,000. The Hauraki Company worked 4193 tons of ore, which yielded $473,465, or $112.92 per ton. The company paid 160% in dividends on $200,000 capital stock.

Queensland.—The production in 1896 was 638,212 crude ozs., equivalent to 526,525 fine ozs., or $10,883,272. The increase over 1895 was 15,212 crude ozs., or 2.4%. The Charters Towers district continues to lead the others, its yield forming about 40% of the total. It is noteworthy that in 1896 about 17.5% of the Charters Towers output was derived from reworking tailings. The second district in importance was Crocodile Creek, in which the Mount Morgan mine is included, which supplied about 24% of the total. The Queensland production is chiefly from quartz mining, the proportion from placers being small. The increase was chiefly due to more active working of the older mines.

South Australia.—This colony produced in 1896 29,004 crude ozs., equivalent to 27,384 fine ozs., or $566,027, a decrease of 16,408 fine ozs., or $339,154, compared with 1895. This was the result of the superior attractions of the West Australian gold fields, which have drawn off many miners, and left the South Australian districts in some cases almost deserted. The colony is of great extent, but has only a small population as yet, and much development is needed. Some work has been done in the northern territory, which belongs to this colony, but the results have not so far been important.

Tasmania.—The output of 62,586 crude ozs., equal to 55,285 fine ozs., or $1,142,295, compared with the year 1895, shows an increase of 6340 fine ozs., or $131,160, a gain of 13%. This was due chiefly to increased activity in the quartz mines, though the Mt. Lyell copper mine contributed, as its ore carries gold.

Victoria.—The production of this colony in 1896 was 805,089 crude ozs., equal at the usual rate, to 756,782 fine ozs., or $15,642,684. In 1895 the output was 740,086 crude ozs., equal to 695,681 fine ozs., or $14,379,726. The gain in 1896 was therefore 61,101 fine ozs., an advance of 8.8% over the previous year. Victoria thus remains far ahead of the other colonies, furnishing over one-third of the total gold of Australasia. Bendigo remained the most important district in Victoria, producing 191,940 crude ozs. in 1896, while Ballarat was second with 160,317 ozs. Gippsland furnished 116,599 ozs., and the Beechworth mines 109,865 ozs. The total dividends for 1896, as reported to the Mines Department, were $2,602,225.

Western Australia.—The production, reported in 1896 at 281,263 crude ozs., was equivalent to 251,646 fine ozs., or $5,201,518. The values given in the official reports show an improvement in the bullion, the average in 1896 having been .895 fine, against .815 fine in 1895. The increase in output, as compared with 1895, was 62,963 fine ozs. Under ordinary circumstances this would be a remarkable growth for one year, amounting to 33%; but it is nevertheless a disappointing result in view of the enormous amount of money invested there during the past three years. On a rough calculation between $275,000,000 and $300,000,000 of English capital has been put into West Australian gold mines, nearly $200,000,000 having been paid for the mining claims themselves—either in cash or stock—and the remainder subscribed for machinery and working capital. Even if the gold reported were all net profit the return would be a very insufficient one on the investment. In reality a few companies made money, but the great majority have so far nothing to show.

THE INDIAN ARCHIPELAGO.

Borneo.—In British North Borneo a large deposit of low-grade ores has been located in the district of Sarawak by an English corporation known as the Borneo Company. Enough development work has been done to warrant the erection of a mill with a capacity of 300 tons daily, and work on this plant is now in progress. The actual production so far has been small. Native miners obtain a small amount of gold in the districts of Miri, Marup, and Upper Sadong.

Celebes.—Mr. S Atherton, mining engineer for the Netherlands India Mining Company, writes as follows: "Two companies only are now producing gold in this island, though the whole of North Celebes—volcanoes included—is covered by applications for gold-mining concessions. The Netherlands India Mining Company shipped from its mines at Paleleh in 1896 110 long tons of ore to London, the returns from which showed an average value of 5.78 ozs. gold and 11 ozs. silver per ton. In the first two months of 1897 about 200 tons of ore were shipped. The company is arranging to put up a reduction plant, and until that is completed will continue to ship its high-grade ores. The second company has opened mines at Simulata, and in 1896 shipped 50 tons of ore, which returned 2.33 ozs. gold per ton. Natives average about 75 ozs. gold a year.

Phillippine Islands.—In addition to a small amount from native sources, two English companies have been at work for two or three years past, and report low-grade deposits of considerable extent. During 1896 their operations were interrupted by the insurrection in the islands.

THE COMMERCIAL MOVEMENT OF THE PRECIOUS METALS.

The price of silver showed only small fluctuations during the year, though in the last half a downward tendency became apparent, resulting from the defeat of the silver party in November, a lessened demand from India and China, and the certainty that the output of silver for the year would reach large proportions.

The Bank of England stock of gold toward the close of the year was decreased by heavy shipments to the United States and the same affected, though in a less degree, the Bank of France also. The Austro-Hungarian Bank continued to increase its holdings nominally in preparation for a currency change. The most remarkable accumulation is in the Bank of Russia, which increased no less than $93,610,000 during 1896; the bank also holds about $110,000,000 in gold belonging to the Imperial Treasury, bringing its total up to $590,000,000. It may be noted that Russia in 1896 absorbed about 43% of the total gold production of the world for the year. Shipments of silver from London to the East were as follows:

Year.	India.	China.	The Straits.	Japan.	Totals.
1895.....	£3,624,496	£1,652,523	£753,883	£175,000	£6,205,902
1896.....	4,838,778	740,285	824,536	1,029,300	7,432,899
Changes.....	Inc. 1,214,282	Dec. 912,238	Inc. 70,653	Inc. 854,300	Dec. 1,226,997

In addition to these exports the shipments of silver from San Francisco to the East in 1896 amounted to $14,840,541, of which China took $7,167,527; Japan, $4,848,305, and India, $85,000. The total showed a decrease from 1895.

AVERAGE PRICE OF SILVER PER OUNCE.

		Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1894	London, pence.....	30.81	29.18	27.28	28.95	28.69	28.68	29.82	28.29	30.88	28.69	39.41	27.78	29.13
	New York, cents.....	66.63	63.43	59.49	62.92	62.96	62.59	62.45	61.83	64.14	63.06	65.13	60.43	63.00
1895	London, pence.....	27.36	27.47	28.33	30.39	30.61	30.47	30.48	30.40	30.54	30.89	30.79	30.40	29.53
	New York, cents.....	59.69	59.90	61.98	66.61	66.75	66 64	66.75	66.61	66.90	67.64	67.40	66.47	65.28
1896	London, pence.....	30.69	31.01	31.34	31.10	31.08	31.46	31.45	30.93	30.19	29.68	29.46	29.70	30.67
	New York, cents.....	67.13	67.67	68.40	67.92	67.88	68.69	68.75	67.34	65.68	65.05	64.98	65.24	67.06

The New York prices are always per fine ounce, or ounce of pure silver; the London quotation is per standard ounce, or for metal .925 fine.

After making deductions for the amount of gold used in the arts, which is a somewhat uneven quantity, but which was probably less in 1896 than in 1895, we find that very little if any of the gold found its way into general circulation. The quantity actually coined and that deposited in the form of bullion was not equal to the amount carried into bank reserves and absorbed in the increase of the treasures of the different European powers.

The leading events of 1896 which affected the distribution of gold and silver were the determination of the United States to maintain the gold standard, and the continued preparations of Russia to resume specie payments on a new basis; abandoning the former gold rouble, and adopting a new rouble the gold value of which is equal to the average quotation for three years past of the credit or paper rouble—that is, about two-thirds of the old standard. Since the close of the year Japan has also begun preparations to adopt the gold standard in place of silver.

UNITED STATES IMPORTS AND EXPORTS OF GOLD AND SILVER.

	1895.			1896.		
	Exports.	Imports.	Difference.	Exports.	Imports.	Difference.
Gold:						
Coin and bullion....	$104,605,023	$32,538,736	Exp. $72,066,287	$56,742,844	$102,766,438	Imp. $46,023,594
In ores.....	362,379	1,857,656	Imp. 1,495,277	209,621	1,963,124	Imp. 1,753,503
Totals.....	$104,967,402	$34,396,392	Exp. $70,571,010	$56,952,465	$104,729,562	Imp. $47,777,097
Silver:						
Coin and bullion....	$53,833,153	$11,286,007	Exp. $42,547,146	$63,029,336	$12,504,477	Exp. $50,524,859
In ores.....	377,933	13,087,340	Imp. 12,709,407	994,405	17,730,280	Imp. 16,735,875
Totals.....	$54,211,086	$24,373,347	Exp. $29,837,739	$64,023,741	$30,235,757	Exp. $33,788,984

COINAGE OF THE MINTS OF THE UNITED STATES.

Year.	Gold.	Silver.	Year.	Gold.	Silver.	Year.	Gold.	Silver.
1891.....	$29,222,005	$27,518,857	1893.....	$30,038,140	$12,560,935	1895.....	$59,596,357	$5,698,010
1892.....	34,787,223	12,641,078	1894.....	99,474,913	6,024,898	1896.....	47,053,060	23,089,899

GREAT BRITAIN—IMPORTS AND EXPORTS.

Year.	Gold.			Silver.		
	Imports.	Exports.	Difference.	Imports.	Exports.	Difference.
1895.....	£36,005,999	£21,369,323	Imp. £14,636,676	£10,669,662	£10,357,436	Imp. £312,226
1896.....	24,468,337	30,123,925	Exp. 5,655,588	14,329,116	15,048,134	Exp. 719,018

FRANCE—IMPORTS AND EXPORTS.

Year.	Gold.			Silver.		
	Imports.	Exports.	Difference.	Imports.	Exports.	Difference.
	Francs.	Francs.	Francs.	Francs.	Francs.	Francs.
1895.....	253,875,195	244,381,198	Imp. 9,493,997	141,152,976	78,459,257	Imp. 62,693,719
1896.....	298,839,299	310,900,667	Exp. 12,061,368	155,750,771	97,260,425	Imp. 58,490,346

Abroad the accumulation of gold by the great banks was a feature of 1896 as of 1895. The following table shows the gold holdings of the leading banks through 1896, those of the New York banks being added for comparison:

Bank.	Gold Holdings in 1896. (a)		
	January 1.	July 1.	December 31.
Associated New York.....	$68,954,700	$61,866,300	$76,342,300
England.....	227,659,000	239,465,845	172,590,460
France.....	386,869,100	409,215,900	381,665,000
Imperial German.....	208,270,000	217,930,000	201,140,000
Austro-Hungary.....	120,203,750	136,210,000	151,070,000
Netherlands.....	17,584,000	13,172,000	13,171,000
Belgian National.....	20,666,000	20,652,000	20,137,000
Spain.....	40,022,000	42,028,000	42,642,000
Italy.....	59,860,000	60,625,000	59,835,000
Russia.....	396,370,000	472,715,000	489,980,000

(a) The holdings of the Imperial Bank of Germany and the Belgian National Bank include silver also.

The Present Development of the Barrel Chlorination Process.

By John E. Rothwell.

In The Mineral Industry, Vol. I., the writer gave a general description of the modern barrel chlorination process as it had been perfected at that time; since then the process has been recognized as a standard method for the extraction of gold from its ore, and a number of mills have been erected in different parts of the country which are uniformly successful in the treatment of different kinds of gold ore. Chlorination mills now in operation have a total capacity of about 1000 tons a day, and before the end of 1897 this capacity will be materially increased. To give an idea of the cost of treatment, it may be remarked that several of these mills buy ores at the rate of $19 per oz. for the gold, or practically 92½% of its mint value less $7 per ton for treatment.*

It is well known that many of the failures in metallurgical processes are due to improper mechanical appliances, which is largely on account of the lack of appreciation of the fact, as one writer has tersely put it, that "there is a vast difference between a test tube and a chlorination barrel." It is therefore my intention to devote the most of this article to a description of the mechanical appliances used in the process, and their operation.

Ores particularly adapted to chlorination are those carrying gold alone. Ores carrying a valuable quantity of silver cannot be treated economically except under particularly favorable conditions, and then a combination process must be used. Gold ores carrying copper, lead, zinc, arsenic, antimony, tellurium, and sulphur require a preliminary roasting. Lime and magnesia are detrimental chiefly on account of the higher consumption of chlorine-generating chemicals which they entail, but they can be rendered harmless, in some ores, by roasting with salt.

The applicability of an ore to chlorination can be determined by crushing a sample of several pounds to pass a 30-mesh screen, and roasting down to 0.25% S or less. Owing to the difficulty of getting similar conditions in roasting an ore in a small lot in a muffle, and in working it in a large lot in a mechanically rabbled furnace, great care should be exercised that the heat in the muffle be as uniform as possible, and not above a bright red. A small sample of the roasted pulp is panned down and the resulting concentrate examined closely under the microscope to learn its nature. The whole lot is then sampled carefully and divided into parts of 10 assay tons each, each of which is charged into a bottle of capacity about three times the bulk of the ore. Water enough is added to cover the ore, and then 5 gms. of chloride of lime and 10 gms. of 66° H_2SO_4 (commercial) are added. The bottle is then tightly stoppered, shaken well, and placed in as dark a place as possible, a tag of data being attached. Each of the lots is charged in the same way, and the whole series

*It is not to be inferred from this statement that the average salvage of gold in the chlorination works is 92½%. In the Cripple Creek district, where are situated the largest chlorination works buying ores in the open market, the actual figure is probably somewhat less. The difference between the actual salvage and percentage paid for is of course charged directly to cost of production. The treatment charge of $7 per ton is, therefore, intended to cover (1) actual cost of treatment, (2) value of gold paid for and lost, and (3) profit.—Ed. Mineral Industry.

agitated frequently. At the end of two hours the first is emptied on a filter, attached to an aspirator, and leached as thoroughly and rapidly as possible; the second one is taken at the end of three hours, and the third at four hours, and so on to the end. The tailings of each after being well washed are dried and sampled, the remainders being subjected to a screen analysis (40, 60, and 100-mesh) to determine the proportion and gold content of each size. A table made up from these data will show the most economical results that can be obtained with this ore, and the time of chlorination and fineness of pulverization necessary.

It has been suggested that hydrochloric acid should be used instead of sulphuric acid for the generation of chlorine from the chloride of lime; but the writer does not think this is good practice since it is likely to give considerably more unreliable results than the small amount of insoluble sulphate of lime left in the tailings when sulphuric acid is used.

After making these preliminary tests one is made on a lot of 50 or 100 lbs., which is rather tedious unless proper facilities are at hand, viz., a small reverberatory roasting furnace and a barrel that will hold 25 lbs. of ore and a proportional amount of solution, together with a number of stoneware vessels. The object of this test is to make an actual extraction and arrive at a result that can be checked with the assay extraction.

The erection of a works having been decided on, the mechanical appliances required to accomplish a desired result and the most economical arrangement to be made are the next problems for the engineer. The question of fuel and water supply having been settled, a site for the works has to be chosen. This may not necessarily be on a hillside, it being often more economical and in many ways more convenient to build on comparatively level ground, thus obviating the necessity of heavy retaining walls, and at the same time giving a construction enabling each department to be more easily and better lighted, and simplifying the supervision of the whole plant.

If the works is to be a public ore buyer, a sampling department will be necessary. In large works of this kind it is a good practice to "bed" the ores for a uniform mixture, but this is not a necessity since ores of different grades and chemical composition can be successfully treated after a complete analysis has been made.

Crushing Department.—In this department the aim of the engineer is to make the whole as automatic as possible, and reduce the amount of manual labor to a minimum. The ore after being fed into the rock breaker need not be touched again by hand, except in the event of hand sampling (quartering down) of a particular lot, until it is collected in the pulp bins that supply the roasting furnace. Ordinarily it will be sampled automatically.

Breakers.—In the preliminary crushing of the rock, a breaker of one of the well-known makes, viz., Blake, Dodge, or Gates, is employed to reduce the ore to about 1-in. cubes. These machines are so well known that a description of them is unnecessary. The principal point to be looked after with them is to examine the jaw-plates often, and see that full duty is being obtained and that a coarser product than the next machine ought to take is not being made.

Dryers.—For drying the ore the revolving cylinder dryer is generally used.

These are made sectionally of cast iron or in one or two pieces of ⅛-in. steel sheets, with the tires as separate pieces fastened to cast flanges, riveted to the shell, thus making it possible to remove and replace the tires when worn out. As the prime principle of all drying apparatus is to heat as large a volume of air as possible and bring it in contact with the material to be dried, the fireplace of the dryer is specially constructed with large air channels to heat the air before passing it to the cylinder. But since a large volume of air passing through the cylinder is likely to carry a large amount of dust into the dust chamber, the latter is arranged above the dryer so that the dust will collect in it and slide back into the cylinder automatically, obviating the necessity of rehandling it.

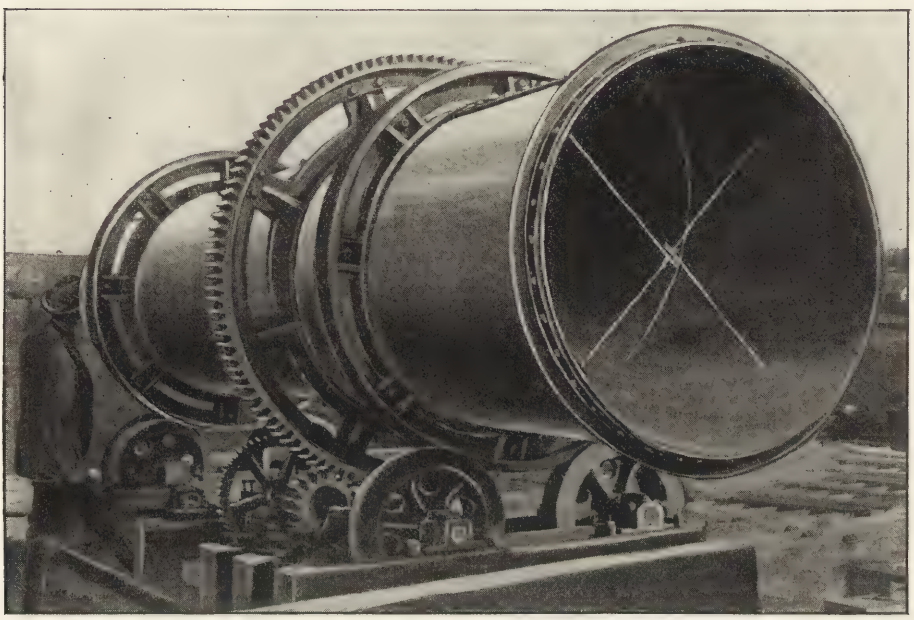


FIG. 1.—REVOLVING DRYER WITH THE ROTHWELL LONGITUDINAL DIAPHRAGMS.

Other appliances are the "shelf dryers" of the Stetefeldt and Krom type, and modifications of the Pearce and Ropp furnaces, but these are much more expensive to install than the revolving cylinder, and as to efficiency and economy there is very little difference, if any.

Crushing Rolls.—To get the most economical work and the greatest capacity from crushing rolls a considerable amount of attention should be given them. One of the principal points in this connection is even and steady feed. If this is observed the rolls will wear down as evenly as though turned in a lathe. The faces should be examined often, which is best done by placing a light below the rolls and looking down between them; for this reason the construction of the foundation should be made convenient for such an examination. When any unevenness can be detected the faces may be trued by putting into the hopper a piece of sheet iron that will deflect the ore away from the

more worn portion to the less; in this way, even with a poor feed, the roll faces can be worn down evenly, and there is no necessity of turning the roll true with a lathe tool or emery wheel, which not only means a loss of time to the crushing plant but also a removal of metal that should be utilized in the crushing of the ore.

The roll-shaft journals should all be well protected from dust, and kept well oiled; the roll faces should never press on each other, but should be kept apart, a little less than the size to which the ore is to be crushed. The pressure on the rolls should be maintained at as even a tension as possible, and should be sufficient to break down the hardest piece of rock that has to be crushed.

Rolls should be run in series as much as possible, with a screen of corresponding size between each set; thus the rock coming to the first (coarse) set should not be coarser than 1½-in. cubes for 36-in. diameter rolls, which should be adjusted about ½ in. apart, crushing to that size. The next reduction should be from this ½-in. product through a set of rolls about ⅛ in. apart, the product from these rolls being screened to remove all that is coarser than No. 6 mesh. The coarser pulp goes back to the intermediate rolls, and the finer that passes the No. 6 mesh but remains on the final mesh, goes to the fine crushing rolls. The last are set as rigidly as possible, and particular attention is paid to the evenness of the feed, since with uneven feeding or a feed too rapid for the peripheral speed of the rolls the tendency is to heat the bearings unnecessarily, and at the same time allow a considerable amount of coarse material to pass through, which is rejected by the sizing screen and returned to the rolls, thus lessening their capacity.

With proper feed and surface velocity not over 10% should be returned. The feed should be so regulated that the finished product will cover a surface equal to the superficial area of the crushing surface to a depth in inches equal to this size; thus if we are reducing the rock to a 20-mesh product, *i.e.*, about $\frac{1}{32}$ in., with a set of rolls 36 in. in diameter and 16-in. face, running 90 revolutions per minute, we have a superficial area of 1130 sq. ft. of crushing surface presented every minute; now if this surface is covered to a depth of $\frac{1}{32}$ in. it would equal a weight of about 300 lbs. of ore per minute, or a capacity of 9 tons per hour reduced to this fineness. From this will be seen the necessity of a perfectly uniform feed, and that to get a good capacity it is not necessary to crowd the rolls; also that the speed can be materially reduced, preventing to a large extent heating of the journals.

Elevators.—Elevators for handling the dry ore in the mill are usually made of a rubber belt, to which the buckets are bolted. The belt should be at least 2 in. wider than the length of the bucket, and of what is known as "special elevator belting," which has an extra heavy coating of gum rubber, to prevent being punctured by the sharp points of the rock that come in contact with it. Head-wheels should not be less than 30 in. in diameter, and for belts wider than 12 in., 36 in. in diameter. Elevators should have an inclination of about ½ in. to the vertical foot. Large buckets placed about 16 in. apart and bolted securely to the belt should be used.

The belt when first put on should be drawn as tightly as possible with belt-

tighteners, and should be allowed to stand a day or so to stretch, then taken up again and the ends lapped and bolted together, the bolt-holes being so placed that when necessary to tighten the belt again the holes, if moved up from 2 to 3 in., will all correspond. The chute leading into the elevator should be somewhat narrower than the length of the bucket, and deliver the ore directly into the buckets, at a point from 16 to 18 in. above the center line of the boot-wheel. The discharge chute should be placed about 10 in. below the center line of the head-wheel. The proper speed for the belt is from 260 to 300 ft. per minute. The chute into which the elevator discharges should be built so that a small amount of ore will be retained at the point at which the ore strikes, the object being to protect the bottom of the chute from the abrasive action of the ore when thrown from the buckets. All chutes must have an inclination not less than 45°, and as much greater as it is possible to give them.

Sizing Screens.—In estimating the superficial area of screens required for a mill of a given capacity it has been found that 4 sq. ft. is necessary per ton of ore in 24 hours when sizing through a 24-mesh. When sizing through 8-mesh 2 sq. ft. of screen will be sufficient. It is, however, necessary to give the screens a thorough brushing several times a day, otherwise the meshes will soon become clogged and materially reduce the capacity. Numerous devices have been designed to keep the mesh clean, but all give more or less trouble, so that the simplest means is to design the casing around the screens to make the latter easily accessible for cleaning with a wire brush by hand, which requires very little time, and at the same time means an examination of the mesh for breaks and holes.

Hexagonal-pyramidal, conical, and cylindrical or hexagonal-prismatic sizing screens are used. The author prefers the hexagonal-pyramidal or prismatic on account of the ease of renewals. These screens should always be made double-mesh, that is a coarser within the finer, to relieve the fine of undue wear from coarse particles of ore.

It is necessary, when using a very fine wire mesh, to place the cross-bars of the screen-frame close together, and in putting on the wire-cloth to draw it as tightly as possible, otherwise there will be danger of its working with the rotation of the screen, and breaking along the edge of the frame.

The proper speed for screens is 170 ft. per minute; with pyramidal screens this speed is measured at the largest diameter. The inclination of the screen or screen-frames should be from $\frac{3}{4}$ to 1 in. to the foot in length. With hexagonal screens there should be always on hand several extra sections to replace those that become torn or broken. The driving gear should be arranged so that when necessary to stop one screen it can be thrown out of gear without stopping the whole mill. The chutes that convey the ore to the screens should be arranged with gates, so that the stream of ore can be shut off at any time when necessary to repair a screen.

A new style of sizing screen that has come into general use recently is called the "Berthelot" patent, after its inventor, who uses it for screening ground cement. Its simplicity is its valuable feature, and, as no power is required to operate it, it reduces the number of mechanical appliances to be looked after

in the crushing department. It is much cheaper to install, requires less screen-cloth, and is much more easily and quickly repaired than the ordinary screens. The arrangement is simply two, four or six screen plates about 10 in. wide, set at an inclination of 45°, on which an elevator discharges the material to be screened, the whole being arranged so that the screen, rolls, and elevator form a unit in a crushing mill. Owing to the angle at which the screen is placed, the mesh of the cloth is always several sizes coarser than the required pulp; thus to get a 50-mesh pulp a 30-mesh cloth is used. The size of wire used is also finer than the ordinary.

Belts, Pulleys, etc.—In preparing the ore for treatment it is very essential that each individual operation should have close supervision. In order that the maximum capacity may be maintained continuously the energetic mill-man will leave nothing to chance, but will give each particular piece of apparatus a careful looking over once every 24 hours; particularly when starting up new machinery it should have the closest attention, and should never be driven to its capacity, as there are bolts to be tightened, pulleys to be looked after, journal boxes that may become heated, and various other little details that must be cared for until "they find themselves."

All driving belts should be put on with belt-tighteners and allowed to stand from 24 to 48 hours before being laced; the belt should not be unnecessarily weakened by lace-holes, and the lacing should be done with the best grade of leather, a sufficient number of laces being put in to attain the maximum strength at that point. The best system of lacing is the lock-stitch; in which each single strand is securely locked, so that should one or two of the strands be cut or broken the whole will not draw out. Proper attention to this detail will save many hours' stoppage, and added cost to the treatment.

Another point the successful mill-man will look to is that all journal bearings are amply protected from dust, and kept well oiled; not lavishly, but sufficiently to insure smooth running; also that the dust that accumulates inside the rim of pulleys is removed often, preventing in a large measure the excessive amount of dust usually found in a mill where this is not done. Although it is practically impossible to prevent dust in a dry crushing mill, with proper precautions in construction the amount that escapes need not be excessive or detrimental to the health of the men employed.

Roasting Furnaces.—The mechanically rabbled hearth-furnace is now most generally used for roasting the ore. Three types have come into use, viz., the "Pearce turret," the "Ropp straight line," and the "Brown elliptical" and "horseshoe," each of which has its particular admirers. All of them have merit, and have been the means of reducing the cost of roasting to a considerable extent from what it was a few years ago. Automatic cooling devices are now added to the furnace, so that the large cooling floor usually required, with its necessary labor, is done away with, and the cost of treatment still further reduced.

The Pearce Furnace.—Figs. 2A and B shows the Pearce "turret" furnace, of which there are more in general use than any other. This furnace is built of the double and single-hearth type, of three sizes, viz., 6, 7, and 8 ft. width of hearth; it is also arranged so that a drying hearth can be made of the top arch and thus

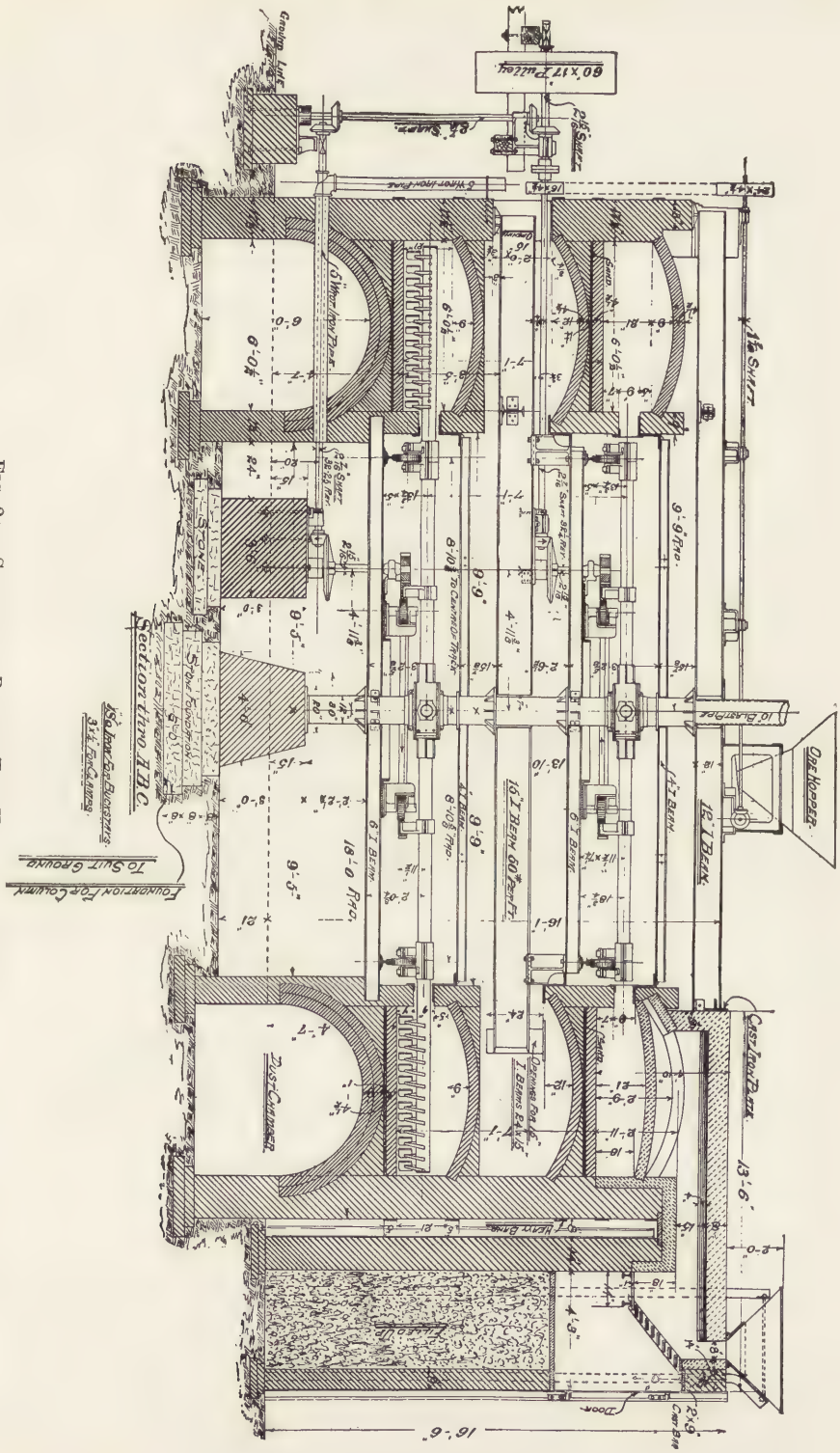


FIG. 2A.—SECTION OF PEARCE TWO-HEARTH FURNACE.

utilize heat that is otherwise lost. The number of fire-boxes varies with the character of the ore to be treated and the nature of the fuel to be used; usually two are all that are needed, but three are used where the ore has to be simply calcined, and a large capacity required. When petroleum-residuum is used the projecting fire-boxes are omitted, and a special combustion chamber built directly over the hearth, through the walls of which the oil burners

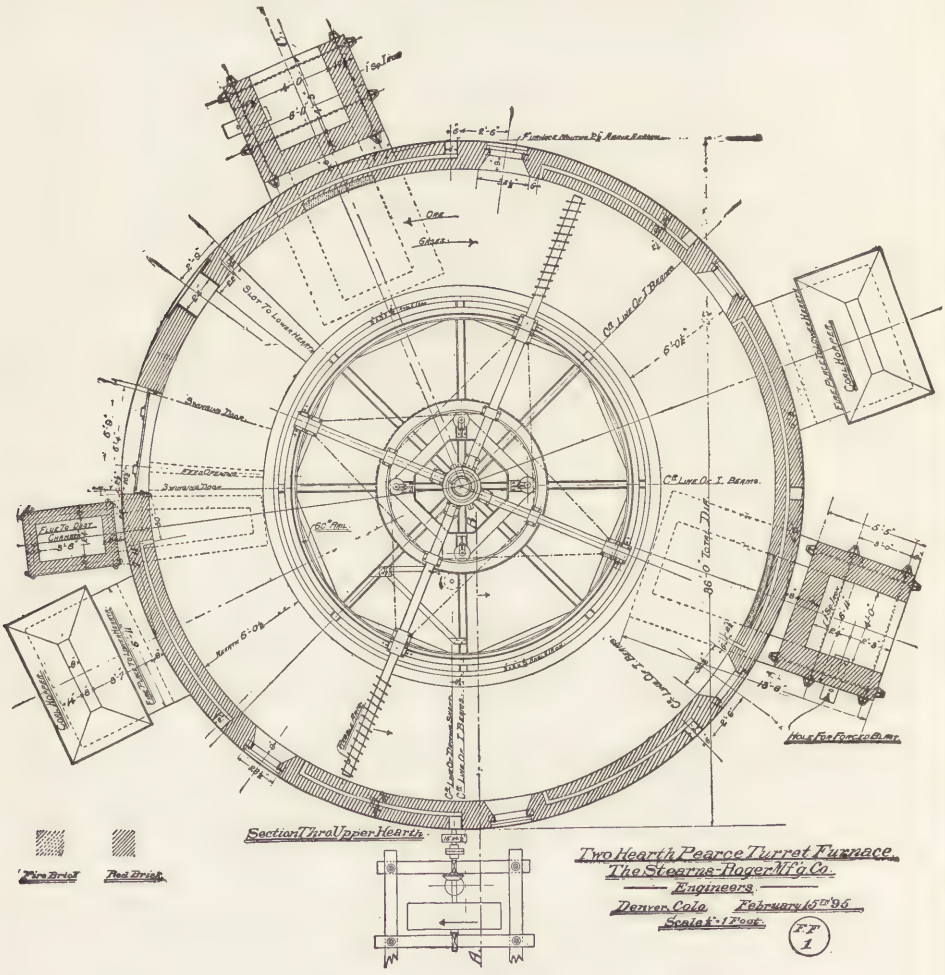


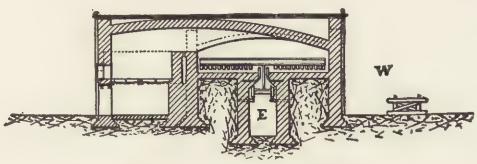
FIG 2B.—PLAN.

project and distribute the flame over the whole width of the hearth, an arrangement which has been found very efficient.

On the latest furnaces the rabble arms to which the plows are attached are water jacketed, thus keeping them cool and adding largely to their life, without materially affecting the consumption of fuel in the operation of the furnace; of course, when the water-jacketed rabble arms are employed the air fan is not used, all the air that is required in roasting being let in through the fire-boxes and the annular channels in the outer wall.

Another arrangement, by which a single rabble arm is used to move the pulp forward on the hearth and rabble, is a special mechanism attached to the driving gear which stops it in the open space, over the discharge hopper, where it remains for a specified time to cool off. This ingenious mechanism has proved very efficient when roasting certain pyritous ores, where continuous rabbling is not necessary, since the rabble arm can be run periodically around the hearth, turning up a fresh surface of ore, from which the sulphur, arsenic, and antimony can be driven off; the time of revolution and length of stoppage being automatically controlled from a uniform engine-belt speed.

The Ropp Straight-line Furnace.—Though this furnace is a comparatively new design, several of them have been put in operation for roasting ore for cyanide and chlorine treatment, but very little data is available as yet as to their economical results. As is usual with all new appliances, various mechanical difficulties have been met with, but it is quite probable that after the necessary experimenting has been finished the furnace will prove a very efficient appliance for the work intended.



SECTION THROUGH XY.

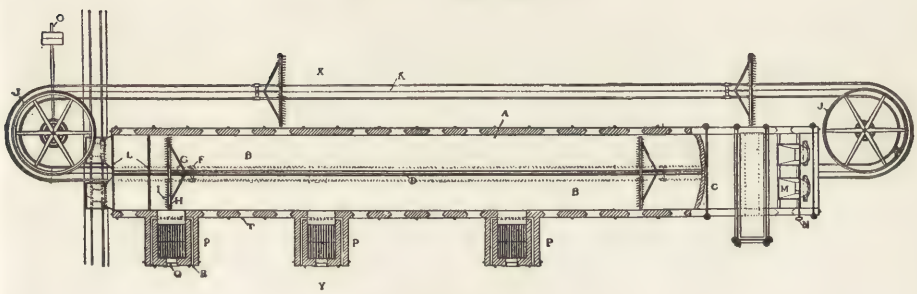


FIG. 3.—PLAN.

The furnace (see Fig. 3) consists of an ordinary reverberatory hearth of a rectilinear form. In the center of the hearth there is a longitudinal slot as shown, through which an arm from a four-wheeled truck running beneath the hearth extends upward and carries the rabbles. The chamber beneath the hearth in which the truck runs extends the entire length of the furnace and is made sufficiently large to admit a man under the carriage to make any repairs that are necessary. Four of these carriages are placed at equal distances from each other, and are attached to the steel-wire rope, by which they are propelled. The rabble arms are so constructed that each side can be adjusted independently of the other by means of screws and slots, and can be set so that the rabbles shall reach the ore on the hearth to the proper depth. The driving mechanism consists of two bevel gears with their shafts, and a horizontal sheave on the vertical shaft around which a steel-wire rope travels. A similar sheave is

mounted on a vertical shaft at the opposite end of the furnace. All this mechanism is outside the furnace, accessible for oiling and repairing. The rope is said not to get hotter than the hand can bear. The ore is automatically fed into the furnace at the feed end through Challenge feeders, and is moved along the hearth by the rabbles at any speed desired, and is automatically discharged into a hopper or car at the other end. During the time required for the carriages to pass over the length of the track exterior to the furnace, from the point of exit to the point of entrance, the rabbles are cooled and in condition to again enter the furnace. Three or more fireplaces are used, depending on the fuel and the work to be done, and can be arranged for either hand or automatic firing. The standard furnace is 105 ft. long and 11 ft. wide in the clear.

The H. F. Brown Furnace.—The furnaces of this name are styled the "horseshoe," "elliptical," and "parallel." The general construction of each has been described so frequently in the technical journals that a further account is unnecessary here. The design, Fig. 4, will give a fair idea of the arrangement of the elliptical furnace used at the Golden Reward Works at Deadwood, S. Dak. This furnace has been in operation continuously for over a year and has given very general satisfaction, but exact data of its operation are not available. Its dimensions are: Roasting hearth, 180 ft. long and 8 ft. wide; extension cooling hearth, 78 ft. long and 8 ft. wide. The capacity of the furnace is stated as 100 tons per 24 hours of ore crushed to 30 mesh, carrying from 2½ to 8% down to 0.3% sulphur, including cooling and delivery to the chlorination barrels. In the construction of this furnace the material used was as follows: Stone-work foundation, 400 perches; red brick, 90,000; firebrick, 8100; lime, 250 bu.; sand, 100 yards; fireclay, 12 tons; iron-work, 50,000 lbs. Mason-work required, 250 days' labor; the ordinary labor account came to 250 days; and mechanical labor, 20 days.

In estimating the capacity of a roasting furnace one must be governed by the character of the ore; thus with a hearth furnace the capacity has been found to be as high as 1 ton of ore per 24 hours to 11 sq. ft. of hearth area. This ore carried about 1.5% S, and the sulphur content was reduced to 0.18%. On other ores carrying from 2½ to 8% S the capacity has been found to be 1 ton to every 14 sq. ft. of hearth, roasting down to 0.3% S, while on concentrates carrying approximately 42% sulphur the capacity is 1 ton to every 55 sq. ft. of hearth area, desulphurizing down to 0.3%.

With revolving cylinder furnaces, the writer has obtained a capacity of 1 ton to every 7 ft. of superficial area of lined cylinder, on an ore carrying approximately 1 to 1.25% S, roasting sufficiently low to give a high extraction by chlorination, or approximately 0.25%; and on an ore carrying approximately 50% volatile matter, arsenic, and sulphur, a capacity of 1 ton to 42 sq. ft. of area down to approximately 0.5% S, has been obtained.

To obtain the best results from a furnace it is necessary that the feeding of the ore should be positively uniform, and the firing maintained at as nearly an equal temperature as possible. For chlorination the ore must be roasted until the sulphur content does not exceed 0.25% S, while 0.14 to 0.18% give more satisfactory extractions.

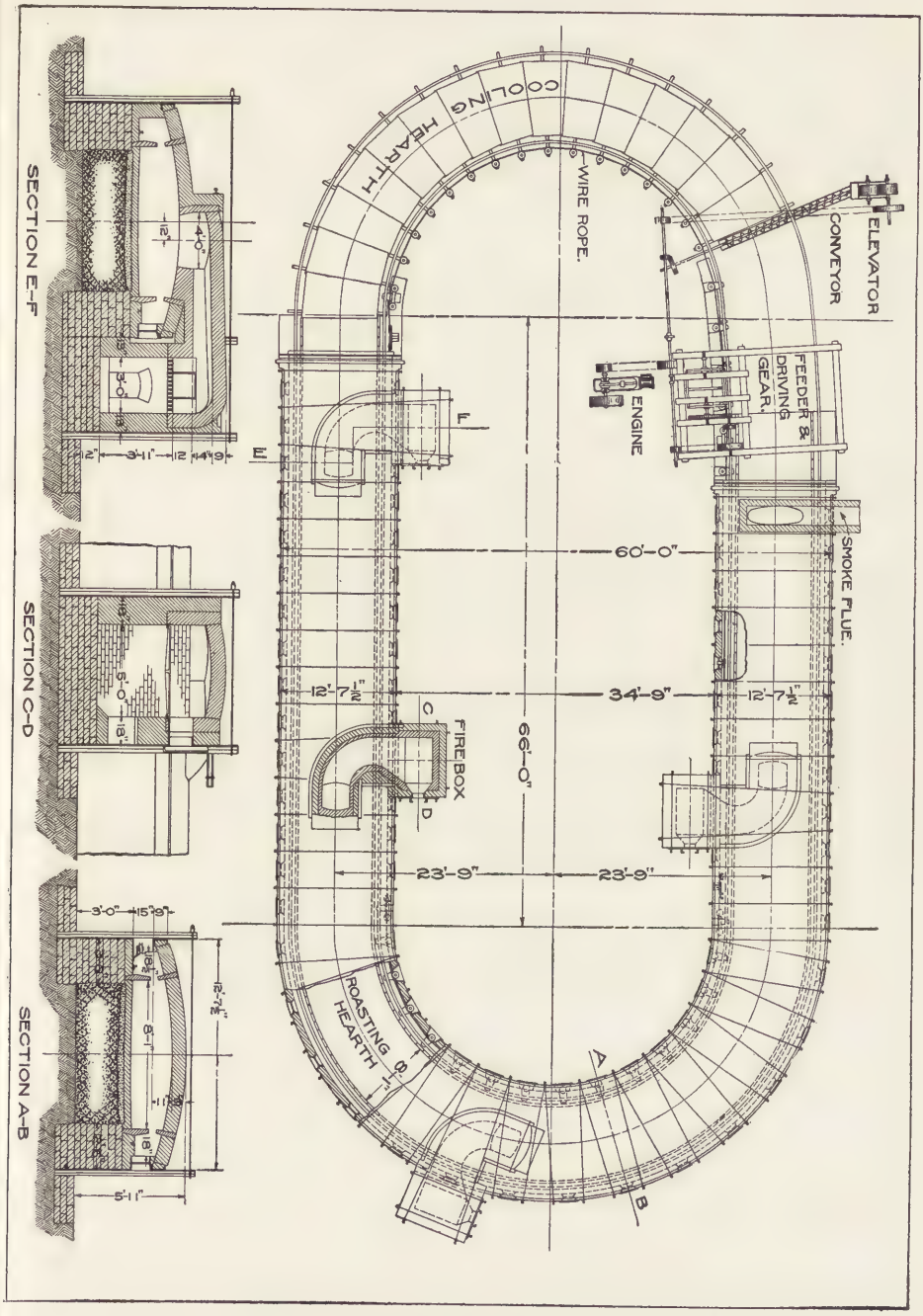


FIG. 4.—THE BROWN COMBINED ROASTING AND COOLING FURNACE.

PLAN AND SECTIONS.

The Chlorination Department.—The accompanying general plan (Fig. 5) will give a very economical and convenient arrangement of this department. The roasted ore is elevated and distributed in a long storage bin, from which chutes, on hinges, are let down and a gate raised to charge the barrels. After the required time for chlorination the barrel is stopped, with charging-hole up, and water connections made to a valve on the barrel; at the same time the valve below is opened and the leach runs into the settling tanks directly below. From these tanks the solution can be drawn into the No. 2 settling tanks and from there decanted through a floating hose-pipe siphon into the precipitat-

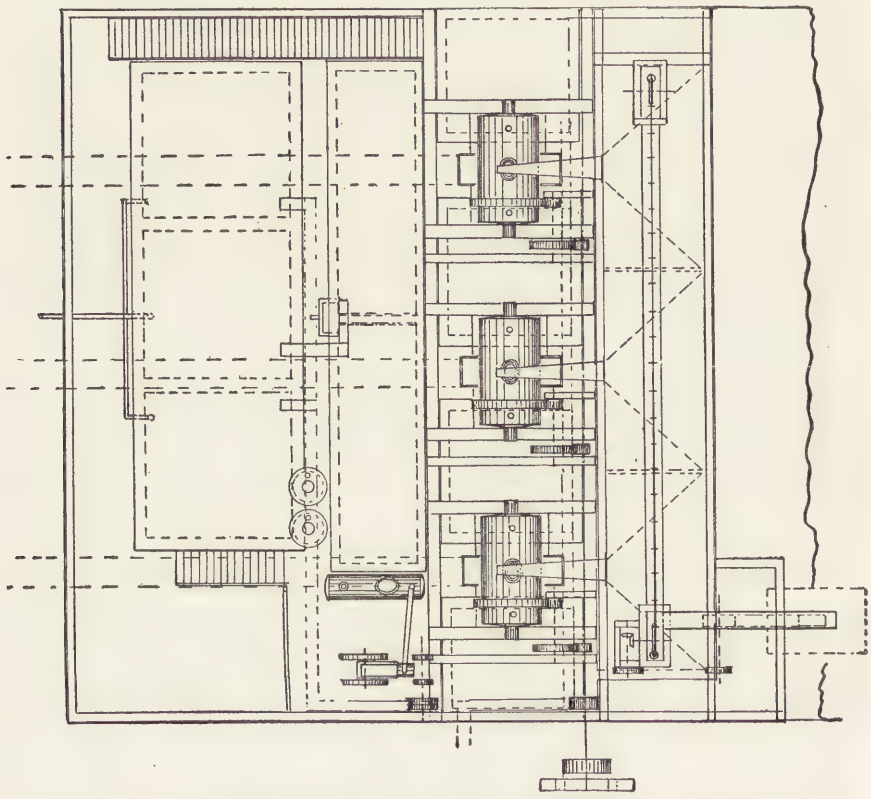


FIG. 5A.—PLAN OF CHLORINATION WORKS.

ing tanks lower down. No difficulty whatever is experienced with this arrangement in getting perfectly clear solution in the precipitation tanks.

The tailings are discharged by removing the charging-hole cover and rotating the barrel, flowing through the tailings chutes to the dump.

The generators for SO_2 and H_2S are placed just above the precipitating tanks, to which they are connected with a short line of lead pipe and rubber hose.

A small belt-driven air compressor and air receiver are located on a frame, close to the end of the No. 2 settling tanks, and the compressor driven at will from a counter shaft, connected by belt to the main driving shaft, with tight and loose pulleys.

From the precipitating tanks a pipe is run to the pressure tanks and filter presses, which are located in a separate building, where also is built the sulphide-roasting muffle furnace, and the melting furnace. An air pipe is also run from the air receiver to the pressure tanks, so that when necessary air pressure can be used to force the solution through the presses, and also for the drying of the sulphide cake in the press. This general plan makes a very simple arrangement and gives an abundance of light in the whole department.

The tank arrangement varies with the location of the mill site. In some mills the barrels are kept down close to the ground and the solution collected in a line of shallow settling tanks, from which it is drawn into a "Montejus" tank and forced to the settling and storage tanks, placed on a framework some

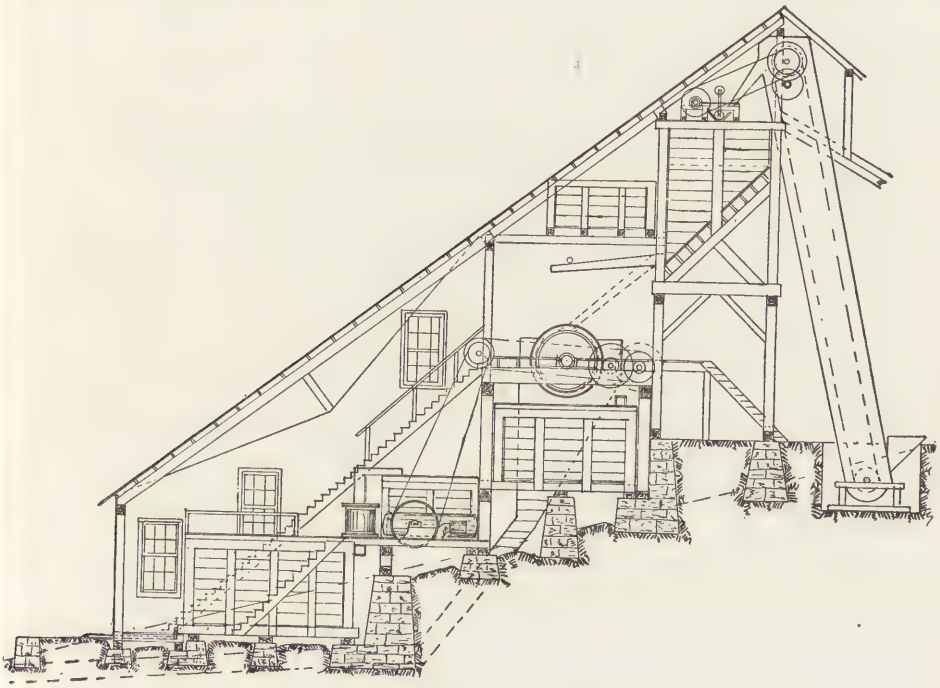


FIG. 5B.—SECTIONAL ELEVATION OF CHLORINATION WORKS.

distance above the ground, and is in turn drawn down from these into the precipitating tanks, and then in another step to the pressure tanks and filter presses; while in a more recent arrangement the tanks are all placed on a level, the barrels being on a line above the settling tanks, and the solution transferred from tank to tank by a very simple system of siphons. These tanks are elevated on a framework or walls about 8 ft. above the ground, and allow of thorough inspection for leakage, the space also serving as storage room for chemical supplies, etc. The pressure tanks and filter presses are then placed on the ground floor directly under the precipitating tanks and require only short connections. This arrangement gives an ample tailings dump even when the mill is erected on level ground.

Bins.—It is usual to have a storage bin of suitable capacity for the cooled ore after roasting, and it is essential that an accurate weight and sample of the ore should be made, between the bins and the barrel hoppers, to check results of work.

Barrels.—The chlorination barrels must be strongly constructed and lined with heavy chemical lead. As built now they can be made to last a very long time without repairs of any kind. The larger sizes, of 5, 7½, and 10-ton capacity,

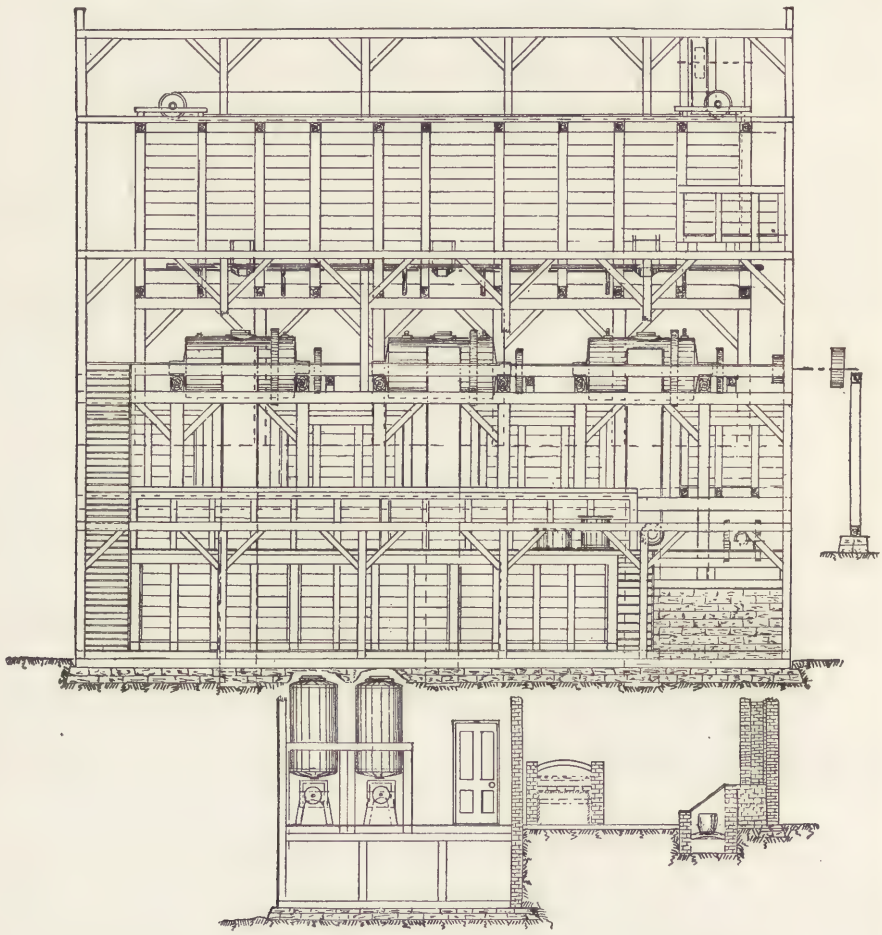


FIG. 5C.—LONGITUDINAL ELEVATION OF CHLORINATION WORKS.

have two charging-holes, so that the ore can be distributed over the internal length of the barrel, and allow of the whole charge being put in without the extra labor of pushing back. It has been found possible in ordinary practice to discharge, wash, and recharge a 5-ton barrel in 18 to 20 minutes, this period being taken from the time one charge had finished leaching until the next charge was in the barrel and the machine again in revolution.

Filter.—The filtering diaphragm has not been changed materially from what

the author described in a previous article in this series.* In some mills practically the same idea is still in use, in others the corrugated plates have been replaced with a sheet of heavy lead (24 lbs. per sq. ft.) perforated all over with holes $\frac{3}{8}$ to $\frac{1}{2}$ in. in diameter. This sheet is supported on a number of segments that fit the shell of the barrel. It is then covered with asbestos cloth, and the cloth with an open wooden grating, the whole being held in place with cross-beams in the usual way. This arrangement has proved quite satisfactory on some ores.

The writer has arranged a filter bed of sand, which does away entirely with the expensive asbestos cloth, and at the same time gives a longer life to the diaphragm; also much less danger of imperfect leaching, or openings that allow the sand to get into the settling tanks. This filter was described in the *Engineering and Mining Journal,* October 19, 1895.

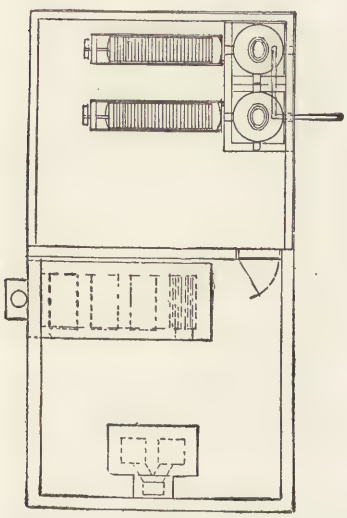


FIG. 5D.—PLAN OF MELTING DEPARTMENT.

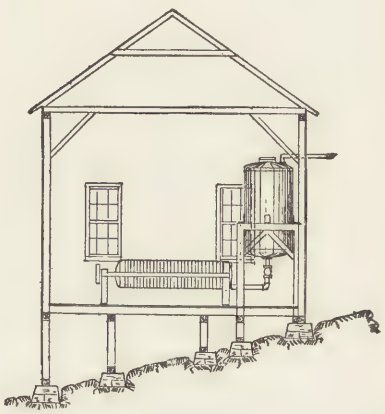


FIG. 5E.—ELEVATION OF MELTING DEPARTMENT.

The general principle involved in a filter as placed in a chlorination barrel is simply a diaphragm of reasonable resistance to the corrosive action of the chemicals used, and porous enough to hold the pulp at the commencement of leaching, since the pulp, as soon as it has settled on the diaphragm, forms a perfect filter of itself, therefore it matters little what this diaphragm is made of as long as it has the qualities specified.

It has been found in some cases that the pulp in that segment of the barrel described by a perpendicular line from the side of the diaphragm is imperfectly leached. This is liable to occur if the diaphragm is confined to narrow limits. It is, however, a point that it is well to watch for, and when found to occur, if the diaphragm cannot be widened, the charge should be rotated once or twice during the leaching, thus avoiding a discrepancy in the bullion output and higher tailings than should be.

* THE MINERAL INDUSTRY, Vol. I., 1892.

Another point to be looked after is the best pressure to leach at, since each ore has its particular leaching quality, and pressure at which the best results are obtained. An excessive pressure may result, if suddenly applied, in an almost complete stoppage of the filtering, or a channeling of the pulp and naturally an imperfect washing of the whole.

It is well in practice to allow the charge to stand a few minutes before starting to draw off from below the filter, thus giving the pulp a chance to settle quietly on the diaphragm; then the valve below is opened and the solution allowed to run out, the pressure of gas usually remaining in the barrel being sufficient to force a large part of it out fast enough; as the flow commences to decrease the hydraulic pressure is put on gradually until it assumes the pressure at which the flow has been found to give the best results. A little thought will show the reason of this consideration. Thus, if a charge is started rapidly, channels are formed, and a very rapid flow results, that when tested will show gold at the start, but a rapid decrease in the quantity and then a slight color for a great length of time, a large quantity of solution, and very likely high tailings. Or a sudden high pressure being applied, the particles of pulp are driven closely together, the fine filling up the interstices between the coarse, and thus practically stopping the flow entirely, which means a greatly lengthened time of washing and usually an imperfect one. From this it can be seen that it is very necessary to keep the mass of pulp as permeable and homogeneous as possible in order to get the best results, and this can only be accomplished by a careful study of the leaching character of the ore.

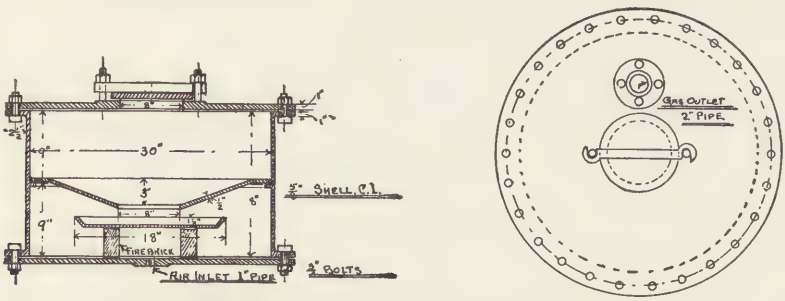
Solution Tanks.—These are usually made of a strong framework of wood, well put together, but not necessarily water-tight, and the whole lined with chemical sheet lead of 6 lbs. per sq. ft., all joints being burned together. For a description of the method used, see an article on lead burning by the author in THE MINERAL INDUSTRY, Vol. IV. It is necessary in designing works to always allow sufficient tank capacity, so that all solutions will have ample time to settle perfectly clear before being drawn down to the precipitating tanks; otherwise any slime allowed to run into the latter will give a low-grade precipitate that is difficult to smelt, besides clogging the filter press quickly, and requiring frequent changing of the cloths, meaning, also, an added expense to the treatment.

Precipitation of Gold from Solution.—Three reagents are used in practice for precipitating gold from solution, viz.: (1) Ferrous sulphate; (2) sulphur dioxide and hydrogen sulphide; (3) charcoal.

Ferrous sulphate is not used extensively where ores are treated in large quantities, because of the slowness with which the gold settles and the large tank capacity required to store the solutions for a sufficient length of time to settle. The precipitated gold is in such a finely divided condition as to be practically impossible of collection on the ordinary filtering medium used in the filter press. Other means of collection have been proposed, but as yet none have been put in practice.

Sulphur dioxide and sulphureted hydrogen is the precipitant most generally used when large quantities of solution are to be handled. These gases are generated in the following manner: The sulphur dioxide generator, shown in

Fig. 6, is made of cast iron, and is suitable for a mill treating 100 tons a day. In operation the amount of sulphur, in the form of crude brimstone or roll sulphur, needed to destroy the free chlorine in the tank of solution to be precipitated, is placed in the burning tray, a small piece of the sulphur is ignited and dropped into the mass, then the cover is put on, clamped tight, and a current of air turned on at the inlet pipe, gently at first, so that the burning

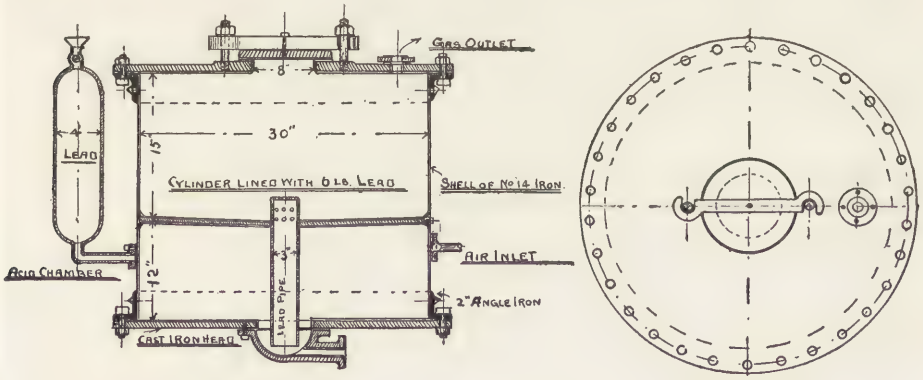


VERTICAL SECTION. PLAN OF COVER.

FIG. 6.—SULPHUR DIOXIDE GENERATOR.

sulphur is not put out, then increased until copious white fumes are noted in the upper part of the tank being precipitated. The entire action requires only a few minutes to accomplish.

The test usually made to show that a sufficient amount of the sulphur has been used is to take a beaker full of the solution and allow two or three bubbles of hydrogen sulphide gas to pass through it, which should immediately start a permanent precipitation of the gold. With rich solutions it has



VERTICAL SECTION. PLAN OF COVER.

FIG. 7.—HYDROGEN SULPHIDE GENERATOR.

sometimes been advantageous and economical to carry the precipitation of the gold with the SO_2 to a considerable extent before introducing the H_2S, the finishing of the precipitation with the H_2S forming sufficient gold sulphide to collect the finer metallic precipitate brought down by SO_2, 30 minutes being sufficient to give an almost perfect settling in a tank of solution 6 ft. deep.

Fig. 7 gives the construction of the hydrogen sulphide generator. It is made of light sheet steel, with angle-iron flanges and a cast-iron cover. The

cylinder is divided with a conical partition having a hole in its center. It is lined with lead, and a lead pipe with a perforated cap is inserted through the hole in the center of the conical partition and fastened tightly to it. An acid chamber is attached to a connection on the side and delivers the acid charge below the partition. In operation the sulphide of iron is placed on the top of the partition in pieces not larger than 1-in. cubes, sufficient water to cover the bottom of the generator about 4 in. deep is added, and the cover closed; then the requisite quantity of sulphuric acid is let into the lower division from the acid chamber. The generator is now charged and when required for use all that is necessary is to turn on a current of air at the inlet, which forces the acid solution up the central pipe into contact with the iron sulphide, and the gas generates. The air is allowed to blow through and carry the gas to the bottom of the tank of solution, at the same time stirring the solution.

The time of precipitation of a tank of 10,000 gals. of solution should not exceed 10 minutes. It has always been claimed that where there is copper and gold in solution together, H_2S will precipitate copper with the gold, and necessarily make a very base bullion to handle; the writer, however, has demonstrated that, with the ordinary large quantities of solution and judicious management, a "fractional precipitation" is possible, in which the gold can be entirely precipitated, leaving the copper still in solution; also, that this precipitation is very thorough, as tests of the solution have demonstrated that barely a trace of gold will remain, and the bullion produced will assay from 820 to 960 fine.

Pressure Tanks.—Fig. 8 shows the general design of a pressure tank of about 700 gals. capacity. It is built of $\frac{3}{8}$-in. sheet steel and $\frac{1}{2}$-in. steel dished heads, the whole shell being lined with 8-lb. lead, well bolted in, and having two openings at the top for the inlet of solution and air, and one opening in the center of the bottom for connection with the filter press. One pressure tank is required for each filter press because it is often necessary to keep one press running on solution while the other is being used to make a clean-up. Air-pressure is used when it is desired to increase the speed of filtration. Care must be taken not to allow the tank to fill up, and then be drawn off without admitting air to fill the partial vacuum formed as the solution runs out; otherwise the lining is liable to collapse and give trouble.

Filter Presses.—The filter press generally used is the Johnson 19-in. square with flush plates and distance frames, with 24 or 36 chambers. The outlet for the filtrate should be through an opening in the lower corner of the filter plate and leading into a trough on the side of the press. The object of having these separate outlets is to be able to examine the filtrate from each plate, to ascertain any break in the filtering medium that may let the precipitate through. Should this happen the outlet of the imperfect chamber can be stopped up until the clean-up is finished. The filtering medium is usually a heavy mat cloth with the nap left on one side, this side being placed next the chamber; on this cloth is placed a light close-woven chain-cloth, or a 10-oz. Canton flannel, with the nap side in. When starting the liquor from the tanks through the press it is necessary to let it on slowly at first; since turning it on rapidly means a loss of precipitate, until the cloths have become firmly bedded and covered.

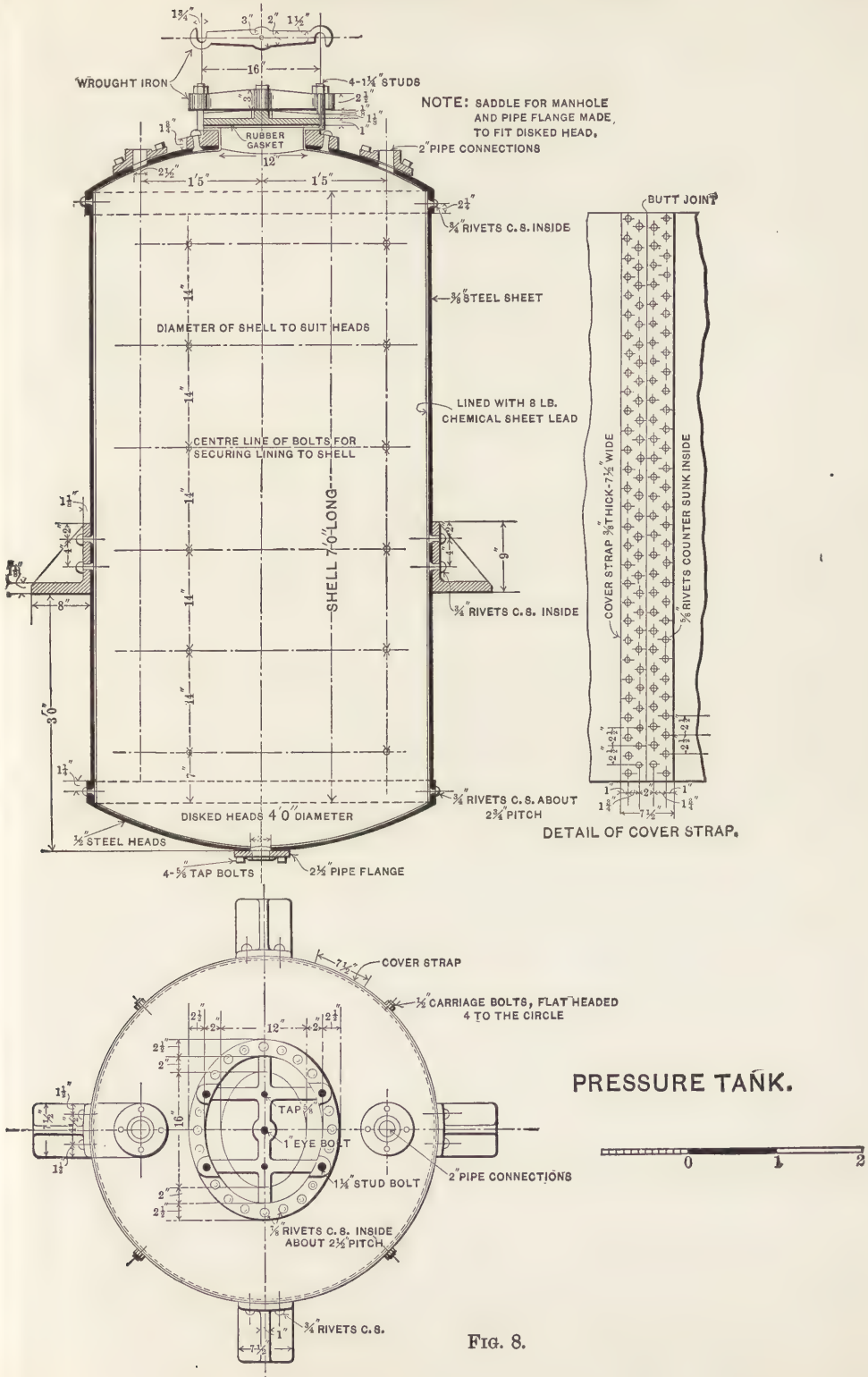

WROUGHT IRON
4-1¼" STUDS
RUBBER GASKET
2" PIPE CONNECTIONS
NOTE: SADDLE FOR MANHOLE AND PIPE FLANGE MADE, TO FIT DISKED HEAD.
¾" RIVETS C. S. INSIDE
⅜" STEEL SHEET
DIAMETER OF SHELL TO SUIT HEADS
LINED WITH 8 LB. CHEMICAL SHEET LEAD
CENTRE LINE OF BOLTS FOR SECURING LINING TO SHELL
SHELL 7'0" LONG
¾" RIVETS C. S. INSIDE
¾" RIVETS C. S. ABOUT 2¾" PITCH
DISKED HEADS 4'0" DIAMETER
½" STEEL HEADS
4-⅝" TAP BOLTS
2½" PIPE FLANGE
BUTT JOINT
COVER STRAP ⅜" THICK-7½" WIDE
⅝" RIVETS COUNTER SUNK INSIDE
DETAIL OF COVER STRAP.
COVER STRAP
½" CARRIAGE BOLTS, FLAT HEADED 4 TO THE CIRCLE
TAP ¾"
1" EYE BOLT
1¼" STUD BOLT
2" PIPE CONNECTIONS
⅛" RIVETS C. S. INSIDE ABOUT 2½" PITCH
¾" RIVETS C. S.
PRESSURE TANK.
0
1
2

FIG. 8.

with a coating of precipitate. The speed with which the liquor can be run through will depend on the condition of the precipitate and the cloths.

Paper has been tried in place of cloth, but it is generally unsatisfactory, because of its liability to tear when the liquor is first turned on, resulting in the loss of some of the precipitate, and the large amount of ash in the paper that increases the amount to be melted, and requires more flux to get rid of it.

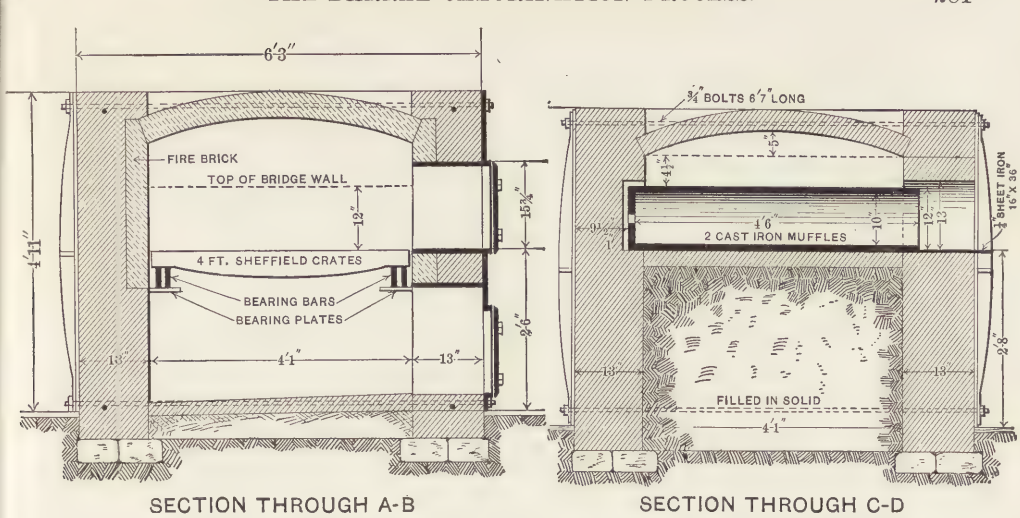
In practice, if the precipitate has been dried to a cake by allowing the air to blow through for some time, there will be little difficulty experienced in getting it off the light cloth and into a roasting pan placed under the filter plates. The light cloths are cleaned as thoroughly as possible at each clean-up, and then at the end of every third or fourth are removed and burned separately and a new set of cloths put on. The heavy cloth will last a long time before being replaced, but as all the liquors passing through are more or less acid, the cloths require an examination at every clean-up, to see that they have not been affected.

The filter plates and chambers are usually of cast iron, and before being put in use are given a couple of coats of acid-proof paint, all ports and inlets being likewise coated. This coat of paint requires renewal occasionally, depending on the character of the liquor passing through the press. Filter presses are now also made of wood, prepared as acid proof as possible. These presses where tried have given very good satisfaction, and are particularly acceptable where they have to be transported to a distance, being much lighter than the iron; also, the wooden plates can be made on the ground, and the press kept in repair for an indefinite length of time by the mill carpenter.

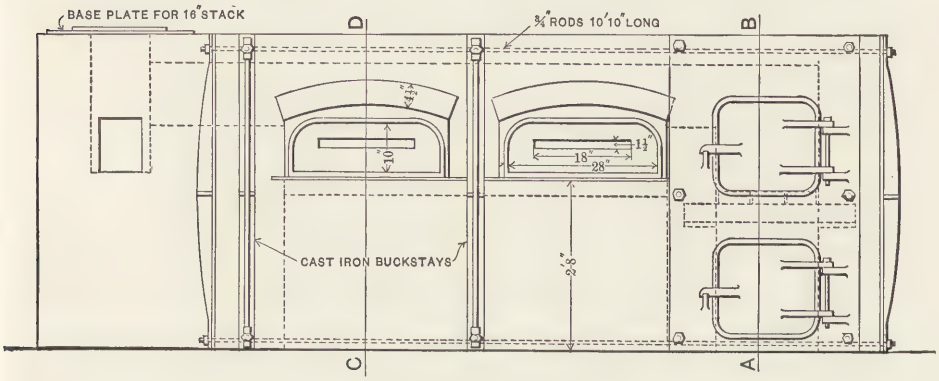
It is usual to run the filtrate from the presses into a shallow tank in the bottom of which there is a filter of tailings sand 6 or 8 in. deep. This sand is assayed from time to time and as its value increases the surface is scraped off, sent back to the roasting furnace, and a new layer of sand put on; if this filter is looked after there will be practically no loss of precipitate.

Sulphide Roasting Furnace.—From the filter press the sulphide cake and cloths are collected in a sheet-iron tray 44 in. long, 24 in. wide, and 4 in. deep, made of one piece of No. 8 iron with corners flanged, riveted, and calked water-tight, the tray having a couple of round iron handles riveted to each end. The tray when charged with sulphide is placed in the sulphide muffle-roasting furnace, the contained sulphur roasted off, and the cloths, if any, reduced to ashes.

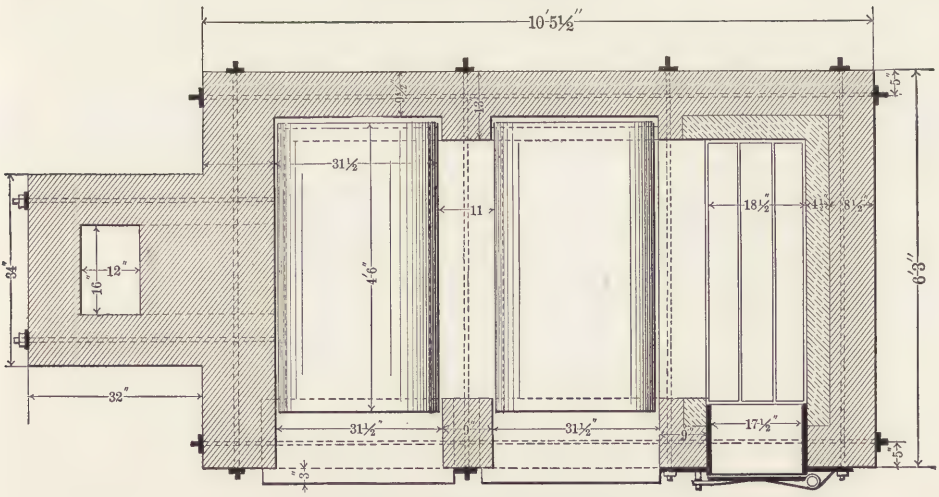
Fig. 9 shows a plan of this furnace of sufficient size for a mill of 100 tons daily capacity. It is built of ordinary red brick, with firebrick fire-box, and the whole is held together with buckstaves and tie-rods. The muffles are of cast iron, 1 in. thick, 4 ft. 6 in. long, 30 in. wide, and 12 in. high, outside measurements. These muffles rest on a hearth of brick. A bridge-wall 12 in. high protects the nearest muffle from the direct action of the fire, the flames passing over the top of each muffle, so that the roasting is done by reverberatory action, no flames or heat passing under the bottom of the muffle. In this way the sulphur is burned off from the top downward, doing away with all necessity of stirring, and the high heat that will distort and destroy the muffles rapidly, if fired from below, besides preventing the formation of sulphide of iron to any



SECTION THROUGH A-B SECTION THROUGH C-D



FRONT ELEVATION



PLAN OF MUFFLE FURNACE

FIG. 9.—SULPHIDE-MUFFLE ROASTING FURNACE.

extent, as the iron tray is not necessarily heated to a sufficient temperature to be attacked by the sulphur. The sulphur gases escape from the muffle through a slot in the back end into the flue in the same way as the oxide of lead from cupellation in the assay muffle. The heat required in the muffle is only a low red, and under ordinary circumstances a tray of sulphides will be thoroughly roasted in one hour. Stirring the sulphides during roasting is quite unnecessary, and only results in the formation of valuable dust that is carried out into the flue and stack.

After roasting and cooling, if proper attention has been paid to keeping the solutions clean in the precipitating tanks, the roasted precipitate will have a light yellowish-brown color, which, if examined under a good glass or microscope, will show the gold in small globules or sponge. The roasted precipitate should assay from 75 to 80% gold. In order to avoid many rehandlings of this high-grade material it is left in the tray in which it was roasted, weighed to ascertain the net weight, the weight of the tray having been taken beforehand, and the required amount of flux to form a liquid slag is added. The flux (borax, soda, and niter) is mixed in the tray and the whole transferred to a crucible of large capacity, fused in a good strong coke fire in the melting

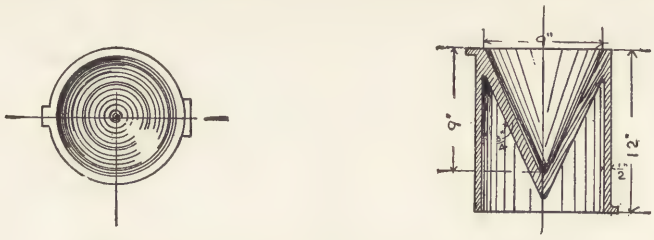


FIG. 10.—SLAG AND BULLION MOLD.

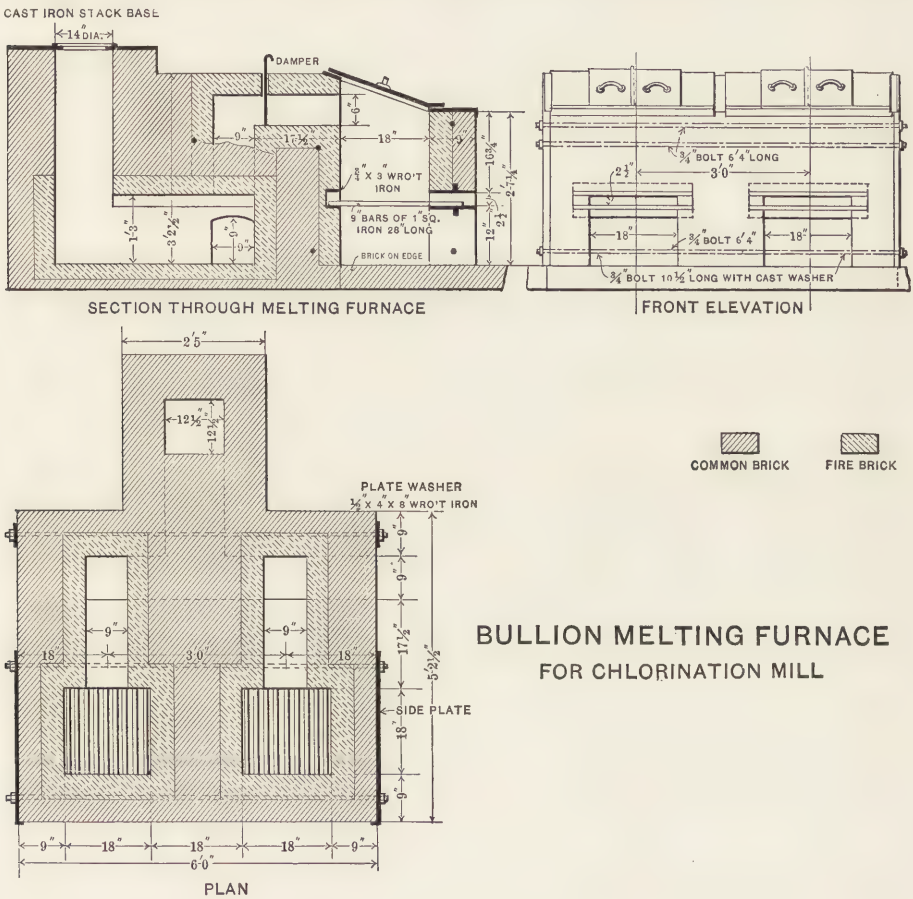
furnace, and when perfectly molten poured into a conical mold (Fig. 10). The gold sinks to the bottom, and when cold the mold is turned over, the gold and slag separating easily and cleanly. Fig. 11 shows a two-pot melting furnace, large enough to hold a No. 100 Dixon graphite crucible, this being the size most generally used.

In some works the practice is to add charge to the crucible during fusion, but when this is done great care is needed to prevent dusting, or considerable losses will be experienced. To overcome this loss as much as possible, the flux should be thoroughly dried before mixing with the gold powder and the mixture charged into the crucible through a funnel; the charging should always be done before the mixture in the crucible is fused down, and the crucible should not be over two-thirds full when finished.

Basket tongs, that fit the crucible snugly, are used to remove it from the furnace, a small block and tackle being required to lift it and hold it while being poured into the mold. The crucible after being emptied, and not required immediately for further use, should be turned bottom up on some dry place to cool off, which will keep the bottom from checking. A crucible should be thoroughly dried and annealed before using it the first time. After the first melting of all the gold powder is finished, the resulting buttons or

cones of bullion are cleaned of adhering slag, remelted with borax glass in a smaller crucible and cast into a bar. The mold should have rather flaring sides, and should be coated with tallow or chalk so that there will not be difficulty in getting the ingot out when cooled.

The metal when poured should be only a little hotter than its melting point, and in pouring some of the slag should be allowed to run into the mold first—enough to cover the bottom with a thin layer—before the metal flows in; in this way a neat, smooth, homogeneous ingot will be the result, while if the mold



BULLION MELTING FURNACE
FOR CHLORINATION MILL

FIG. 11.

is too hot, and the metal too hot, the resulting ingot is full of blow-holes, and usually very difficult to sample correctly.

If it is intended to refine the gold before sending it away the final melting should be done in a clay crucible, placed inside a graphite crucible, because of the destructive action on graphite of the niter used in refining. It is also necessary to use a small amount of quartz sand to protect the crucible from the corrosive action of the fused potash. If much impurity is present and it is desired to make very clean bars, the slag will have to be skimmed off several

times and fresh flux added; however, in general practice this is seldom done. The slag resulting from the fusion of the gold is saved, and when a quantity has been collected it is crushed to about a two-mesh size, mixed with about 1 lb. of litharge to 7 lbs. of slag, and smelted in an old crucible, the resulting lead being cupelled or sold to a refiner. The writer considers this method of treating the slags from bullion melting to be far superior to the one usually suggested, namely, crushing the slag fine and separating the prills of metal by panning, since there may be, in many cases, some imperfectly fused gold powder inclosed in the slag that would be entirely lost in wet concentration.

Too much attention and care cannot be given to the reduction of the sulphides into bullion, as it is at this point more than at any other that serious losses are likely to occur.

Future of Chlorination.—The greatest rival of the chlorination process is of course the cyanide process; however, there are many ores to be treated in which one is more applicable than the other. In the operation of either similar mechanical apparatus is required.

It has been questioned as to whether we are working in the right direction in using barrels in which the pulp and chemicals are agitated together to facilitate solution of the gold, the claim being made that tank-lixiviation with dilute chlorine solution for a sufficient length of time is as efficacious as agitation. It will be admitted that the chemical reactions are identical in each case, so that the mechanical conditions are the ones in which any difference should lie, which is practically a question of revolving barrels *vs.* large leaching tanks. As to the cost of installation for a given capacity by either system it will be found that this difference is inappreciable. The power required to rotate the barrels is very little more than that for rehandling the dilute solutions to be repassed through the leaching tanks.

Also, in the case of the charging this is a comparatively simple operation with barrels, since the ore is run in from a charging hopper, and requires practically no manual labor, while with large tanks it must be loaded into cars, trammed to the tanks, and distributed evenly over the whole surface, which must all be done by manual labor, unless a complicated system of mechanical distribution that requires power to operate is applied. Moreover, in the discharge of the tailings from barrels the whole charge is accomplished by removing the covers and revolving the barrel, while with tanks the charge has to be shoveled out, or where there is plenty of water can be sluiced out, either method requiring considerable manual labor. Another item of considerable importance, particularly in a field where there is considerable cold or inclement weather, is the difference in size of the buildings to cover the plants of similar capacities, and the extra cost to heat the larger tank building and keep piping, etc., from freezing. When everything is taken into consideration it will be found that barrels of reasonably large capacity are likely to be the most economical to operate. While none have been used yet to take a larger charge than 10 tons, I see no reason why charges of 25 tons cannot be as easily and successfully treated. There are certainly no metallurgical reasons to the contrary, and the mechanical difficulties are not insurmountable.

Chloridizing Roasting of Calcareous Silver Ore Containing Arsenic in Large Quantities.

By Ottokar Hofmann.

In December, 1886, I was engaged by the Anglo-Mexican Mining Company to investigate and improve the roasting of their ores at Yedras, Sinaloa, Mexico.

The ore is treated by the lixiviation process, and consists of argentiferous arsenical pyrites, fine-grained black zinc blende arsenical fahlerz, some iron pyrites, and occasionally ruby silver, while the gangue consists of silicious limestone and calcspar.*

I do not think that the chloridizing roasting of highly calcareous silver ores in combination with arsenical pyrites has often been the subject of a thorough investigation on a large scale, and an extract from my numerous experiments may be of interest and practical value.

Chloridizing roasting of complex ores is undoubtedly one of the most delicate of metallurgical operations. The treatment has to be modified in accordance with the character of the ore, and the character of an ore in relation to chloridizing roasting depends on the nature of the different sulphide minerals and the gangue accompanying them. The sulphide minerals should be classified as: (1) Those like iron and copper pyrites, gray copper ore, and silver copper glance, which in roasting form sulphates and decompose sodium chloride, liberating the chlorine; (2) those like galena and zinc blende, which form sulphates remaining indifferent; (3) antimonial and arsenical silver minerals, which form antimoniates and arseniates of silver.

The gangue remains either indifferent like quartz and porphyry, or it takes an active part like limestone, calcspar, manganese, and minerals containing magnesia.

If an ore consists of minerals of the first class together with an indifferent gangue, chloridizing roasting offers no difficulties, and a high chlorination can be obtained without much loss of silver by volatilization, and no special skill is required; neither does it matter whether the salt is added to the charge before entering the furnace or after it has been subjected to a partial oxidizing roasting.

The process of chloridizing roasting becomes more difficult if one or both minerals of the second class are present in large quantities, even if associated with an indifferent gangue.† With such ores the time the salt is added becomes very important. If added before the charge enters the furnace a very inferior chlorination is obtained, as is also the case if the salt is added before the oxidizing period has

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*The only analysis of the ore which I could obtain is the following, made in San Francisco several years before I came to Yedras:

	Per Cent.		Per Cent.
Silica	15.13	Zinc	4.92
Sulphur	13.31	Lead	1.78
Arsenic	9.82	Carbonate of lime	33.78
Iron	17.33	Magnesia	2.58
Alumina	1.35		

This analysis does not represent the average of the ore, of which I convinced myself by frequent concentration tests, in which I always found more gangue matters than the analysis shows. However, it may serve to give a general idea of the ore. During my experiments I felt the want of a chemical laboratory and chemist at the works.

†I made a thorough investigation of the chloridizing roasting of this class of ore, the results of which were published in the *Engineering and Mining Journal*, Feb. 9, 16, 23, March 9 and 16, 1889.

sufficiently advanced; moreover, the temperature and air supply require much attention.

The roasting is still more difficult if all three classes of ore are represented in connection with a gangue like limestone, which takes an active and injurious part in the operation. Such is the ore of Yedras, to the treatment of which this article refers.

The roasting facilities at the Yedras Mill consisted of four revolving Brückner cylinder furnaces, each 16 ft. long, and eight long reverberatory furnaces. The Brückners had been abandoned for several years, because former operators, so I was told, could not get satisfactory results with them. The chlorination was exceedingly low (40 to 50%), and the loss of silver by volatilization extremely high, while the main part of the roasted ore was rolled up into balls ranging from the size of an orange up to 15 in. and upward in diameter, the inside of which was not roasted. The subsequently-erected reverberatory furnaces also gave very poor results—65 to 70% chlorination with a loss of silver by volatilization of 20 to 35%, though fewer lumps were formed.

Roasting in the Brückner Furnaces.—I knew by experience that as a rule ores can be roasted with less loss of silver by volatilization in the Brückner than in the reverberatory furnace, and as the enormous loss of silver which the ore had sustained in the reverberatory was the most important question, I started the long-abandoned Brückner furnaces again. The previous failures with them I ascribed to the application of too high a temperature and an insufficient supply of air. The formation of balls I expected to diminish by judiciously regulating the heat and the revolving speed of the furnace, and if I succeeded in chloridizing well these lumps, their formation would not be a serious matter, for they could be pulverized in a ball-mill before charging the ore into the leaching vats.

My improvements on the Brückner cylinder, which consist of a fireplace and flue arrangement attached to each end of the furnace, enable me to apply the flame alternately through either end, and thus to use much longer cylinders, and provide fully for the free access of air between the fireplace and the throat of the furnace, which, strange to say, is not the case with the common Brückner furnaces. In a reverberatory furnace air can enter through the working doors; Stetefeldt provided his furnace with air-channels through which the required supply can be regulated; the Howell furnace is provided with an air-door, but the common Brückner furnace has no proper means of regulating the supply of air, and either the fire-door must remain open or only a very limited amount of air can enter the furnace. If the whole firebox is built of brick it is easy enough to make the proper change, but if it is made of boiler-iron lined with brick, it is not easy to make the necessary alterations in a remote mining camp. In this case, the fireboxes being of brick, I added 12 in. to their length, moved the grate bars toward the front, and inserted an air-channel behind the new fire-bridge. The fresh air entering through this chamber not only assisted in oxidizing the ore, but also aided the combustion of the fuel. The speed of the furnace, which had been 2½ revolutions per minute, was reduced to one revolution in 1¾ minutes, which materially reduced or altogether avoided the formation of large balls and diminished the quantity of dust, while on the other hand the speed was sufficient to expose every particle of the ore to the action of air and heat; in fact

a still slower speed, possibly even one revolution in 3 or 4 minutes, would have been preferable had it been obtainable.

The ore was crushed dry in the stamp battery with 7% of salt, and passed through a 24-mesh screen. The furnace charge was 4 to 4½ tons, and a strong fire was kept in order to quickly ignite the ore. In about an hour, and before the sulphur commenced to burn, heavy arsenic fumes were given off, entirely obscuring the interior of the furnace. After 1½ or 2 hours a sulphur flame could be observed to enter the flue, which was a sign that the sulphurets were sufficiently ignited to continue combustion without the aid of fire. The fire was then allowed to go out and the fire and air-doors were kept wide open. The temperature gradually increased by the combustion of the sulphur until it reached a certain maximum, at which it remained for several hours. After 2 or 3 hours the arsenic period was over, the heavy fumes disappeared, the interior of the furnace became clear, and the glow of the ore presented a beautiful pink color. No chemical loss of silver took place during the arsenic period. The oxidizing period continued for 3 to 4 hours more, the ore remaining at about the same temperature throughout, then fumes commenced to rise from the ore, and gradually increased, but never became so dense as during the arsenic period, though still dense enough to make the interior of the furnace invisible. The chloridizing period had commenced, and the odor of sulphurous acid and chlorine could be observed in the samples taken during the first period, but later the odor of sulphurous acid disappeared and only chlorine could be detected. In many instances, however, no smell of chlorine could be noticed during the whole time of roasting, as explained further on. Looking through the open fireplace, which by this time had cooled down, the whole interior seemed to be glowing, though the ore itself was not visible on account of the zinc fumes, which were illuminated by the glow of the red-hot ore and made a beautiful sight, and at the same time offered a good opportunity to observe the temperature in the furnace. So soon as a decrease in temperature was noticed, fire was started again and kept up for 3 to 4 hours, when the charge was considered finished. It took altogether from 12 to 14 hours to roast a charge.

During the oxidizing period the ore maintained an almost level position in the cylinder and had a liquid-like appearance. The ore particles on the surface, however, could be seen constantly moving; on the side where the furnace moves up fresh ore came to the surface as if emerging from a liquid, moved slowly across, and sank as soon as it touched the down-moving side of the cylinder. The ore increased considerably in volume during the first part of the chloridizing period, but after it had reached the maximum it commenced to shrink again, assuming a heavy sandy condition, where before it was loose and woolly, and finally occupied no more space in the furnace than the raw ore did. A short time before the second fumes commenced to rise the charge assumed a more inclined position, attaining nearly 45°.

There was a great and puzzling irregularity in the results. Sometimes a number of successive charges gave satisfactory results; then at once the chlorination dropped without any apparent cause, the ore, the amount of salt, the temperature and the treatment being unchanged. The only noticeable difference was the amount of silver contained in the ore, and as a rule the richer ore gave the better

results, though it was evident the quantity of silver as such could not influence the result materially. Close investigation finally showed the true cause of the trouble to be the more or less favorable proportion between the gangue (carbonate of lime) and the sulphureted matters in the ore, and this explanation corresponded with the fact noticed that richer ores usually gave better results than poorer ores. No marked difference between the two could be noticed in an inspection of the ore before it went into the battery.

The following tables of the results obtained with ores containing less and ores containing more carbonate of lime illustrate these differences:

ORE CONTAINING LESS CARBONATE OF LIME.

(7% salt mixed in the battery.)

No. of Charge.	Value of Raw Ore Per Ton.	Value of Roasted Ore Per Ton	Value of Leach Tailings Per Ton.	Chlorination.
	Ozs. Silver.	Ozs. Silver.	Ozs. Silver.	Per Cent.
65	72.12	76.12	16.20	79.6
66	(a)	64.86	12.60	80.5
67	65.64	64.80	9.00	86.2
68	68.04	68.46	10.56	84.6
69	66.30	72.16	13.68	81.1
70	66.78	73.62	12.36	86.0
71	68.16	71.16	11.88	83.3
72	66.24	68.64	12.90	81.3
73	68.52	73.82	14.70	80.0
74	66.00	75.24	11.16	85.4
75	64.38	66.42	12.06	81.9
76	63.78	66.30	9.86	85.9
77	68.28	68.10	11.42	83.3
78	58.50	61.02	10.80	82.4
	862.74	970.72	169.18	1161.5

(a) Sample of raw ore lost.

Average of raw ore .....66.36 ozs. per ton.
Average of roasted ore. .....69.33 ozs. per ton.
Average of leach tailings.....12.08 ozs. per ton.
Average of chlorination.....82.96%.
The roasted ore contained 2.97 ozs. silver more per ton than the raw ore.

ORE CONTAINING MORE CARBONATE OF LIME

(7% salt mixed in the battery.)

No. of Charge.	Value of Raw Ore Per Ton.	Value of Roasted Ore Per Ton	Value of Leach Tailings Per Ton.	Chlorination.
	Ozs. Silver.	Ozs. Silver.	Ozs. Silver.	Per Cent.
79	61.06	60.36	15.90	73.7
80	62.46	64.20	19.14	71.2
81	60.18	64.08	16.14	74.9
82	56.70	56.52	11.76	79.2
83	55.08	55.44	12.72	77.1
84	57.12	53.88	20.80	61.4
85	60.34	58.98	14.52	75.4
86	59.64	57.60	20.04	65.3
87	58.50	56.10	18.18	67.6
88	49.92	52.86	15.18	71.3
89	55.38	54.90	12.84	76.7
90	55.80	52.56	12.90	76.2
91	57.60	51.36	9.12	81.3
92	57.00	55.80	13.20	76.4
93	55.02	55.74	15.00	73.1
94	58.32	56.64	14.64	74.2
95	57.00	57.36	7.62	86.8
	977.12	964.38	249.70	1261.8

Average of raw ore.. .....57.47 ozs. per ton.
Average of roasted ore,.... . .....56.72 ozs. per ton.
Average of leach tailings..... ...14.68 ozs. per ton.
Average of chlorination..... ....74.22%.
The raw ore contained 0.75 oz. silver more per ton than the roasted ore.

Comparing the average results, we find them to be decidedly in favor of the higher sulphureted ore. To obtain further information I concentrated some of the ore, and made mixtures of certain proportions of concentrates and barren gangue, mostly carbonate of lime, and roasted these different mixtures, as well as the concentrates in the muffle, treating all samples alike, using 7% of salt, and roasting each half an hour. The concentrates used were obtained from ore which did not represent the average richness, assaying only 35.70 ozs. per ton, and therefore the different proportions I made were much poorer in silver than I afterward found corresponding ones in the bulk of the run. Only the concentrates were assayed, the values of the mixtures being calculated.

These tests showed that the silver-bearing minerals of this ore do not offer any difficulties to a good chloridizing roasting; on the contrary, they are easy to roast to a high percentage without showing any tendency to ball. The difficulties the ore offered were therefore caused by the gangue, and the great variation in the results were due to a greater or less favorable proportion of sulphureted matters to lime.

No. of Sample.	Concentrates.	Barren Gangue.	Value of Mixture. Per Ton.	Value of Leach Tails. Per Ton.	Chlorination.
	Per Cent.	Per Cent.	Ozs. Silver.	Ozs. Silver.	Per Cent.
1 (*a*)	100.0	.....	96.0	2.91	97.0
2 (*b*)	75.0	25.0	72.0	5.38	92.6
3 (*c*)	62.5	37.5	60.0	4.72	92.2
4 (*d*)	50.0	50.0	48.0	5.38	88.8
5 (*e*)	25.0	75.0	24.0	5.47	77.2

(*a*) Fuming profusely during chloridizing period; strong and pure smell of chlorine; a great deal of base-metal chlorides soluble in water were formed; no tendency shown to form lumps.
(*b*) During chloridizing heavy fumes; distinct but moderately strong smell of chlorine; no base-metal chlorides formed; no tendency shown to form lumps.
(*c*) Much less fumes; very little chlorine observable; none in water soluble base-metal chlorides; no tendency to form lumps.
(*d*) Very little fumes; no chlorine; no base-metal chlorides; showed tendency to form lumps.
(*e*) No fumes except at a high heat, and then but little; no chlorine; none in water soluble chlorides; showed much tendency to form lumps.

Carbonate of lime in presence of heated sulphureted minerals will change partly into calcium sulphate, which does not act on sodium chloride, and partly into caustic lime, which decomposes the metal sulphates and chlorides, and also, though less rapidly, the silver chloride. If, however, carbonate of lime is greatly in excess, only a very small amount, if any, of base-metal sulphates are formed to decompose sodium chloride, the greater part being changed directly into oxides. If no salt is present these sulphates are, of course, quickly changed into oxides. For this reason no chlorine can be detected if the salt is added after the ore has been subjected to a partial oxidizing roasting.

Some of the more calcareous charges were followed closely, but at no stage of the roasting could any soluble salts or chlorine be detected if the salt was added after the ore had been oxidizing for some time. If the salt was pulverized with the ore in the battery traces of such salts were found, but never in large quantities.

As there were neither sulphates, which at a high temperature decompose salt, nor chlorides, which at a high heat give off chlorine, an increased temperature during the last part of the roasting could not be of any benefit. In fact it had even a very bad effect. The caustic lime, which at a low temperature seemed to be comparatively indifferent to silver chloride, decomposed it energetically at a high temperature. The roasting of such ores had therefore to be finished at as low a temperature as possible, contrary to the usual practice. This fact must be taken into consideration in constructing long reverberatory furnaces. The arch of the first and second hearths nearest to the fire should be very high.

Charge No. 86 of the more calcareous ore was roasted for 8 hours without fire, then when the ore commenced to lose heat a second fire was applied for 3 hours. Before starting the second fire a sample was taken through the entire length of the furnace, and the chlorination was found to be 73.4%, and after 3 hours' second fire it was only 65.3%.

The moderately increased temperature at that stage of the roasting reduced the chlorination 8%. A number of such observations were made and finally the mode of roasting was changed, and the ore was allowed to roast entirely by the oxidation of the sulphureted matters without the application of a second fire.

Highly calcareous charges, containing only about 20 to 30% sulphurets, had a dead and sandy appearance, no chlorine could be noticed, and it required a high heat to make the ore fume. These fumes were light and were probably zinc

oxide. Not even at a very high heat could any chlorine be detected. If, however, a very strong draft of air was allowed to pass through the furnace from the beginning and during the whole time of roasting, some chlorine was generated and could be detected. The ore fumed moderately at a low temperature and 70 to 75% of the silver was rendered soluble in sodium hyposulphite. This indicated that by a rapid supply of air more sulphuric and less sulphurous acid was generated, and that some of the sodium chloride was decomposed by the former.

The beneficial effect of a great surplus of air was still more noticeable with high sulphureted ore in the reverberatory furnace. Differences as high as 25 to 30% in chlorination were observed between two charges of the same ore, which were treated alike, except that one furnace had excessive draft while the other had insufficient air. Silicious ores can be well chloridized with a moderate supply of air, while calcareous ores need an excess. This fact is of great importance, and due attention should be paid to it in constructing furnaces.

An ore which by hand concentration showed 40 to 45% sulphurets, generated considerable chlorine; it fumed more freely and at a lower temperature, and though no base-metal chlorides were formed, 80 to 86% of the silver was chloridized when temperature, time, and supply of air were properly regulated. Ore showing by hand concentration 50 to 55% of sulphurets permitted a chlorination of 90% and over.

In the Brückner furnace the ore behaved very differently, according as the salt was added in the battery or in the furnace, and it was therefore necessary to examine each case separately.

Adding the Salt in the Furnace.—If the ore was allowed to oxidize without salt a certain percentage of the silver would be rendered soluble in sodium hyposulphite but not in water, owing to the arsenic contained in the ore. This change takes place principally at the very beginning of roasting and at a very low temperature. A sample taken from charge No. 128 an hour after the combustion of the sulphurets had commenced, while the arsenic was still fuming strongly and oxidization had but just commenced, the appearance of the sample being still that of raw ore, yielded no less than 44.58% of the silver with sodium hyposulphite. Continuing the oxidation, the amount of soluble silver increased, but not so rapidly as in the first hour, as was shown by the following tests:

Charge No. 128, after 1 hour oxidizing roasting, yielded 44.58% of its silver with sodium hyposulphite.
Charge No. 128, after 1½ hours oxidizing roasting, yielded 43.44% of its silver with sodium hyposulphite.
Charge No. 128, after 2½ hours oxidizing roasting, yielded 45.32% of its silver with sodium hyposulphite.
Charge No. 128, after 3½ hours oxidizing roasting, yielded 57.66% of its silver with sodium hyposulphite.

No marked increase in soluble silver could be noticed by continuing the oxidation still further.

Charge No. 115, after 5½ hours oxidizing roasting, yielded 42.80% of its silver with sodium hyposulphite.
Charge No. 116, after 5 h. 20 m. oxidizing roasting, yielded 53.10% of its silver with sodium hyposulphite.
Charge No. 121, after 5 h. 25 m. oxidizing roasting, yielded 49.82% of its silver with sodium hyposulphite.

A certain percentage of silver can be extracted from some ores with sodium hyposulphite without roasting, and in order to determine whether this extractable form of silver was originally contained in the ore, or was formed during even such a slight oxidizing roasting as charge No. 128 showed, where in one hour

44.58% became extractable, a sample of raw ore was lixiviated, but not even a fraction of 1% of silver could be extracted. As this silver combination soluble in sodium hyposulphite was principally formed in the first stage of oxidizing roasting, during the time the arsenic was being expelled, it was undoubtedly silver arseniate.

When the salt was added at any time during the oxidizing period and well stirred in with a hoe, the percentage of soluble silver either increased suddenly a few per cent. or decreased, but in neither case could this percentage be maintained or increased. Decomposition commenced immediately and the percentage of extractable silver diminished rapidly, when after 2 to 4 hours, according to the amount of salt added, the maximum was reached.

In the following tables are given the detailed history of three charges, roasted with 10, 7, and 4% of salt respectively. The maximum of decomposition was reached in 2, 3, and 4 hours:

CHARGE NO. 116.

(10% salt added during roasting.)

Description. (*a*)	Value of Roasted Ore Containing Salt. Per Ton.	Value of Leach Tailings. Per Ton.	Soluble Silver.
	Ozs. Silver.	Ozs. Silver.	Per Cent.
Raw ore.....Ozs. per ton, 62.52	.....	.....	.....
After 5 hours and 20 minutes roasting and just before adding salt.....Ozs. per ton, 61.32	.....	.....	.....
The same calculated to contain 10% salt.....Ozs. per ton, 55.19	.....	28.80	53.1
Directly after adding salt and stirred.....	57.24	26.04	54.5
Twenty minutes after adding salt.....	62.04	31.92	48.6
One hour after adding salt.....	61.98	39.48	36.3
Two hours after adding salt.....	63.24	46.68	26.2
Three hours after adding salt.....	62.10	42.90	31.0
Four hours after adding salt.....	64.32	41.64	35.3
Five hours after adding salt.....	64.20	40.02	37.7

(*a*) After 5 hours and 20 minutes oxidizing roasting 10% salt was added and the charge stirred with hoes. The maximum of decomposition was reached in 2 hours after the salt was added, at which time 51.92% of the soluble silver was rendered insoluble. A decomposition of 10.82% took place in the first 20 minutes; in the next 40 minutes, 22.57%; in the next 60 minutes, 18.53%.

CHARGE NO. 123.

(7% salt added during roasting.)

Description. (*a*)	Value of Roasted Ore Containing Salt. Per Ton.	Value of Leach Tailings. Per Ton.	Soluble Silver.
	Ozs. Silver.	Ozs. Silver.	Per Cent.
Raw ore.....Ozs. per ton, 58.62	.....	.....	.....
After 4 hours roasting and just before adding salt—Ozs. per ton, 59.70	.....	21.36	64.27
One hour after adding salt.....	59.84	27.00	54.75
Two hours after adding salt.....	56.04	35.40	34.69
Three hours after adding salt.....	55.92	41.16	26.40
Four hours after adding salt.....	54.60	34.08	37.59
Five hours after adding salt.....	54.00	30.96	42.65
Six hours after adding salt.....	57.06	27.12	52.48
Seven hours after adding salt.....	50.52	23.10	54.28
Eight hours after adding salt.....	55.20	20.04	63.70

(*a*) The charge, subjected for 4 hours to oxidizing roasting before 7% salt was added, yielded 64.27% of the silver soluble in sodium hyposulphite. The maximum decomposition was reached in 3 hours after the salt was added, at which time 58.92% of the soluble silver was rendered insoluble. The decomposition during the first hour was 14.81%; the second hour, 31.21%; the third hour, 12.90%. During the next 5 hours we find the amount of soluble silver gradually increasing until at the end of the fifth hour nearly the same amount of silver was rendered soluble as the ore contained before the salt was added. This would indicate that by a continuation good results may finally be obtained, but the heat of the charge is exhausted before that time, and a second fire causes decomposition again.

CHARGE NO. 125.

(4% salt added during roasting.)

Description. (a)	Value of Roasted Ore Containing Salt. Per Ton.	Value of Leach Tailings. Per Ton.	Soluble Silver.
	Ozs. Silver.	Ozs. Silver.	Per Cent.
Raw ore.....Ozs. per ton, 59.70	.....	.....	.....
After 3 hours roasting and just before adding salt—Ozs. per ton, 58.32	.....	26.58	54.42
Just after adding salt and stirred	52.44	20.84	61.79
One hour after adding salt	56.16	23.94	57.38
Two hours after adding salt	56.10	29.40	47.60
Three hours after adding salt	54.90	42.24	23.07
Four hours after adding salt	58.56	46.80	20.09
Five hours after adding salt	56.50	39.66	29.25

(a) After 3 hours oxidizing roasting 54.42% of the silver in the ore was soluble in sodium hyposulphite, then 4% salt was added and mixed with the ore by hoes. A sample then taken through the entire length of the furnace showed 61.79% soluble silver, or a gain of 7.39% in these few minutes. Decomposition soon set in, however, and the maximum was reached in 4 hours after the salt was added, at which time 67.48% of the soluble silver were rendered insoluble. The decomposition during the first hour was 7.13%; the second hour, 15.83%; the third hour, 39.70%; the fourth hour, 4.82%.

Summing up the results of these experiments it was clear that (1) at a very early stage of oxidizing roasting quite a high percentage of the silver was converted into a combination (undoubtedly silver arseniate) which was soluble in sodium hyposulphite, and which seemed to resist well the decomposing action of the lime. (2) A part of this soluble silver was decomposed by the action of the salt, the decomposition commencing almost immediately after the addition of salt and continuing until a maximum was reached, when a reaction took place by which soluble silver was again formed—most likely silver chloride. (3) A larger percentage of salt produced the decomposition quicker, but not so thoroughly as a smaller percentage, viz.:

Charge No. 116, with 10% salt, reached the maximum in 2 hours, with 51.92% of soluble silver decomposed.
Charge No. 123, with 7% salt, reached the maximum in 3 hours, with 64.27% of soluble silver decomposed.
Charge No. 125, with 4% salt, reached the maximum in 4 hours, with 67.48% of soluble silver decomposed.

The singular fact was developed that when salt was added in the reverberatory furnace during the oxidizing period no such decomposition took place. Lixiviating tests made hourly and half-hourly after the salt was added showed a gradually increasing chlorination. This strange behavior of the ore in the Brückner seems to be a reaction of silver arseniate and sodium chloride, but why such a reaction does not take place in the reverberatory I cannot explain. No lumps or balls were formed in this mode of roasting, but the peculiar reaction prevented a higher chlorination and rendered this method of roasting impracticable.

Adding the Salt in the Battery.—Self-roasting.—When the salt is added in the battery it becomes thoroughly mixed with the ore, and both enter the furnace together. Though there is also a considerable part of the silver converted into a salt soluble in sodium hyposulphite at a very early stage of roasting, due to the arsenic in the ore, yet no decomposition of the arseniate of silver seems to take place. On the contrary, a gradual increase of soluble silver is observed up to a very advanced stage of roasting, when all the sulphureted minerals are converted into oxides. If the roasting is continued beyond this point, or if

the temperature is raised before or after this point is reached, then the caustic lime acts on the silver chloride and decomposes it.

When beginning the experiments I conducted the roasting in the Brückner the same as with certain highly sulphureted silicious ores, viz.: (1) By raising a strong heat to start combustion; (2) oxidizing without fire until the temperature commenced to decrease; (3) chloridizing and finishing at a higher temperature with a second fire. In the course of the experiments, however, the employment of the second fire caused the decomposition of a considerable part of the soluble silver already formed. A charge was therefore roasted without using a second fire, leaving the ore to complete the roasting in the heat created by the combustion of the sulphurets, and much better results were attained. For the sake of comparison I roasted and finished a number of charges without the additional fire and then a number with the second fire, using ore of the same grade and maintaining similarity in all other conditions, and found that with self-roasting the average chlorination was 5% higher, while the consumption of wood was reduced more than one-half. With a second fire the average value of the roasted ore was 0.46 oz. silver less than the value of the corresponding raw ore, while in self-roasting the average value of the roasted ore was 1.91 ozs. silver higher than that of the raw ore, indicating smaller loss by volatilization. I call this mode of roasting "self-roasting," because the ore, once ignited, requires no further attention, and is left entirely to itself until the heat has nearly died out. Self-roasting simplifies the manipulation. It is only necessary to maintain fire for about two hours, when the charge can be left to itself until the time of discharging, which is about an hour before the red heat dies out. Thus one man can attend to quite a number of furnaces. Care has to be taken, however, to make large charges, as small charges do not maintain the heat long enough, and they "freeze" before the roasting is completed.

Having obtained by this mode of roasting an average of 82.9% chlorination, with occasional results of 85 and 86% (see Table) with calcareous ore which offered so many difficulties to chloridizing roasting, it seems that it may be possible to properly chloridize nearly all kinds of highly sulphureted ores in this way, but, as large charges work better than small ones, it is advisable to have the revolving cylinders large enough to hold 5 or 6 tons of ore.

By roasting in this manner more subchlorides are formed, and less volatile chlorides expelled; this, however, does not interfere much with the subsequent lixiviation. The advantages gained will more than overbalance the slight extra expense caused by the increased consumption of sulphur, and the refining of a somewhat baser precipitate, especially if the diminished loss of silver by volatilization, which loss is principally caused by the expulsion of the volatile base-metal chlorides, is taken into consideration. If the ore contains copper, an increased formation of cuprous chloride will even be beneficial for the subsequent extraction by lixiviation.

The consumption of wood in self-roasting was found to be only one cord for each 10.6 tons of ore, while in roasting with a second fire only 4 to 4.5 tons could be roasted with a cord of wood.

Balling of the Ore.—Another noticeable difference in the behavior of the ore, due to the addition of the salt in the battery or in the furnace, was the formation

of balls or lumps in the first case, while in the latter instance the ore remained loose without forming balls, and when discharged ran on the cooling floor like water. When the salt was added in the battery it assumed a more solid form, did not spread over the cooling floor, and did not dust, but contained a great many balls. These balls originated during the early part of the chloridizing period and at first were of the size of a pin's head. Gradually they assumed larger dimensions, and when the charge was finished the majority of them were from the size of a pea to that of a walnut. They were smooth, hard, and heavy, consisting of concentric shells, and were formed even if a second fire was not used. By reducing the speed of the furnace from $2\frac{1}{2}$ revolutions per minute to one revolution in $1\frac{3}{4}$ minutes, and by adopting self-roasting, which was equivalent to roasting at the lowest possible heat, the number and size of these hard balls were greatly reduced, but were not altogether prevented.

Roasting during the chloridizing period was tried with an intermittent motion. The furnace was allowed to make one revolution and was then stopped for 15 minutes, when another revolution was made, and so on until the charge was finished. This reduced but did not prevent the formation of balls. Then the furnace was stopped entirely during the chloridizing period, intending to allow the completion of the roasting process without any further movement, though it extended the time required for roasting. This method would very likely have had the desired effect had not another difficulty made it impracticable. The surface of the ore commenced to harden, forming a crust, which increased in thickness, was spongy and porous, and would not have interfered with the lixiviation, but threatened that by the time the charge was finished the whole mass might have hardened, which would have caused great difficulty in discharging, and would also have endangered the furnace through the whole mass clinging to one side and then dropping suddenly as the furnace revolved.

Though annoying, this formation of hard balls would not be a serious obstacle, as by repeated tests it was found that they were well roasted; in fact, always a slight percentage better chloridized than the fine stuff. By separating the coarse from the fine and crushing the former in a ball-mill or through rolls, the ore would be well prepared for lixiviation. This extra handling does not signify much when the great difference in expense between roasting in a Brückner and roasting in a reverberatory furnace is taken into consideration, to say nothing of other advantages. Other lumps were formed toward the end of the operation which were larger, but soft and porous, had a rough surface, fell apart when brought in contact with water, and did not interfere with lixiviation.

The sulphureted minerals of this ore had no tendency to bake and form lumps. When the concentrates, free from gangue, were roasted, the pulp remained perfectly loose and sandy, even if only occasionally stirred during roasting; the formation of these small hard balls must therefore have been caused by the gangue. They were soft while hot and were readily crushed, but when cold became hard and brittle, and when broken showed concentric layers. They were not caused by excessive heat, because they formed even when the heat was kept as low as the combustion of the sulphureted minerals permitted; neither was their appearance that of overheated ore. They were probably caused by the formation of a double salt of calcium sulphate and sodium sulphate (glauberite), $Na_2SO_4+CaSO_4$,

a salt which fuses easily. This would explain the singular fact that no lumps were formed if the salt was added in the furnace after the charge had been oxidizing for some time; in which case the salt could not be thoroughly mixed with the ore, and therefore did not come in such close contact with the sulphureted matters as the lime did, on which sulphuric acid acts so much more energetically than on sodium chloride that very little, if any, of the sodium chloride was converted into sulphate. For the same reason free chlorine could be detected only in exceptional cases if the salt was added in the furnace. Neither was it possible to produce a chlorination of the silver. The 40 to 60% soluble silver was arseniate of silver.

In my muffle experiments with different mixtures of concentrates and gangue (limestone) the tendency to form lumps commenced with the proportion of 50 concentrates to 50 gangue, and this tendency increased with the percentage of gangue; it did not exist if the mixture contained more sulphurets than gangue, and if the ore had been assorted at the mine to meet this requirement the formation of balls would doubtless have been entirely avoided, the chlorination would have been much better, and the Brückner furnaces could have been used instead of the more expensive reverberatories. It may be mentioned that the ore from the mine had to pass successively through three large ore-bins before reaching the mill. These were always kept full, in order to have ore in reserve, and it took 10 to 14 days from the time the ore was dumped into the bin at the mine before it reached the battery; quick changes for experimental purposes were therefore impossible.

Roasting in the Reverberatory Furnaces.—There were in all 8 reverberatory furnaces in operation, of the following dimensions:

Four furnaces, 50 ft. long by 10 ft. wide, containing 5 hearths each.
Two furnaces, 40 ft. long by 10 ft. wide, containing 4 hearths each.
Two furnaces, 30 ft. long by 10 ft. wide, containing 3 hearths each.

The arch at the highest place near the fire was 27 in. above the hearth, farther away only 20 in., and at the last hearth 18 in. The sides were 10 in. high. The furnaces were built in pairs, placed back to back, and each hearth had but one small working-door 8 by 12 in.

While tolerably good chlorination may be obtained with silicious silver ore in furnaces of the above-described construction, they surely were not of suitable design for the roasting of calcareous silver ores, as in the first place insufficient provision was made for a free and well-located air inlet; the small working-door furnished air only for the ore near it, which fumed, while further in and toward the back of the furnace it presented the appearance of an inactive glowing mass of high temperature, and did not emit any visible fumes. In the second place, the arch, especially the one over the hearth nearest to the fire, was not high enough to keep the temperature low enough for calcareous ores during the chloridizing period. There was no remedy for these defects except reconstruction. By keeping the fire-door continually wide open much better results were obtained, but they were not satisfactory until the furnaces were reconstructed.

Our reverberatory roasting furnaces of standard construction do not comply sufficiently with the required conditions for chloridizing roasting the majority

of the ores, which are treated by lixiviation or amalgamation. In the endeavor to reduce the consumption of fuel to a minimum, other and more important points have been neglected or sacrificed to this one object. As a rule we find the arch and sides too low, the fire-bridge too low and narrow, the working-doors and flues too small, and no extra provision made for the access of air to dead places in the furnace. The flame being pressed down by the low arch comes in contact with the ore and therefore exercises a reducing action, while the space between the ore and the roof is almost completely filled with gases from the combustion of the fuel, and the fresh air has no opportunity to enter deeper into the furnace. The little that passes in through the working-doors is forced to the side, doing some good work on its way to the flue, but the main portion of the ore depends for its oxidization on the small volume of undecomposed air mingled with the fire gases. Thus a furnace may have plenty of draft but not enough air. A low and narrow fire-bridge causes that part of the charge nearest it to become intensely heated, and to require more attention and give more trouble than the larger balance of the charge. It also necessitates the changing of the charge from the farther end to the fire, and from the fire to the farther end, which in a long furnace of proper construction is not required. These defects are felt still more if in a long furnace an ore has to be roasted which cannot stand a high heat without baking or sustaining a great loss of silver, because it is not possible to keep the heat in the finishing hearth as low as required and at the same time insure the working of the more remote hearths to the best advantage.

For highly sulphureted ores I make the arch of the first few hearths, on which the chemical reactions principally take place, not less than 3 ft. high, the sides 16 to 20 in. and the fire-bridge not less than 18 in. high and 18 in. thick. The arch of the most remote hearths, where the ore is merely heated, I make 20 to 27 in. high. The arch of the finishing hearth can even be made higher if the nature of the ore requires it.

An air-channel is constructed through the fire-bridge with openings leading into the furnace; this keeps the bridge cooler and brings air to the otherwise dead spaces of the furnace. If the back of the furnace does not contain working doors, it ought to be provided with air-holes 4 ft. apart; or if this is not possible, on account of two furnaces being built back to back, then air-ducts should be constructed under the hearth, leading from the front to the back of the furnace.

In a furnace of such construction the flame does not come in contact with the ore, but keeps to the arch and leaves sufficient room for the air to enter between it and the ore. There are no dead spaces, the ore fumes over the whole surface, and each hearth has a uniform heat; the roasting is much better and more quickly performed, the formation of lumps is greatly reduced, and at the same time larger charges can be attended to more thoroughly and easier.

I found that through insufficient supply of air not only the chlorination suffers, but the loss of silver also increases. Where ores which contain considerable lead have to be roasted for amalgamation, a high-arched furnace offers an additional advantage, for in it nearly all the lead is converted into a sulphate which is not decomposed by mercury, and therefore does not amalgamate, while in a low-arched furnace a great deal of lead chloride is formed, which does amalgamate and cause very dirty amalgam and base bullion. Mr. G. Kustel, in

working the ore of Plomoso, Mexico, obtained by amalgamation from ore roasted under a 20-in. arch bullion only 300 to 400 fine, while when the same ore was roasted under a 30-in. arch the resulting bullion was over 900 fine. I found high-arched furnaces provided with the necessary air-channels also of great advantage in roasting gold-bearing sulphurets for the Plattner process, much less magnetic iron being formed and a perfect oxidation attained in a far shorter time.

At Yedras I was astonished to find that the ore in the reverberatory funaces on the finishing hearth was subjected to almost a white heat, while care was taken to avoid cooling the furnace by excluding the air as much as possible. The fire-door was kept closed and the very small working-doors, of which there was one on each hearth, were closed as soon as stirring was completed. One of the charges roasted in this way consisted of one ton of ore; four men were attending the furnace, which was 50 ft. long, and 5% salt was added on the third hearth after the ore had been nearly 5 hours in the furnace. The following is a detailed record of same:

1. Raw ore without salt. Sample of ore taken from the hopper before charging gave 67.08 ozs. silver per ton.

2. Same charge. Sample taken from the first hearth after the ore had been 2 hours in the furnace, and just before the charge was moved to the second hearth, the hearth being dark, gave off a pretty strong smell of sulphurous acid and contained 65.40 ozs. silver per ton of ore.

3. Sample taken from the second hearth, after having been there for 2 hours and 20 minutes, and just before the charge was to be moved to the third hearth. Charge dark-red, strong fume (arsenic period), with strong smell of sulphurous acid, gave 66.06 ozs. silver per ton of ore.

4. Sample taken from the third hearth just before the salt was added. Charge red-hot, strong smell of sulphurous acid, gave:

Ore.....67.92 ozs.
Leach tailings.....31.44 ozs. } 53.08% silver soluble in sodium hyposulphite.

5. Sample taken from third hearth after having roasted just 1 hour with salt. Strong fumes and smell of sulphurous acid, color after cooling, light-brown, gave:

Ore.....63.60 ozs.
Leach tailings.....29.04 ozs. } 54.4% silver soluble in sodium hyposulphite.

6. Sample taken from fourth hearth after the charge had been there 1 hour, and had roasted 2 hours with salt. The ore woolly and sample exposed to air fumed strongly; color, after cooling, red-brown; temperature red, gave:

Ore.....60.72 ozs.
Leach tailings.....28.44 ozs. } 53% silver soluble in sodium hyposulphite.

7. Sample taken while charge was being moved to fifth hearth and after roasting 3 hours and 20 minutes with salt. Temperature light red, sample fumed strongly when exposed to air; inside of the furnace clear, fumed only near the working door; slight smell of chlorine and sulphurous acid; ore commenced to assume a sandy consistency, and had the appearance of an overheated ore; color after cooling red-brown, gave:

Ore.....70.98 ozs.
Leach tailings.....27.12 ozs. } 61.8% silver soluble in sodium hyposulphite.

8. Sample taken from fifth hearth after roasting 5 hours with salt. No smell of sulphurous acid, very little of chlorine; temperature very light red, almost white; ore sandy; color after cooling, greenish-brown, gave:

Ore.....68.94 ozs.
Leach tailings.....23.94 ozs. } 65.2% silver soluble in sodium hyposulphite.

9. Sample taken when the ore was discharged, after having been 11 hours in the furnace and 6 hours roasted with salt. Very little smell of chlorine, none of sulphurous acid; ore very sandy, mixed with lumps; color after cooling greenish-brown, gave:

Ore.....65.40 ozs.
Leach tailings.....21.06 ozs. } 67.2% silver soluble in sodium hyposulphite.

The ore sustained during roasting a loss in weight by volatilization of 13.8%, and the loss of silver by volatilization amounted to 16%.

By comparing the different samples it was found that at the time the salt was added there was 53.8% of the silver soluble in sodium hyposulphite. This soluble silver was an arseniate and was formed in the same manner as that obtained in the Brückner furnace during the arsenic period. During the next two hours of roasting with salt there was no change in the soluble silver. This is unlike the behavior of the ore in the Brückner, in which during this period a marked decomposition of the soluble silver took place. Although in the reverberatory the salt was also added after the ores had been oxidizing in the furnace for nearly 5 hours, and the arsenic period was over, no soluble silver was decomposed by the salt. During the next two hours it remained the same, and commenced to increase only after the third hour with salt, and after the ore had reached the region of light-red heat on the fourth hearth. The increase, however, amounted to only 8%, and on the finishing hearth, where the ore was exposed to an almost white heat for 2 hours and 40 minutes, a further increase of 5.4% occurred—in all 13.4%—while the loss of silver by volatilization, which took place principally during this period, amounted to 16%. Of the 67.2% of silver rendered soluble in sodium hyposulphite, 53.8% was due to the action of the arsenic, and only 13.4% to the action of the salt. To gain 13.4% chlorination, 16% of silver was sacrificed, a very thoughtless operation, and this was by no means one of the worst results. With most of the charges 25 to 30%, and even more, of the silver was lost, to gain a small percentage in chlorination, as is shown further on.

The low chlorination was due to the insufficient supply of air, which prevented the formation of sulphuric acid to satisfy the lime and to act on the salt, while the great loss of silver was caused by excessive heat and insufficient air. The different behavior of the ore in the two furnaces as regards the decomposition of the soluble silver after the salt was added is remarkable, and thus far unexplainable.

While experimenting with the Brückner furnaces, the roasting in the reverberatory furnaces was conducted in the manner just described, and the experiments with these commenced after those with the Brückners were completed. One preliminary experiment, however, was made, to demonstrate to the metallurgist in charge the effect of the air, for he insisted that the Yedras ore could only be chloridized at a very light-red heat, and he excluded the air as far as possible, so as not to cool the furnace.

As mentioned above, the furnaces were not of proper construction, and the

only means of getting more air into them was by leaving the fire-door wide open. The beneficial effect of the air was quite striking, though no change was made in the temperature, the roasting proceeding at the same high heat as before. The interesting record is given in the following tables:

ROASTING WITH 5% SALT AT A VERY HIGH HEAT WITH CLOSED FIRE-DOOR.

Reverberatory Furnace, No. 3.

Date.	Value of Raw Ore Per Ton. (*a*)	Value of Roasted Ore Per Ton.	Soluble Silver.	Loss of Silver by Volatilization.
1887.	Ozs. Silver.	Ozs. Silver.	Per Cent.	Per Cent.
January 13	63.03	56.52	67.7	17.8
14	62.04	50.04	69.6	25.5
15	63.78	49.44	57.1	27.2
16	63.78	52.80	49.8	25.4
17	64.08	60.00	43.0	13.9
18	61.20	49.08	43.2	26.3
19	60.60	45.00	63.2	31.7
20	61.56	49.50	68.8	26.1
21	60.00	48.48	68.1	25.7
22	58.20	45.48	76.8	27.8
Average	61.82	50.63	60.7	24.7

(*a*) Difference of value between raw and roasted ore, 11.9 ozs. per ton.

ROASTING WITH 5% SALT AT A VERY HIGH HEAT WITH OPEN FIRE-DOOR.

Reverberatory Furnace, No. 3.

Date.	Value of Raw Ore Per Ton. (*a*)	Value of Roasted Ore Per Ton.	Soluble Silver.	Loss of Silver by Volatilization.
1887.	Ozs. Silver.	Ozs. Silver.	Per Cent.	Per Cent.
January 23	57.66	49.56	85.3	20.6
24	55.50	47.04	79.7	21.6
25	54.36	50.40	79.6	14.3
26	63.30	55.20	83.9	18.0
27	63.84	59.22	77.7	13.9
28	64.86	61.08	73.5	12.6
29	65.52	57.00	65.5	19.2
30	66.78	59.40	77.2	17.4
31	66.00	58.80	74.5	17.3
February 1	62.10	55.68	79.6	16.7
2	57.30	53.40	75.7	13.7
3	54.54	48.96	76.2	16.6
4	55.20	52.80	77.5	11.2
5	57.60	51.96	75.8	12.0
6	60.66	55.14	80.5	15.6
7	61.68	53.70	72.1	19.2
8	64.50	65.64	68.9	5.5
9	55.80	55.56	75.5	7.6
10	57.96	48.48	80.7	22.3
11	62.46	56.52	76.8	16.0
12	67.02	59.94	69.5	10.7
Average	60.70	55.02	76.4	15.3

(*a*) Difference of value between raw and roasted ore, 5.68 ozs. per ton.

By comparing the two tables we find that from the very first day the fire-door was kept open a marked improvement in the results took place. The averages show this plainly:

	Per Cent.
Average loss of silver—fire-door closed	24.7
Average loss of silver—fire-door open	15.3
Difference in favor of open fire-door	9.4

	Per Cent.
Average of soluble silver—fire-door open	76.4
Average of soluble silver—fire-door closed	60.7
Difference in favor of open fire-door	15.7

Thus a reduction of 9.4% in the loss of silver and an increase of 15.7% in soluble silver, or a total saving of 25.1% of the silver in the ore was effected by simply admitting more air into the furnace.

Preparatory to my experiments with the reverberatory furnaces I reconstructed some of them, to adapt them to the requirements of the ore. The arch was raised to 3 ft. with the exception of the hearth furthest from the fire on which the ore was charged and which was much lower. The working-doors and the flue were enlarged and air-channels constructed to permit air to enter along the fire-bridge and along the back-wall of the furnace.

Roasting was then commenced at a much lower temperature, and with a more liberal supply of air, much better results being attained. The average chlorination of one month rose to 81.7%, while the loss of silver was reduced to 1.7%. In the new furnaces the ore fumed over the whole hearth instead of only near the working-doors as before, and the temperature was much more uniform in all parts of each hearth. To roast at a low heat requires much greater attention and skill than at a high heat, and it took some time before the men became accustomed to it.

After numerous experiments to determine the proper temperature and draft, the following rules were adopted and will apply to all calcareous sulphureted silver ores:

1. The fumes evolved must be kept in motion in all parts of the furnace. If they stagnate around the ore, or if the furnace assumes nearly a uniform heat throughout its entire length, it is always a sign of insufficient draft, and if the draft is not increased the result will invariably be a high loss of silver and a low chlorination.

2. In no part of the furnace should the ore attain a light-red heat. The fire should be regulated entirely according to the temperature required at the finishing hearth nearest the fire, where the temperature should be kept so that the ore has rather a dark surface if not stirred, but a dark-red heat when the hoe enters it. The ore, however, should fume, and the temperature never go below the point required to evolve fumes. The presence of fumes always indicates that the temperature is not too low, while the dark surface of the ore shows that the temperature is not too high.

An arch still higher than 3 ft. facilitates the maintenance of the conditions prescribed in the second rule, and at the same time permits a stronger fire for the more remote hearths without injuring the charge on the finishing hearth. I rebuilt furnaces Nos. 3 and 4 and made the arch of the finishing hearth 5 ft. high, the sides 3 ft. 4 in., and the fire-bridge 2 ft. high; the next two hearths were raised so that the arch was only 3 ft. high, the sides 16 in.; the next arch 27 in. high, sides 16 in., and the charging-hearth arch 24 in. high and sides 14 in. These gave the best and most uniform results of any of the furnaces; the proper temperature on the finishing hearth being maintained much easier, because the flame following the roof was so far above the ore that it would have taken an excessive fire to overheat the charge. The chlorination obtained was very satisfactory, and the loss of silver by volatilization was reduced to a minimum. The average results of six weeks' working were: Chlorination, 83.8%; loss of silver, 0.8%.

The consumption of wood in these furnaces was larger than in the others; in the reverberatories with 3-ft. arch it amounted to 0.098 cord per ton of ore roasted, and in the furnaces with a 5-ft. high arch 0.32 of a cord more per ton of ore was used, but the better results, and the ease with which the temperature could be controlled, overbalanced the extra consumption of wood.

When the roasting was properly conducted the percentage of chlorination invariably depended, as in the Brückner furnaces, upon the proportion of sulphureted minerals and calcareous gangue of which the ore was composed. The more sulphureted matters the ore contained the better was the result. But little difference was noticed in the formation of lumps whether the salt was added in the battery or in the furnace; in both cases lumps were formed, but as a rule they were more porous and softer than those of the Brückner, and could easily be leached.

Loss of Silver by Volatilization.—Silver chloride as such is not volatile, but it is influenced by the volatilization of other chlorides and becomes volatile. A high heat therefore indirectly causes a larger loss of silver by the expulsion of larger quantities of volatile chlorides. Other conditions being equal, we shall always find the loss of silver to be in direct proportion to the chemical loss the ore sustains in weight. In other words, the charge which during roasting sustained the least chemical loss in weight sustained also the least loss of silver by volatilization. While the operator endeavors to produce a high silver chlorination he should at the same time be careful to expel as little of the volatile chlorides as possible. He will be greatly assisted in this endeavor by keeping the ore in a thick layer, and by using low heat and plenty of air. If a small charge is thinly spread over a large hearth more volatile chlorides will be expelled and the ore will lose more in weight and in silver than when a larger charge is roasted in the same furnace. This is the reason why, as a rule, the loss in weight and in silver in a large Brückner furnace in which the ore lays 2 ft. thick is less than in a reverberatory.

It will be found that ore roasted at a low heat with sufficient air will lose less in weight, because a large part of the volatile chlorides which at a higher heat would be expelled will then remain in the ore. For amalgamation it is desirable, in fact necessary, to expel the volatile chlorides as much as possible, because they take an active part in amalgamation and make the quicksilver smeary and inactive, causing a poor silver extraction and a very low-grade bullion. These chlorides, however, do not seriously interfere in the lixiviation process, especially if trough-lixiviation is employed, because by this method the proportion of ore and water can be so regulated that the resulting chloride solution is so dilute as not to dissolve any silver.

It is one of the principal advantages of lixiviation over amalgamation that in it the volatile chlorides do not need to be expelled, and therefore the roasting of most ores can be conducted with only a nominal loss of silver by volatilization. In metallurgical works we always find the great loss of silver pointed out as an objection to all processes which require chloridizing roasting of the ore. Formerly chloridizing roasting was principally used and studied in relation to amalgamation; little or no attention was paid to roasting for lixiviation, or to the fact that this process allowed a modification of roasting by which its objectionable

features could be obviated. I made chloridizing roasting the subject of special study, and found that it could be conducted with just as little loss of silver as oxidizing roasting, if care was taken to expel as little as possible of the volatile chlorides. The chemical reactions between salt and the sulphates take place at a very low heat, in fact at a lower heat than is generally believed, while on the other hand it takes quite a high heat to thoroughly expel the volatile chloride; therefore in roasting for lixiviation the temperature can be kept as low as the nature of the ore permits during oxidizing, and very low also during chloridizing, and still prepare the ore well for the subsequent extraction of the silver. During chloridizing the ore ought to be kept in a thicker layer and stirred only at intervals to diminish the volatilization of the chlorides.

The calcareous ore which was treated in the Brückner cylinders by self-roasting lost only 3.5% of its weight and 1.8% of its silver, while the same ore, roasted in the reverberatory furnaces of old construction with insufficient supply of air, lost 7 to 13% in weight and 15 to 25% or more of silver.

In order to roast skillfully it is of great importance to frequently ascertain the loss of silver by volatilization, but to do this it is necessary to know the loss in weight the ore sustains during roasting. This, however, is accompanied with great difficulty if it is done in the old way by actual weighing of the charge before and after roasting, necessitating the careful cleaning of the furnace and the dust chambers before and after the process. In many cases this is not possible without seriously interfering with the regular work, and at all events it is accompanied with so much trouble and expense that if the shrinkage in weight is once ascertained this figure is used in all subsequent calculations, though the conditions under which the roasting is performed, such as heat and draft, or the character of the ore, may have changed. That such figures are not very reliable will be readily understood, but still more incorrect is the method some adopt of roasting 10 or 20 gms. in the muffle and then taking the difference before and after roasting as the loss in weight; by this means the loss of that particular sample in the muffle is ascertained, but not the loss the ore would lose in the furnace. To properly conduct the roasting it is not of great importance to know how much the ore loses by dusting, for this is merely a mechanical loss, and the fine ore particles are easily collected in dust chambers. The loss due to the volatilization of the chlorides is the serious one. These fumes are often richer in silver than the ore, are very difficult to collect, and easily escape. We have therefore to find how much the ore loses in weight by volatilization in order to obtain a correct basis for a calculation of the loss of silver in roasting.

I adopted the following method, which I published in the *Engineering and Mining Journal*, April 3, 1887, and which gives sufficiently correct results for practical purposes, and can be performed in the assay office in a few hours: Ten gms. of the raw pulp, containing salt, are placed in a roasting dish, and roasted in the muffle for half an hour or an hour, then the sample is removed from the muffle, allowed to cool, weighed, returned to the muffle, roasted again for half an hour, and then weighed again. This is repeated until two weighings are alike, or until in the last half-hour the ore does not lose more than 2 or 3 mgms.; then the difference between the original weight and that of the last weighing, expressed in percentage, gives the highest possible loss the raw ore can sustain.

Ten gms. of a sample of roasted ore, corresponding with the sample of raw pulp, are placed in a roasting dish, and also roasted in the muffle until two weighings agree, or the difference between two consecutive weighings is not more than 2 or 3 mgms. The difference between the first weighing (10 gms.) and the last, expressed in percentage, gives the weight which the roasted ore is still capable of losing if subjected to prolonged roasting. If we deduct, therefore, the loss of which it is capable from the highest possible loss, we obtain in percentage the loss in weight the ore has suffered during roasting by volatilization.

Conclusions.—My experiments in roasting the calcareous and arsenical ore of Yedras have shown:

1. That the main difficulty in chloridizing the ore to a high percentage was caused by the excess of lime in the ore. The ore at Yedras was assorted, as in all silver mines, with the view to obtaining a certain grade in silver, and no attention was paid to the relative proportion of lime and sulphureted minerals, which was of so great importance. In order to obtain regular chlorinations of over 90% it was necessary that the ore should not contain less than 50% of sulphurets, but to accomplish this it was necessary to break the ore smaller, so that more of the barren limestone and calcspar could be thrown out, and enable the sorters to pick out a poor grade of second-class ore. Pieces containing much iron pyrites, even if they did not contain silver, should be thrown with the first-class ore.

The accumulation of a second-class ore was not very desirable at Yedras, because it could only be treated by concentration, and the water supply was very limited, except during the rainy season. However, I consider it more rational to accumulate a second-class ore dump at the mine for future treatment than a rich tailings pile at the mill, because it is very difficult to extract the silver from tailings, while the chances are that after further development the mine may produce more water; but even if it should not, the accumulation of second-class ore would not be more than could be concentrated during the rainy season. This question is a very important one and due attention should be paid to it. The outlay would be comparatively small compared with the large amount of silver lost in the tailings.

By improvements in the mode of roasting I succeeded in increasing the chlorination from 65 to 81% and above, and in reducing the loss of silver from 25 to 1.7% and less, thereby nearly doubling the production. If the ore had been properly sorted the increase would have been at least 10% more.

2. I proved that such ores could be roasted with a trifling loss of silver by volatilization if the roasting was conducted as described above, and that the enormous loss which the ore formerly sustained was caused by too high a temperature and an insufficient supply of air.

3. I demonstrated that such ores, properly assorted, could be successfully and very cheaply roasted in the Brückner furnace by the self-roasting process, if the salt is added to the ore in the battery, provided suitable provisions be made to separate and pulverize the hard but well-chloridized balls which form in these furnaces.

Parting and Refining Gold and Silver.

By Titus Ulke.

During the year 1896 the following improvements in the parting and refining of gold, silver, and platinum alloys are to be noted:

Electrolytic Parting of Doré Bars.—B. Moebius now gives his depositing tank with endless belt a much simplified form. In each tank there are 24 wooden frames 1 in. deep, over the bottom of which a muslin diaphragm is stretched. The anodes, 12 by 3 by ½ in. in size, rest in the frames on four hickory cross-rods, placed about ½ in. above the diaphragm. About ½ in. below the latter, the cathode silver belt, weighing about 40 lbs., passes and carries the deposited silver to a revolving scraper or brush placed at the end of the tank. This scraper removes the silver and permits it to fall into a hopper at one end of the tank.

Treatment of Platiniferous Slags.—The following process for treating the platiniferous slag obtained in refining the gold residues from acid parting with niter is used at the New York Assay Office: In order to take out any remaining gold prills in the slag, the latter is placed in an old black-lead crucible in the furnace and the metallic portion sweated down to a king, which is separated. The platiniferous slag is now added to a molten bath of zinc. The latter alloys with the platinum and platinum metals, and leaves a clean slag, consisting of silica, potash (from the niter), oxides of zinc, and other base-metal oxides. The platiniferous zinc is then treated with dilute sulphuric acid, which dissolves the zinc, but leaves the platinum and its associates in metallic form. This metallic residue can now be worked up in the usual way into pure platinum and the metals of the platinum group.

Electrolytic Refining of Impure Gold.—In the Mineral Industry, Vol. IV., p. 360, attention was called to the fact that gold is separated from platinum and other metals by electro-deposition at the Norddeutsche Affinerie of Hamburg, Germany. The following is a detailed description of the method practiced there, as based on the German patent No. 90,276 of April 16, 1896, and abstracted from the Director of the Mint's report on the production of precious metals in 1896.

The patent covers an electrolytic method in which high current densities are used, so as to secure the gold in marketable shape in the shortest possible time.

In electrolyzing a neutral solution of pure gold chloride between gold electrodes, chlorine will escape at the anode, even when very low current densities (less than 10 amperes per sq. meter) are employed. This loss of chlorine cannot be prevented by simply increasing the temperature of the solution. The same phenomenon occurs with weak acid solutions, such, for example, as are obtained by evaporating an aqua regia solution of gold, when one uses higher current densities, yet lying far under 100 amperes per sq. meter.

Investigation has shown that this undesirable escape of chlorine, which increases with the current density, can be reduced and even be prevented altogether, within certain limits, by adding an excess of hydrochloric acid and by heating the electrolyte. In this way it is possible to utilize all the chlorine for dissolving gold at the anode, even in comparatively weak solutions of gold chloride and with current densities as high as 1000 amperes, and above, per sq. meter. The invention,

therefore, covers the process of electro-refining gold bullion in a heated solution of gold chloride, in the presence of an excess of hydrochloric acid.

The best results are obtained by keeping the electrolyte at 60° to 70° C., the gold contents at 25 to 30 gms. Au per liter, and the quantity of hydrochloric acid between 20 and 50 c.c. strong HCl (of 1.19 sp. gr.) per liter of solution, according to the current density. Any appreciable fall in the temperature or in the amount of acid present is immediately shown by the evolution of chlorine gas.

At the proper temperature, acidity, and current strength, the gold, platinum, and palladium, and nearly all of the impurities in the anode are dissolved. The undissolved residue consists of the larger part of the iridium present and some other platinum metals, silver chloride coming from whatever silver there is in the bullion, a portion of the lead as lead chloride when the electrolyte is saturated with lead salts, and bismuth as oxychloride when there is not sufficient acid to dissolve it. In the residue is always found, besides anode scrap, a quantity of very finely divided gold, amounting to about 10% of the weight of the anode. The presence of this gold is explained on the assumption that some AuCl is formed at the anode, which decomposes immediately into $AuCl_3$ and Au.

While in this way the components of the anode partly go into solution and partly collect in the slimes which gather on the bottom of the vat, the gold deposited even with current densities exceeding 500 amperes per sq. meter is absolutely pure. The metal that is deposited on the cathode is replaced in the solution chiefly by the gold dissolved at the anode, partly by equivalent quantities of other soluble metals, and also by the precipitated chlorides of silver, lead, and bismuth. The percentage of gold in the solution therefore steadily decreases as the electrodeposition proceeds, unless gold chloride is from time to time added. The corresponding increase in the percentage of impurities in the solution, under these circumstances, has no injurious influence on the purity of the deposited gold; on the contrary, the presence of a certain amount of impurities is even necessary in order to secure a dense electro-deposit, otherwise unattainable when high current densities are employed.

It appears that the above method, as carried out at Hamburg, is a continuous process, and reduces the consumption of acid to a minimum. Whereas the usual wet method of refining gold requires fresh hydrochloric and nitric acid and precipitant for every new portion of the material to be dissolved, the new electrolytic method permits the repeated use of the acid once added, so that this same small quantity of hydrochloric acid is used in dissolving and precipitating very many portions of the gold. Another advantage of the above method lies in the possibility of concentrating the small quantities of platinum in the anode, so that, after taking out all the gold from the solution, the platinum may be obtained as ammonium-platinum chloride.

For example, if 20 kgms. of gold bullion containing an average of 0.1% Pt are refined daily, in 100 days nearly 2 kgms. of platinum will have accumulated in the solution. Now as it has been demonstrated that the purity of the deposited gold is not injuriously affected as long as the quantity of platinum in the solution does not exceed twice the amount of gold present, the electrolytic process may be continued up to this point or to a time when the working up of the platinum appears advantageous.

When the weight of the palladium in solution exceeds 5 gms. per liter, this metal begins to deposit with the gold in traces; but as the amount of palladium rarely exceeds small fractions of the amount of platinum present, the above considerations alone need govern the renewal of the electrolyte.

In carrying out the process at the Norddeutsche Affinerie, the highest practical current densities are used to secure the most rapid output possible, and the anodes are made very thin (4 mm.) owing to the high value of the metal. With a current density of at least 400 amperes per sq. meter of anode surface, the anodes are consumed to within one-tenth of their original weight in 24 hours. The scraps are then remelted for further treatment, together with a like amount of the finely divided gold that has accumulated on the vat bottom. Rolled sheets of fine gold are used as cathodes. They are made narrower than the anodes on account of their gradual enlargement as the deposition proceeds. As the gold is not deposited loosely even with high current densities, the electrodes may be placed 3 cm. apart, without causing short-circuiting or deterioration of the deposit. This distance between electrodes must naturally be increased with pure solutions, or the cathodes must be changed frequently, as the gold thrown down from such solutions is not very dense.

The process is carried out in vessels of stoneware, porcelain, or of other material capable of withstanding the action of hot dilute muriatic acid. Automatic devices keep the height of the electrolyte in the vessel at a uniform level. When this level begins to fall, owing to evaporation of the hot electrolyte, etc., solution is automatically fed from supply-bottles filled with wash-water containing small quantities of gold chloride, obtained in washing the precipitates and residues.

In treating gold alloys to secure pure gold, gold chloride must be added to the solution the more frequently the larger the percentage of metals other than gold in the alloy, and the lower their electro-chemical equivalent weight. If the alloy to be parted is largely composed of silver, a compact, adherent, non-conducting layer of silver chloride soon covers the anode and, by reducing the anode area, causes the evolution of chlorine gas. Consequently the anode does not dissolve as fast as usual, and the percentage of gold in the solution is rapidly reduced if the layer of silver chloride forming on the anode is not periodically removed by mechanical means, such as brushes or scrapers. In a similar way, interruptions of the process due to the collection of oxychloride of bismuth may be prevented. If the alloy contains lead, besides silver and other metals, the muriatic-acid solution soon becomes saturated with chloride of lead, which spreads over and soon covers both electrodes, electrolyte, and the depositing vessel. An uninterrupted process, however, is even here possible if sulphuric acid is added to the solution from time to time, and if the lead sulphate which forms on the anode is removed mechanically.

In conclusion, attention may be called to the fact that the new method has many advantages over the old method of refining gold by acids and fluxes, and offers a cheap way of securing proof gold.

The Hydro-Metallurgy of the Precious Metals.

By Henry Wurtz.

This paper is introductory to a much more comprehensive work which is now in an advanced stage of preparation, and is necessarily but a rather fragmentary synopsis, setting forth in a general way some important points of the progress in the chemical metallurgy of the precious metals. In 1846–47, two metallurgists whose names are pre-eminent in the literature of science—Plattner and Percy—simultaneously and independently conceived the idea that chlorine, which had been known ever since its discovery 80 years previously to be a solvent of gold, might be used to extract gold from ores. Percy proceeded at once, as the record shows, to experiment on this hypothesis in his own laboratory. Plattner also made tests on ores in his laboratory, but did not put them on any record which has yet been discovered. Other chemists, however—Cöster, an apothecary of Patschkau, Dr. Duflos of Breslau, well known as a learned chemist, and Herr Lange, a director of metallurgical works—published experiments in 1848–49, fully confirming Plattner's statements.

In the meantime, Percy had previously made his experiments, in 1846, on an ore of gold and silver, but did not communicate the results until 1848, at a meeting at Swansea of the British Association for the Advancement of Science. By reason of the fact that the proceedings at this association meeting were not published in full until 1852, four years later, Percy's discovery was lost sight of, and it is very rarely that his name is referred to in works on metallurgy in this connection. An account of it will be found, however, in the London, Edinburgh and Dublin *Philosophical Magazine* for January, 1850, pp. 1–8. It is there stated to have been "communicated by the author, having been read at Swansea in 1848." The ore had been previously roasted. "It had a quartz gangue and contained blende largely, with galena, pyrite, and chalcopyrite. Silver was present as sulphuret. Assays showed 250.75 ozs. silver and 9 ozs. 15 dwts. gold per ton of the roasted ore. The precious metal regulus computed to 96.22% silver and 3.78% gold, and therefore 10 grains contained 7.676 pure silver and 0.301 pure gold."

The following statements are condensed greatly: In his experiment No. 5 he treated this ore with water in a stoppered bottle with a stream of chlorine gas for an hour. After going through a number of manipulations, hyposulphite of soda being also employed, the final result was a bead weighing 5.385 grains from 1000 grains of raw material, containing 0.1 grain of gold. In another experiment, No. 7, the resulting bead weighed 6.636 grains with gold 0.197 grains. He remarks that this bead contained, therefore, 2.968% gold (compare with the total contents as above). He obtained several other evidences of extraction of gold by the chlorine. He experimented also with chloride of lime.

The writer, after having patiently looked up and studied all the documents in this case, is convinced that the credit of the discovery of the art of chlorination of ores, though almost invariably conceded to Plattner alone, should be shared by Percy.

In connection with the Plattner-Percy discovery, it is a further fact of

historical interest that the first patent issued, in any country, for a practical method of chlorination of gold ores, comprising details of construction as well as method, was issued in America. The patentee was Charles F. Spieker, of New York City, the number 8729, and the date (antedated) August 10, 1851. This was but two years after the publication by Herr Lange of his practical experiments on a working scale on the auriferous mispickel of Reichenstein in Silesia, etc. The Spieker patent is not a mere adaptation of the Plattner-Percy discoveries, but presents internal evidence of much originality of invention in different directions and of apparent practical value.

The hydro-chemical arts of precious metal extraction may be regarded, generally, as roughly divisible into two successive stages.

(1) Solution of the precious metals from ores by the action of water combined with various chemical agents, solid, liquid, or gaseous, or two or more of these jointly; and with the aid of air (at the natural or higher pressure, or percolating in currents through the solvent), or of other gases, or of gases liquefied by pressure, or of heat, or of electric currents, or of light, or of agitation; or of any two or more of these agents or agencies combined; thus extracting either one, or the other, or both of the precious metals, in conjunction with, or without, other baser metals or compounds, according to the agents or agencies employed.

(2) Precipitating or causing to separate from such solutions, by any chemical agent, or by any mechanical agent or agency, or by heat, light, or electricity, evaporation, filtration, electrolysis, etc., one or the other or both of the precious metals, with or without other metals or compounds held in the solution.

Sometimes, however, the two stages may progress simultaneously, as often in electrolytic processes; and other complexities inherent in chemical operations may render this division a somewhat uncertain one.

The first general topic which will be taken up now with some little detail, but with no pretense as to completeness, is:

Amalgamation: Simple and Compound.—This branch of chemical metallurgy —which is almost, if not quite, always hydro-chemical—does not yield in importance to any other branch. Indeed it appears doubtful whether any system of metallurgy, of ores in which gold is likely to occur in the form of "free gold," will ever be discovered that can dispense altogether with amalgamation. This is because all aqueous solvents of gold naturally require more time to dissolve a particle of gold the larger it is. In amalgamation it is just the reverse. Quicksilver is more likely to get the coarser than the finer gold.

In the matter of age amalgamation ranks before all other chemical methods. The Romans, whether they utilized it or not (which does not appear), knew of amalgams of gold. Both Vitruvius, just before Christ, and Pliny, just after Christ, refer to them. But the record of amalgamation of ores dates back historically quite three and a half centuries. It appears first in early Mexican metallurgy, and is stated to be first described in print by Acosta in 1590; as being used in Mexico in 1557, only 38 years after the Conquest, and in Peru in 1574. It was even then in Spanish-American countries, as it is there still, a true hydro-metallurgic method—aqueous solutions of common salt, with mixed ferrous and cupric sulphates, proceeding from the roasting of chalcopyrite,

being essential auxiliaries—the latter under the name of "*magistral*," a word akin to our word nostrum or universal remedy, or *magisterium*. The series of chemical reactions induced by the cupric sulphate of the *magistral*, with the common salt—the nature of which must of course have been a total mystery to those who first handled it, and regarded as a species of magic or alchemy—were explained about 1833 by Boussingault (*Annales de Chimie et de Physique*, LI., p. 337), and since then by other chemists, as follows:

1. Double decomposition between the salt and the blue vitriol: $CuSO_4+2NaCl=CuCl_2+Na_2SO_4$.

2. The cupric chloride then reacts with the silver sulphide of the ore: $Ag_2S+2\,CuCl_2=2AgCl+Cu_2Cl_2+S$; the products here being all insoluble in pure water; but, by reason of the excess of common salt present, both the AgCl and the Cu_2Cl_2 pass into solution as soluble double chlorides, and the latter reacts on a fresh equivalent of: $Cu_2Cl_2+Ag_2S=Cu_2S+2\,AgCl$. There being still an excess of salt present, the silver-chloride remains in solution. Then the mercury steps in, not only as an amalgamating, but also as a decomposing agent, as follows: $Hg_2+AgCl=HgAg+HgCl$. Besides silver amalgam here, there is formed insoluble calomel, which is lost. For each pound of silver obtained by this older method, about 1.83 lbs. of mercury are lost utterly.

In the modern "Washoe Process," the silver is usually chloridized in advance by roasting with salt, and the silver chloride is then decomposed in the iron pan, or by addition of iron, electrolytically, without any agency of the mercury except to form amalgam; and little or no loss therefore ensues through formation of calomel, unless indeed, as does happen, mistakes are made through lack of comprehension of the simple chemical principles involved.

Numerous modifications of the ancient Mexican, and of the modern Washoe processes of amalgamation will be cited in the forthcoming work; and it is curious to see how great a variety of transmutations of the simple chemical principles involved have been devised and proposed, and doubtless often tested, by inventors and experimenters, during the last half-century. This collection has been made as complete as experience, labor, time, and care could make it. It is believed that it cannot fail to be interesting, suggestive, and valuable to all who are concerned, directly or indirectly, in the hydro-metallurgy of the precious metals, and in the profitable application of this series of arts. Of course all the published methods involving other agents as auxiliary to the mercury are cited, such as acid and alkaline agents, metallic solutions, including several of mercury itself, cyanides, soluble sulphides, heat, steam, superheated steam, mercurial vapor, carbon, amalgams of lead, copper, iron, zinc, bismuth, aluminum, manganese, thallium, and other metals, hyposulphites, various mattes and reguluses, chlorine, chloride of lime, fluorides, chromates, bichromates, chlorates, ammonia (caustic and carbonate), permanganates, bromides, niter cake, salt cake, alum, petroleum, sawdust, borax, glucose, maltose, levulose, mannitose, and so on. Also those by chemical precipitation of mercury on the gold in the ore, from various solutions. Electrolytic amalgamation also, with a great variety of electrolytes, and combinations and mixtures of electrolytes, and curious forms and compositions of anodes and cathodes. (See more on electrolysis further on.)

Sodium Amalgamation.—There has been one development of amalgamation —now dating back about a third of a century—some special remarks on which appear to the writer to be justifiable here, by reason of new developments made during the last decade, which should revive interest therein, and attract attention thereto. The development referred to is the use of the alkali-metals as auxiliary to amalgamation, sodium being the one naturally selected as most readily and cheaply producible. The discovery of the availability of sodium (or its amalgam) in this way was made about 1858 by the present writer; but no opportunity occurred to test the invention on ores until about 1863, when such experiments were made in New York, with the co-operation and advice of Dr. John Torrey, then United States Assayer in New York, and of his son, Herbert G. Torrey, now the successor of his father in the same position. The results of these experiments, with some made subsequently in California by Professor B. Silliman, and elsewhere by others, were so highly favorable that steps were taken to secure patents in the United States and Great Britain. The United States patent was granted December 27, 1864; but issue was delayed at request of assignees of the inventor (as the British application was then pending) for six months, as provided by the law. The United States and British patents were both issued, therefore, on the same date, six months subsequent to the allowance in the United States. About six or seven weeks had elapsed of this six months' allowance, after the granting of the American patent, when it was announced, and rapidly became a matter of publicity, that Professor William Crookes, of London, had just made this same discovery and invention, and had recorded specifications for patents therefor both in the United States and in Great Britain. No such patent, however, has been found on the record, in the United States patent office at least.

Among other remarkable contemporaneous circumstances, which at the time seemed scarcely credible, was that certain reports of Professor Silliman upon experiments made for the present writer, and on his specifications, in California, were published verbatim by Professor Crookes, as an interpolated chapter in his version of *Kerl's Metallurgy,* as experiments made for him (Crookes) to test his (Crookes') discovery. Attention was called to this at the time, in a review of this edition of Kerl, in the *Engineering and Mining Journal,* then entitled the *American Journal of Mining.*

This is referred to now only as a matter of history. So far as patents are concerned, these of course are dead. But the real discoverer is in justice entitled to claim priority, as proved by the facts and dates which have never been denied. The patents themselves yielded no commercial profit. This, mainly, though there were then other incidental reasons, through the impossibility of obtaining at the time enough of the essential material—the sodium.

This obstacle has virtually vanished, since through the inventions of Castner and others sodium can now be obtained cheaply and in quantities. It is believed that it can now be furnished in all the more accessible mining sections of America, and other continents, at less than $1 per lb.—this price being also readily reducible by one-half when the market shall have grown sufficiently. At such prices the cost of its use in amalgamation would be scar worth counting. It is to be remembered that the utility of sodium as

an aid in amalgamation has no reference to its power as a chemical reducing or deoxidizing agent. Its amalgamating value is dependent on a sort of contact-action, often called (without any intelligible meaning) a "catalytic" action or influence; but which the present writer prefers to call "chemical polarization." This is an action by contact, without necessary consumption of the excitant material. Nevertheless the sodium is slowly consumed by decomposing the water of the ore-pulp operated on, so that it has to be supplied gradually in small quantities.

There is another little chapter in the same history which should further interest users and handlers of quicksilver, for amalgamation or for any other purpose. This again, relating to another invention of the writer, has, he believes, likewise a value for the public though none now for himself. In one of the expired patents mentioned above there was a claim to the use of sodium in the "packing and transportation of quicksilver." It was pointed out that the prevailing usage is expensive and troublesome; with risks, amounting almost to certainty, of loss in handling and transfer. But when it is alloyed with from 4 to 5% of sodium it forms a solid metal of surprising hardness. This is crystalline and not difficult to break, but with cheap filling, as with dry hay, can be packed and conveyed even in thick glass or stoneware jars, and can be handled without any loss. Some tons of it were prepared by the writer and cast into circular disks of one pound each, which were packed like rolls of coin, in vulcanized rubber bags made for the purpose, 25 in each, which fitted snugly in rectangular strongly screwed wooden boxes. These packages were sent in 1867–68 to various distant parts of the world without any loss or accident. When fluid pure "quick" is wanted, the sodium is readily dissolved out of the amalgam by placing in water for a time.

Under the head, "Precipitation of the Precious Metals," some further points will be introduced relative to other uses of amalgam of sodium in hydro-metallurgy.

Chlorination, like amalgamation, has been the subject of a very great and very varied amount of experiment and invention during its half-century of existence. The present review can comprise little more than generalities. The methods tried and proposed may be chemically divided into two kinds: those in which chlorine is employed as such, in its elemental form; and those in which are employed what we may call chlorinogenic agents, mixtures or compounds which give up chlorine only when in contact with metals, or other bodies which have what is called an "affinity" for chlorine, this word signifying that the bodies have a natural tendency to combine with chlorine. Such mixtures or compounds, however, do not give up their chlorine to many other bodies that have also the so-called affinity for chlorine. Therefore there has been invented a hypothesis, apparently widely accepted as truth, that there is a peculiar state called the "nascent state," the state of "being born," which expresses the condition of the chlorine while leaving its existing combination to combine anew with other bodies. But there is here a lack of rationality. The property is not a property of chlorine itself, but one of other bodies; and is indicated by the behavior of other bodies toward the so-called nascent chlorine, as follows from the above facts. In other words, the affinity of the

chlorine, or its tendency to combine with other bodies, cannot be concerned so much as the tendency of the other bodies to combine with it. The other bodies have this peculiar tendency to such a degree as to be able to break up certain compounds of chlorine (as also of some other elements). Almost all metals have this tendency, or influence it might be called. Some chemists long ago abandoned the myth of the "nascent state" and invented another term, "predisposing affinity." This term, however, only states the fact, without throwing any light on it. The writer classifies all these phenomena together under contact-action, or so-called "catalysis," and believes that it belongs also to the same class of phenomena as voltaic action of metals, and other electric conductors, on liquid compounds. He therefore proposes, or rather adopts, the term, "chemical contact-polarization," to generalize all these phenomena of metals and their behavior toward liquids, with or without voltaic resultant, or electrolysis. The novelty resides here mainly in including "catalysis" in the generalization, that is, in making all chemical contact-phenomena arise from a general principle in nature. The term chemical polarization, and part of the conception here conveyed, is traceable in the writings of Berzelius, Mitscherlich, and Graham.

To show further the irrationality of the term "nascent state," we may consider the case of aqua regia—one of the commonest of the chlorinogens, used in fact to dissolve gold out of ores in the chemical analytical laboratory probably since the days of Basil Valentine, 400 years ago, since this "iatrochemist" states that gold may be dissolved in it. It was proved long ago by Gay Lussac that aqua regia is made up substantially of two acid compounds of chlorine, nitrogen, and oxygen, with hydrogen as their basic component. Add gold, and the hydrogen and oxygen combine to water, or rather to nitrogen hydracids, and terchloride of gold results. It is just as rational here to assert that the aqua regia contains nascent water or nascent nitrogen acids; as these are just as inevitably "born" of the reaction as is the chlorine or the gold chloride itself. The writer proposes further to regard chlorinogens as containing potential chlorine, or chlorine-energy in the potential form, this giving us a new analogy between chemical and mechanical phenomena. Mechanical potential energy is developed and made active by introducing a new agent or condition of external origin. Just so the presence of the gold is a new condition introduced into the chemical status of the chlorinogen. To take another instructive example, exactly the same thing occurs when gold is put into a solution of cyanide of potassium in the air. It is known that here absorption of oxygen from the air plays an essential part. The gold can only take the cyanogen in its presence, and with joint production; from 2 KCy, H_2O, Au_2 and O; of 2 AuCy and $K_2H_2O_2$ (caustic potash). In recent controversies about the "cyanide process" in courts of law, much use has been made of the mythic "nascent cyanogen." Here, however, we have just as plainly nascent potash, in the mixture set forth above.

Chlorination has, like amalgamation, been subjected by inventors and projectors to a vast number of experiments, both alone and in conjunction with amalgamation. Chlorine has been used directly in its gaseous form—as first by Plattner—under ordinary pressure and under elevated pressure, sometimes

under pressures high enough to liquefy it (that is, 9 to 10 atmospheres) at ordinary temperatures; also in solution in water and in liquids which make stronger solutions than water. Also in very weak solutions, such as are employable in open vats, without suffocating the operatives. It has been applied after exhaustion of air from the ore. Instead of passing the gas through the ore, the ore has been passed through the gas. Many chlorinogens, containing potential chlorine, have been used. Indeed one of the commonest of these in use was employed by the earliest investigators, Percy, Duflos, and Lange. This is a mixture of common bleaching powder (chloride of lime) and muriatic acid. With this Duflos reported good results, and Lange got better results than with free chlorine as proposed by Plattner. This mixture has been since often reinvented and patented, with more or less slight and trivial modifications by inventors devoid (as they very often are) of any reliably complete knowledge of the previous history of the art.

Chlorination, in its simplest form, is but the dissolving of free gold out of ores, raw or roasted. But the problem is rarely as simple as this. Gold and silver, in elemental forms, are twin elements, occurring as natural alloys, which could only have been formed (or at least could only be *artificially* formed, so uniform is the alloy in general) by chemical, possibly electrolytic, deposition from solution. Alloys formed by fusion are hard to obtain of such uniformity. Nature has herself evidently practiced hydro-metallurgy to an enormous and world-wide extent. Now, chlorine does not convert silver as it does gold into a soluble chloride, and the insoluble silver-chloride may interpose a complete obstacle to the access of the chlorine to the gold. Hence, invention has been needed, as will be explained subsequently, to circumvent this difficulty. Copper, lead, zinc, and other "base" metals, are often in the way extensively, being taken up also more or less by the chlorine; and chemical ingenuity is very often called for to convert these from obstacles into—what they should be—sources of profit. Thus the number of chemical problems that beset the chlorination of ores is large.

In addition to the chlorinogenic mixtures already mentioned, there are others. The first patentee of a chlorination process, Spieker, used bleaching salt with sulphuric acid, instead of hydrochloric. This gives hypochlorous acid, $HClO$, a very strong chlorinogen. Subsequently others have used hypochlorous acid, first isolated and freed from the precipitate of gypsum which must have embarrassed Spieker. Mixtures of oxalic acid and bleaching salt have been claimed to present advantages. Also mixtures of bleaching salt and ferric chloride. Both these latter are transportable in solid forms, and hence present conveniences.

Some inventors have claimed that the product of chlorination, gold terchloride, may be practically volatilized at certain temperatures in atmospheres of chlorine and other gases, and have based processes on this. Ingenious combinations of chlorination and amalgamation have been made, one by a well-known and famous American metallurgist, now deceased, in which he proposes to obtain amalgams of the precious metals only, yielding pure bullion on retorting, by using just enough chlorine to eliminate the base metals in solution. This is perfectly rational and feasible.

In recent years there have been increasing crops of inventions of methods of

chlorinating by electrolysis, the processes being highly various and interesting. Many different chlorides are used as electrolytes, and a great deal of mechanical ingenuity exercised in devising new apparatus and mechanical devices, fitted to promote the various results aimed at.

Thus, as some examples: Lixivia, produced by various leaching processes, are made to flow in continuous currents between anodes and cathodes. Trough-shaped anodes, with elongated cylindrical cathodes revolving in them, convey currents of lixivium. Mechanical devices remove deposited metals and oxides from surfaces of electrodes, continuously, as fast as formed; and convey the same to receptacles provided outside the bath. Filtering apparatus is so constructed as to contain anodes and cathodes of granular carbon; in order that such cathodes may become saturated with the metallic deposit, which is then obtainable therefrom by dissolution or by combustion. Cathodes are made cellular, porous, or spongy in internal structure, so that the lixivium can flow in a continuous current through them. Water then replaces the lixivium, to wash the deposit. Then succeeds a solvent of the deposit, the cathode being converted into an anode; and so on, with repetitions indefinitely. All this may be made automatic. Gold is deposited from certain electrolytes, on anodes, as auric oxide and compounds thereof with bases. The potential of a current is so graduated that some metals only are carried down, leaving others in solution; also so that some only are dissolved out of mattes, etc., leaving others behind. Electrolytes are used adapted by chemical composition, degree of concentration and of conducting power, chemical solvent action, temperature of bath, etc., so as to take up only certain metals from anodes. Electrolytic baths are so constructed as to rock, and allow mercury at the bottom to flow from and to the anode and cathode compartments alternately; thus becoming charged with a metal in one compartment, which is then dissolved out again in the other compartment. There are more of such chemi-mechanical devices, opening new fields in hydro-metallurgy.

Chloridization.—This is a mode of operation often applied to ores rich in silver—consisting in first roasting with salt, to chloridize the silver; after which chlorination is applied to bring both the precious metals into solution, the gold as terchloride, and the silver as a double chloride of silver and sodium. After roasting, in the hydro-metallurgic stage more salt is often added to help dissolve the silver chloride; and sometimes hyposulphites of sodium or calcium are added to aid in this. The mode of roasting with salt is no doubt quite ancient. In *Poggendorff's Annalen*, in 1827, a method is described, by no less a chemist than Leopold Gmelin, of extracting silver ores by roasting with salt, and then dissolving out the silver chloride with ammonia. This use of ammonia very rarely occurs in recent metallurgy of silver. It should be economical, as the ammonia is recoverable by distillation; but there has probably been some other objection, real or fancied.

Bromination.—The substitution of bromine for chlorine for dissolving gold from ores has attracted some attention of late years. The drawback to it is that bromine is not abundant enough, and doubtless never will be so, to be consumed without special additional devices, both chemical and mechanical, and additional labor, for its recovery; whereas chlorine is so abundant that the cost of

the agent itself is but trifling. Theoretically the amount of chlorine required to produce gold chloride costs less than 0.006 part of the value of the gold in the chloride. One point of interest relative to bromination is that, just as in the case of chlorination, the first patent in any country, so far as has been discovered, was to an American—Charles A. Schaeffer, of Ithaca, N. Y. Application was filed June 20, 1880.

Precipitation of the Precious Metals from Solutions.—The number of methods for accomplishing this is considerable. There are solid precipitants, mostly metallic, though not all so. Liquid precipitants, comprising various metallic salts and haloids in solution, and some organic bodies, have also been employed, and gaseous precipitants, of which several were used long ago.

Metallic Precipitants.—The precipitation of the precious metals by the so-called baser metals comprises methods which have been among the most useful and important in the practice of hydro-metallurgy. In searching through the general literature of chemistry a century back an interesting discovery was made, illustrating the early period at which the study of this important branch of our subject was taken up by chemists. A book was encountered entitled, *Lavoisier's Elements of Chemistry*, translated from the French by Robert Kerr, F.R.S., Edinburgh; published by William Creech, 1790. (This date is but four years previous to that of the cruel murder of Lavoisier by the guillotine.) On p. 232 of this is a table of the "Combinations of Oxygenated Muriatic Acid" (chlorine) "with the Salifiable Bases, in the Order of Affinity." In a note it is added that this order was discovered by Monsieur Berthollet in 1786. Berthollet's experiments on the "order of affinity" of metals for chlorine were made by determining the order in which they separated or precipitated one another from solutions of their chlorides. Several of the metals included in the table were probably unknown in a pure state at that day. Eliminating these, twelve may be selected which doubtless were so known and used by Berthollet. The order of these stated is as follows: Zn, Fe, Pb, Sn, Cu, Bi, Sb, As, Hg, Ag, Au, Pt. In 1827 Fischer of Breslau stated that gold is precipitated from its chloride *quickly* by Fe, Sn, Cu, Pb, Zn, Bi, and As; *slowly* by Cd, Ni, Hg, and Ag; and *most slowly* by Sb.

In more recent chemical literature there is much more on this subject. Gore was an industrious investigator in a branch field, that of the order of precipitation of the metals electrolytically. He worked with many classes of electrolytes besides chlorides.

Zinc and iron are most commonly used as metallic precipitants at the present day, mainly by reason of their cheapness. Aluminum is not too dear now, especially in the form of scrap, and presents certain advantages. When the lixivium contains copper which is to be saved that metal may sometimes be used to advantage to precipitate previously a high-grade bullion, to which a little copper is, however, likely to adhere.

In connection with precipitation by metals, some further remarks may here be introduced, looking to the further revival of the subject already brought in above, of the utilization of metallic sodium in hydro-metallurgy. At its recent low commercial valuations, of $1 or less per lb., sodium, considering its low equivalent, its purity, its great energy, and the great solubility of its oxide,

possesses the qualifications of a rapid, clean, and powerful metallic precipitant in a higher degree than any other known metal.

Sodium itself, as a precipitant, could hardly be handled by any known means. But when combined with mercury its enormous energies are effectually harnessed and brought under perfect control. In a previous paragraph, headed "Sodium Amalgamation," allusion is made to certain patents granted to the present writer in 1864. Further reference is now made to an article on this subject in the *American Journal of Science* for March, 1866, Vol. XLI., pp. 216–225, in which is found the following paragraphs:

"When gold has been obtained in solution, either from ores or from other materials, by the action of chlorine, aqua regia, cyanide of potassium, or any other solvent, also when silver has been obtained in solution, in hyposulphites or otherwise, the most rapid and thorough mode of saving these metals will be found to be their conversion into amalgams, by precipitation with metallic iron in contact with magnetic quicksilver, more especially when the solutions are dilute.

"The greater rapidity and perfection of the precipitation, in these cases, are obviously due to the absolute contact at once established with the iron surfaces by the magnetic quicksilver, and the perfect and powerful voltaic circuits thus kept up constantly throughout the two metals and the solution."

(The term "magnetic quicksilver," used in each of these paragraphs, was a "trade name" at that time—applied, as explained in another part of the same article here quoted, to simple amalgams of sodium.)

It will be readily understood that what is here quoted was substantially contained in the old expired patents that have been referred to, and is therefore now public property.

Computing on the basis of the chemical equivalents of sodium and gold, sodium being rated at $1 per lb., and pure gold at $330 per lb. avoirdupois, sodium as a precipitant for gold (as sodium amalgam) costs $1 for each $2810 worth of gold theoretically precipitated. The waste of sodium, with skill in handling, should not amount to much. For gold, therefore, the cost as a direct precipitant is very small. In the case of silver, at its present valuation, about $9.25 per lb. avoirdupois, the cost of sodium computes to $43.40 in silver for $1 in sodium as direct precipitant. So that this is not beyond reasonable bounds. Presence of much copper in a lixivium would increase the cost appreciably. Lead could be first precipitated with sodium sulphate. In order that the price of sodium should come down one-half, a commercial demand only is necessary.

In the forthcoming work other modes of precipitation by common metals will be explained. One example, an American invention, might be briefly noticed, as an instructive illustration of the combinations of chemical and mechanical devices, toward which, as already pointed out, there is a present strong tendency. Zinc is, in this case, the precipitant metal, this metal being cast into the form of spheres, which are made to roll around on circular zinc pathways or grooves. The lixivium to be precipitated circulates in continuous currents up around the moving balls, passing through a settling tank in its course. The friction constantly detaches the precipitated metals. Though not

referred to by the inventors, electrolysis here clearly takes part in the precipitation, as the balls will have surfaces of clean anodic zinc, contiguous to other surfaces coated with electro-negative precipitated metal, which will be cathodic toward the lixivium.

Precipitation by other Solid Precipitants.—Among these are metallic sulphides and carbon. The favorite sulphides in use are those of iron and copper. Sulphide of iron—though the fact seems to have been forgotten—was described as a precipitant for gold in 1853. Sulphide of copper was probably first used about 1883, though supposed to have originated some five years later. Precipitation by carbon, which has been reinvented often, turns out on investigation to date back to the early years of the century. Recent metallurgical writings have attributed its discovery to Dr. John Percy in 1869, but the writer has traced it at least half a century further back. Witness the following extracts from an American edition of 1819, edited by the illustrious Dr. Robert Hare, from the eighth London edition, of 1818, of the *Elements of Experimental Chemistry*, of William Henry, F.R.S., Vol. II., p. 53, where is found, besides the one referred to above, another passage of hydro-metallurgic interest, relating to the solution of gold by pure chlorine, long before Plattner and Percy, with its advantages over aqua regia.

"The proper solvents of gold are the oxymuriatic acid (chlorine) and nitro-muriatic acid. Oberkampf (*Annales de Chimie*, Vol. LXXX., p. 140) prefers the former (chlorine) because a purer solution is obtained, and one which can more easily be had free from excess of acid. If gold leaf be suspended in water into which chlorine gas is passed, it is dissolved, and the solution may be concentrated by evaporation. The solution of gold has an orange yellow color, due to excess of acid, and passes to a brownish-red as soon as the redundant acid is neutralized or expelled by heat (Oberkampf). The solution gives a purple stain to the skin, and is susceptible of crystallization."

On p. 54 (same vol.) we find: "Into a dilute solution of muriate of gold contained in a glass jar, put a long narrow slip of charcoal, and expose the whole to the direct light of the sun. The gold will be revived, and will appear on the charcoal in a metallic state, exhibiting a very beautiful appearance. The same change ensues without light, if the solution be exposed to a temperature of 212°."

The following is quoted from a work called *An Essay on Combustion*, by Mrs. Fulhame, published in 1794 in London. [Reference may be made also to Count Rumford in the *Philosophical Transactions* for 1798, p. 449, for the same.]

"Moisten a piece of white taffeta riband, with the dilute solution of gold, and expose it to a current of hydrogen gas. The gold will be reduced, and the riband will be gilt with the metal. By means of a camel's-hair pencil the gold may also be so applied as to exhibit regular figures when reduced.

"The same experiment may be repeated, substituting phosphureted hydrogen for common hydrogen gas."

The reader will find details of various experiments of a similar kind in this document.

O'Driscoll attributes the discovery of precipitation by charcoal to Newbery of Australia (pp. 146–8); citing, in support, Egleston, *Proc. Am. Inst. Min.*

Eng., February, 1880, Vol. VIII., p. 454. See also the American patent to Wm. Morris Davis, of Philadelphia, No. 227,963, February 11, 1880.

As to the precipitation of silver by carbon, a knowledge of this also by chemists is found to date back more than half a century. It was experimented on by Weppens as early as 1846 (*Revue Scientifique* for 1846, p. 251). Of this there is a notice also in the *American Journal of Science*, 1846, p. 260.

Precipitation by Saline Solutions.—Green vitriol, or ferrous sulphate, is the oldest of these in the history of chlorination, though of course known far longer as a gold precipitant in the chemical laboratory. It was used by Plattner and his disciples in 1847–49. Many metallurgists of the present day still prefer this to all other precipitants. Ferrous chloride has some advantages over the sulphate, and when lime is present, as sometimes happens, it is preferable; but it does not keep well in the air, and the sulphate is in this respect advantageous. Stannous chloride solution is among the oldest precipitants and is still sometimes used in special cases.

Organic Solutions as Precipitants.—Oxalic acid has been considerably used, especially to obtain finely metallic and crystalline precipitates. Sugar syrup made alkaline, and boiled with the lixivium, has been used by Mitscherlich to produce "gold black." Crystalline gold for dental use has also been precipitated with sugar and with gum by special chemical management. Sugar was claimed, in form of molasses, by an American metallurgist in 1885 to be a preventive of the precipitation of lime as sulphate with the gold, in green-vitriol precipitation.

Under this head may be brought in some remarks, with new suggestions of the writer, relative to peculiar, and as yet mysterious, forms of the precious metals, which have from time to time attracted some attention, but not so much as is due. What is sometimes called "float gold" is among these, though these forms often occur, as the writer is persuaded, without receiving a name, or being recognized at all. Since the results of Muthmann, Carey Lea, and others (on what is now called soluble "colloidal silver") have been published, the present writer has gone back to and carefully studied, in the original, Faraday's experiments on certain supposed precipitated forms of gold published 40 years ago (*Philos. Trans.* for 1857, Vol. CXLVII., p. 155), from which he (Faraday) concluded that his ruby-colored liquids contained gold suspended in infinitesimal division, because he found that a cone of light from a lens was partially reflected with a golden color. The writer has concluded from this study that Faraday's, Muthmann's, and Carey Lea's preparations are all of the same nature, not suspended forms, but real solutions, and even not true colloids. Faraday's golden reflection presents the characteristics of a fluorescence. Moreover, he (the writer) has become convinced that these forms of solutions of gold occur in nature, and may be formed in the operations of hydro-metallurgy; and that we need, not only tests of their presence, but modes of ready conversion of them into polymeric forms, which will settle and can be collected.

It would appear that these phenomena, as resulting from the handling of gold and gold ores in the arts, and in the gold mill, have not escaped notice altogether. H. L. Sulman, a London chemist, has obtained British patents,

No. 24,939 of 1893, and No. 8405 of 1894, for the use of soap for coagulating "float gold." He appears to look on the material, however, as of the nature of suspended ore-pulp, and does not set forth any such views as the above.

It is to be added that, so far as his (Sulman's) invention is concerned, he has been anticipated for 14 years by an American inventor (John Tunbridge, of Newark, N. J.—United States Patent No. 228,004, filed November 4, 1879) who used soap exactly in the same way, for separating gold from jeweler's wash, mine-waters, etc. He also calls it suspended gold. He adds lime to the water first, and then soap solution. His patent expires in May, 1897.

Gaseous Precipitants.—Sulphureted hydrogen is of course the oldest of these, having been from time immemorial a laboratory agent for precipitating gold and silver from solutions. It was also prescribed in what appears to be the first British chlorination patent, to a Frenchman, Edouard Primard, No. 1931 of 1857. Hydrogen gas also has been long known as a reducing agent for gold chloride, as shown in the above extract from the book of Mrs. Fulhame, published more than a century ago. It may be of some interest to state that in *Henry's Chemistry* even earlier references to this property of hydrogen exist. In Dr. Thomas Thomson's *Treatise on Chemistry*, an edition of 1810, Mrs. Fulhame's essay on "Combustion" is quoted, with the same statements as above, and others, regarding the effect of sunshine and of heat in bringing about the precipitation by hydrogen. Hydrogen has been recommended highly by some as a precipitant, but others regard the necessity of heating the lixivium, even up to 140° F. sometimes, as a serious objection. Others have stated that too much time is occupied by the hydrogen in accomplishing its work. Sulphur dioxide is another gaseous precipitant, highly lauded by some.

Precipitation by Electrolysis.—Some allusions have already been made to this method, or class of methods, of precipitation, which is likely to go far, under favorable conditions, to displace all others. It has been of late years the subject of experiments on a great scale, especially in the Transvaal regions. It presupposes of course the command of electricity, which, when at command, will naturally lead to adoption of modes of solution and precipitation jointly, through its instrumentality. In connection with this subject, the past work of Gore, now very little known to metallurgists in general, will again, as already hinted, come up as of much importance. This relates to the electrolytic positions and relations of the elements, in various electrolytes, and at various temperatures.

Solution of Precious Metals by Sulphur Compounds.—Of these methods there are, so far, three. More may yet be discovered. Of these three, but one, and that the one yet least developed, appears to apply especially to gold. This one involves the conversion, or attempts at conversion, of the gold of ores into a soluble sulphaurate of an alkali metal. Such compounds—not difficult to obtain in the laboratory—have the general formula $M_2Au_2S_4$; or M_2S, Au_2S_3—M standing for an alkali metal. It was claimed by William Henderson, in the specification of a British patent issued to him in 1857, that if an auriferous pyritic ore be smelted with copper ore to a matte containing 30 or 40% of copper, and this matte crushed, and then fused for an hour or two with sodium sulphate and carbon, then thrown in water, that the heated water

then dissolved out a soluble sodium sulphaurate. W. A. Dixon, F.C.S., in a paper read to the Royal Society of New South Wales in 1877, recounts experiments made by him with ores of that country by Henderson's plan, in which he failed to get any gold in solution. But these experiments should be repeated, in varied forms, with other ores. The soluble sulphaurates ought to be producible from their elements as extant in ores, just as they are from the same elements elsewhere; and human experience has sufficiently proved that the components of ores do not differ in chemical properties and relations from the same components in any other association.

The two other classes of sulphur compounds alluded to—the hyposulphites and the bisulphites—have apparently comparatively little solvent power for gold, but great power over silver and its compounds, in ores and elsewhere. This applies especially to the soluble hyposulphites. Their introduction into silver hydro-metallurgy is attributed to Percy and Patera; but the laboratory knowledge of their solvent power over silver chloride and other insoluble silver compounds is much earlier, and appears to be referable primarily to Sir John Herschel, to whom it is credited by Gmelin, in his *Handbuch der Chemie*, Heidelberg edition of 1844, Vol. III., p. 621, without giving reference. Herschel, it appears from his biography, studied chemistry for some 20 years of his earlier life. He is said to have perfected the art of photography, as we now have it, by introducing the use of hyposulphite of soda, but this must have been at a later date than the above. At present, under this head of hyposulphite, attention will only be further directed back to the extract given above from the *American Journal of Science* of 1866, relative to the use of amalgam of sodium for the precipitation of precious metals from their solutions. It may there be observed that hyposulphite solutions are among those specified as precipitable in this way. For these solutions, soluble sodium sulphides are in general use for precipitation, on the ground that the hyposulphite is thus regenerated, for re-use. It may be further pointed out now, with relation to the aid of sodium amalgam in this precipitation, that not only is the sodium hyposulphite thus directly regenerated, but the precipitate obtained is metallic amalgam, and not sulphide of silver.

Cyanogenation.—Prussian blue was discovered as long ago as 1704, and used thereafter as a valuable pigment. The first record we have, however, of any investigation into its nature by a competent chemist, was that of Scheele, in 1782. Scheele extracted from the blue pigment the wonderful liquid, prussic acid. He tried to analyze this, and reported that it contained ammonia, aerial acid (carbon dioxide), and phlogiston. To translate this into our nineteenth century chemistry, we must remember that phlogiston was no myth. It meant energy. Carbon dioxide was (according to the scientific faith of only four generations ago, the faith in which Priestley died at Northumberland, Pa., in 1804) carbon which in burning had lost its inherent phlogiston or energy. Give back to the product of combustion, to the carbon dioxide, its phlogiston, and you have carbon again. So that Scheele's analysis, with us, indicates nitrogen and hydrogen (of the ammonia), and carbon. Of course it was at first inferred that the prussian blue was a compound of the new acid with oxide of iron; that is, a prussiate of iron, and the product formed by the

action of potash on it was "prussiate of potash," which is now a trade name for both ferrocyanide and ferricyanide of potassium (yellow and red prussiates of potash). A confusion of names thus arose, the effect of which seems to have scarcely yet died out altogether (*vide infra*); although as long ago as 1815 chemists became familiar with the true constitution of all these bodies, after Gay Lussac's great discovery of the compound radical cyanogen.

To show, however, that the erroneous but time-honored names still remained in use, even in chemical writings, it may be well to make two or three citations from a schoolbook of the earlier part of the century, published seven years subsequent to the discovery of cyanogen. This is entitled *Rudiments of Chemistry*, by Samuel Parkes: London, 1822. On p. 161, it is said: "Prussiates are easily formed by boiling the alkalies or earths with prussian blue. The salts which are there formed retain, however, some iron, and are called "triple prussiates." Those of potash and soda crystallize readily, and the former is used as a test to detect the presence of iron," etc. On p. 194, under the head of "Cyanogen:" "It is obtained from prussiate of mercury by distillation." From this it is clear that cyanohydric acid (still called, as a trade name and in general popular parlance, "prussic acid") was confounded, even in chemical books, with ferrocyanohydric acid, a product from the "yellow prussiate" or potassium ferrocyanide. Elsewhere, under "Iron," Parkes says: "Prussiate of potash is the usual test for iron present as peroxide." Other later chemical books, even up to the date of the Elkington patent, 1840, kept up this fashion; just as now at the present day many objectional terms are too much sanctioned by custom to be abolished; as for example the popular term benzene or benzine (both pronounced alike) for petroleum naphtha, carbolic acid for phenol, and a great many others. Indeed, this very term prussiate of potash—meaning cyanide of potassium—is to be found in patent literature fully a generation later than the date of the Elkington patent.

The true metallic cyanides were of course well-known in the laboratory; and must have been regarded as desirable products to obtain on a larger scale; for we find in 1834 a paper was published in the *Philosophical Magazine*, by F. and E. Rodgers, presenting substantially the method now in use for obtaining the true cyanides readily and economically from the yellow prussiate, a method regarded as of such importance, and advocated so earnestly by Liebig that his name became attached to it, and it is known, up to this time, as "Liebig's method." The method in the original form of the Rodgers memoir was as follows: "Cyanuret of potassium may be prepared by exposing a mixture of anhydrous ferrocyanuret of potassium to a moderate red heat in a covered porcelain crucible for about 20 minutes," etc.

The fact of the dissolution of insoluble compounds of gold generally in cyanides was apparently observed by Scheele, as indicated in the following citation made by Gore (1856) from Scheele's *Chemical Essays:* "If these calces" (by which term he designates cyanides of gold and silver, which he had discovered and was then examining) "have been precipitated, and a sufficient quantity of the precipitant be added in order to redissolve them, the solution remains clear in the open air, and in this state the aerial acid (meaning CO_2, as before) does not precipitate the metallic calx."

This passage, it is stated by Gore, attracted in 1840 the attention of a chemist of Birmingham, a Dr. Wright, and led him to make experiments which showed, among other things, that the metals gold and silver could both be dissolved by cyanide solutions, and be precipitated from such solutions by electrolysis, as copper was then being deposited by the Elkingtons for a multitude of purposes.

Dr. Wright was therefore the discoverer, in 1840, of two new arts, namely, cyanogenation of refined metallic gold and silver, and electro-gilding or electro-silvering. The famous patent obtained in 1840 by the Elkingtons for electro-gilding, etc., was based on communications to them from Dr. Wright.

In this patent, cyanide of potassium is called throughout "prussiate of potash." For example, on p. 6, lines 27–29, they specify: "We take oxide of gold prepared by any of the known methods, or metallic gold in fine division, and dissolve the same in a solution of prussiate of potash or soda." Again, on p. 7, lines 23, 24, we find: "Metallic gold in fine division dissolved in prussiate of potash."

It appears quite probable that this solvent power of cyanides over the precious metals was known to some chemists even before this date. It was certainly known, and published in America, not as a novel thing, but seemingly as something already more or less known. The writer possesses an American edition, published in Philadelphia in 1843, of *Graham's Chemistry*, in which there is, on page 320, a footnote by the editor, Dr. Robert Bridges, as follows: "Cyanide of potassium in solution dissolves in a rapid manner stains produced by solutions of gold, silver, and copper."

The next important chapter in the history of cyanogenation was "Prince Bagration's Memoir," published in 1843 in the *Bulletin of the Russian Imperial Academy of Science*, and translated into the *Journal für praktische Chemie*, 1843, p. 367. Bagration was not aware of the experiments of Wright; and both he and Elsner (three years subsequently) attribute the discovery of the facts to the Elkingtons. Bagration endeavored to dissolve gold which had first been precipitated in a fine state of division by ferrous sulphate, with the aid of a voltaic current, in cyanide of potassium. He quickly found, as he states, that such current took no part whatever in effecting the solution, which was altogether due to direct solvent action of the cyanide on the gold. It is, however, curious, as well as unfortunate, that a half-century of patentees since this date have remained in the state of mind that Bagration was then cured of; being and remaining imbued, somehow, with the idea that a current helps to dissolve gold out of an ore, or ought to do so, even without any metallic contact therewith. Hence much waste of time, labor, and patent-literature in dictating laws to Nature.

Dr. Scheidel, in his admirable "Report to the California Mining Bureau," in 1894, has recognized the above point in Bagration's results, as well as another important point therein, namely that hydrocyanic acid itself, as isolated in solution from its alkaline compounds, still retains it solvent power over gold.

The distinguished chemist, Dr. E. A. Schneider, recently announced this last fact, in a communication to the *Engineering and Mining Journal*, evidently unaware of his anticipation for half a century by Bagration. It is the

more remarkable that Scheidel, in common with some other writers, when referring to the Elkington patent, has rested under the impression that this patent, when specifying "prussiate of potash" really means ferrocyanide. Scheidel distinctly says that Bagration was the first to dissolve metallic gold in a cyanide solution. There is, however, conclusive internal evidence in the Elkington patent that cyanides were meant, and it has not been called in question by any who have read the patent with care. The term prussiate was employed in deference to the popular usage of the term.

To the next important development in the art of cyanogenation, the great and leading discovery of Elsner, published in 1846, of the essentiality of aerial oxygen, adequate space cannot now be given, and it must be deferred.

Notwithstanding the very conclusive evidence that we have glanced through, of the solution of gold without voltaic action by cyanides, there seems to have been room still for the claim that, so far as gold existing in its native forms in ores is concerned, there was no evidence that these forms had been known to be amenable to these solvents, previous to the issue of certain patents some ten years since, which patents first opened this up, as asserted, to the knowledge of chemists and of mankind in general. The vast literature of chemistry had therefore to be sifted to find proof that it had been known previously that gold in ores possessed the same chemical properties and relations as other gold. In a previous paragraph there has been cited, in another connection, a passage in chemical literature dating thirty years back, showing a knowledge on the part of the present writer of the solvent power of cyanides over gold and silver in ores. So far, this is the earliest (1866) distinct statement of this fact; but it is not improbable that earlier ones may exist. The writer is able further to refer to another passage, in another paper of his own, of about the same date, which contains distinct allusions to experiments then in progress with cyanide of potassium in the battery and in the pan, in Colorado and California. This is in the *American Journal of Mining* (the predecessor of the *Engineering and Mining Journal*), of December 7, 1867, pp. 354–5, in a footnote.

About one decade later than the above date there was read to the Royal Society of New South Wales (on August 1, 1877) a most important paper by Mr. W. A. Dixon, F.C.S. This has already been alluded to in a previous paragraph on another branch of the subject. The whole of Mr. Dixon's extensive experiments, made with numerous agents and processes, as related in this memoir, were made on Australian ores; and in the paragraphs relative to experiments with cyanides to dissolve gold and silver, there is unmistakable internal evidence that he was operating on gold, silver, and sometimes copper as they occur together in native ores. Of these ores he gives the results of elaborate analyses and assays. The following extracts give some idea of Mr. Dixon's work.

After describing chlorination and other methods, he goes on to say: "The only other salts of gold which possess sufficient stability to render their use possible for its extraction are the auricyanides and the bromides of the alkali metals." The following are condensed notes of his experiments with cyanides. He found that with the addition of "alkaline oxidizing agents, the solution of the gold was sufficiently rapid. With calcium hypochlorite, ferricyanide of

potassium, and binoxide of manganese, all the gold was dissolved; with potassium chromate none; with ferrocyanide, no result. He used heat, as it appears, in these experiments, for he goes on to say: "In the cold, however, with the exception of ferricyanide of potassium, none of the above oxidizing agents had any effect. At 212°, however, they all reacted rapidly, and potassium permanganate the same." He points out that "both the gold and the silver of the ore are dissolved by the cyanide, and that they could then be precipitated separately." He relates many other and varied experiments with cyanides on various kinds of ores, both raw and roasted with salt; also with roasted arsenical pyrites. He spoke with much favor of the results in general when cyanides were used with various oxidizing agents.

GRAPHITE.

THE American production consisted of pure graphite from Ticonderoga and impure graphite from Rhode Island. In the latter State the New England Manufacturing Company opened a large seam at Cranston, which contained from 40 to 65% carbon, and produced some mineral from it. A 14-in. vein of graphite of remarkable purity was reported to have been discovered about 5 miles south of Coon Rapids, Iowa, but nothing further is known concerning it.

GRAPHITE INDUSTRY IN THE UNITED STATES.

Year.	Production. (a)		Imports.		Consumption.	
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1892.....	1,298,363	$64,920	23,354,000	$667,775	24,652,363	$732 695
1893.....	882,912	39,731	28,882,000	865,379	29,764,912	905,110
1894.....	770,846	34,689	11,640,000	225,720	12,410,846	260,409
1895.....	377,450	17,286	20,124,160	260,111	20,516,168	277,751
1896.....	405,006	18,225	33,824,000	437,189	34,229,006	455,414

(a) Production of refined graphite only; of amorphous graphite there were produced 840 short tons ($4700) in 1895 and 574 short tons ($3850) in 1896.

Since the English graphite deposits and those in the Tunkinski Mountains in Siberia have been exhausted the purest graphite of Europe and the best for the manufacture of lead pencils, has been obtained from Bohemia. In Austria large quantities of graphite are also found in Upper Styria, Moravia, and the Duchy of Austria, but these occurrences are generally impure and find only a limited market. The Bohemian deposits are at Schwarzbach and Mugrau in the southern part of the kingdom, where they are found imbedded with gneiss, and accompanied by calcite. There are 12 mines, but only 3 are worked. Their annual production is about 13,000 tons, of which two-thirds is exported.

WORLD'S PRODUCTION OF GRAPHITE. (IN METRIC TONS.)

Year.	Austria	Canada	Ceylon.	Germany.	Great Britain.	India.	Italy.	Japan.	Russia.	Spain.	United States.	Totals.
1891.....	21,346	236	21,026	3,824	*Nil.*	(a)	2,415	2,464	164	63	272	52,245
1892.....	20,978	151	21,300	4,036	*Nil.*	(a)	1,645	601	82	34	707	49,461
1893.....	23,807	20	21,900	3,140	*Nil.*	(a)	1,465	3	311	*Nil.*	634	51,046
1894.....	24,121	350	10,718	3,133	*Nil.*	1,623	1,575	.....	(a)	10	400	41,879
1895.....	28,443	199	13,711	3,751	41	.....	2,657	.....	.....	*Nil*	349	48,973

(a) Not reported in the Government statistics.

GYPSUM.

THE occurrence, methods of mining, and uses of gypsum have been fully described in previous volumes of THE MINERAL INDUSTRY. In 1896 there were no new developments in the industry worthy of special note. The production fell off materially in nearly every State. The imports also declined heavily. There is no gypsum exported from the United States, and the sum of the domestic production and the imports into the country represents consequently its consumption.

PRODUCTION OF GYPSUM IN THE UNITED STATES.

(In tons of 2000 lbs.)

States.	1895.			1896.		
	Tons.	Value.		Tons.	Value.	
		Total.	Per Ton.		Total.	Per Ton.
California	1,230	$12,300	$10.00	4,800	$48,000	$10.00
Iowa	18,600	43,710	2.35	18,000	42,300	2.35
Kansas	74,163	380,929	5.14	52,172	246,508	4.72
Michigan	80,601	174,067	2.17	56,000	117,600	2.10
New York	55,323	109,281	1.98	47,873	99,621	2.08
Ohio	33,510	101,107	3.02	21,341	45,142	2.12
Texas	5,670	32,045	5.56	5,290	30,029	5.67
Virginia	12,435	40,195	3.23	8,844	30,008	3.39
Other States (*a*)	17,040	80,585	4.73	17,329	80,591	4.65
Total short tons	298,572	$974,219	$3.26	231,649	$739,799	$3.18
Total metric tons	270,864	Per met. ton..	3.59	210,152	Per met. ton..	3.52

(*a*) Includes Indian Territory, Missouri, South Dakota, Utah, and Wyoming, in each of which the output is reported from only one company.

GYPSUM IMPORTED INTO THE UNITED STATES. (IN METRIC TONS.)

Year.	Ground or Calcined.			Unground.			Value of Manufactured Plaster of Paris.	Total.
	Quantity.	Value.	Per Metric Ton.	Quantity.	Value.	Per Metric Ton.		
1888	3,393	$20,764	$6.11	159,204	$170,023	$1.06	.....	190,787
1889	5,554	40,291	7.25	173,701	179,849	1.03	.....	220,140
1890	7,689	55,250	7.18	174,030	174,609	1.00	.....	229,859
1891	9,713	97,316	10.01	112,021	129,003	1.15	.....	226,319
1892	6,992	75,608	10.81	184,002	232,403	1.26	.....	308,011
1893	3,417	31,670	9.26	166,929	180,254	1.08	.....	211,924
1894	2,059	16,823	8.17	165,100	179,237	1.08	.....	196,060
1895	3,348	21,526	6.42	195,630	215,705	1.10	$10,352	247,583
1896	3,348	22,058	6.58	183,165	193,549	1.06	.....	215,607

IRON AND STEEL.

By Frederick Hobart.

In 1896 the United States alone of all the chief iron-producing countries showed a smaller output than in 1895. All over Europe the year was one of great activity in the iron and steel works. Production was large and demand everywhere kept pace with increasing output. The course of production in this country followed the general condition of trade, or was an index of it.

Production of Iron Ore.—The consumption of iron ore in the United States in the manufacture of pig iron is estimated for 1896 at 15,332,800 long tons or 17,172,736 tons of 2000 lbs. The imports of iron ore, chiefly from Spain, amounted to 682,806 long tons or 764,743 short tons, leaving the domestic product consumed approximately 14,650,000 long tons or 16,424,500 short tons. Of this quantity 64.4% came from the Lake Superior mines. The imports amounted to 4.4%, and the remaining 31.2% came from the Eastern and Southern mines. However, the demand in the earlier part of the year was sufficient to cause considerable activity in the Eastern Pennsylvania, Lake Champlain and Hudson River districts, and even in the old region of Western Massachusetts and Connecticut, where certain mines which had been idle for years were reopened. In the Alabama mines work continued steadily throughout the year.

PRODUCTION OF IRON ORE IN THE UNITED STATES. (*a*) (IN LONG TONS.)

Year.	Red Hematite.		Brown Hematite.		Magnetite.		Carbonate.		Total Production.		Imports.		Total Supply.
	Tons.	Per Cent.	Tons.	Per Cent.	Tons.	Per Cent.	Tons.	Per Cent.	Tons.	Per Cent.	Tons.	Per Cent	Tons.
1892..	11,646,619	68.1	2,485,101	14.5	1,971,965	11.5	192,981	1.2	16,296,666	95.3	806,585	4.8	17,103,251
1893..	8,272,637	68.3	1,849,272	15.3	1,330,886	11.0	134,834	1.1	11,587,629	95.7	526,951	4.3	12,114,580
1894*a*.	8,660,000	71.9	1,820,000	15.1	1,280,000	10.6	120,000	1.0	11,880,000	98.6	168,541	1.4	12,048,541
1895*a*.	12,520,000	75.7	2,110,000	12.8	1,280,000	7.8	78,000	0.5	15,988,000	96.8	524,153	3.2	16,512,153
1896*a*.	11,500,250	75.0	2,056,500	13.4	1,020,000	6.7	73,250	0.5	14,650,000	95.6	682,806	4.4	15,332,806

(*a*) Distribution estimated.

The imports of iron ore were chiefly from Spain, although a few cargoes came from Cuba, Algeria, and Greece. Most of the imports were on contracts made during the last quarter of 1895, when nearly all the Eastern anthracite furnaces were at work, and there seemed a probability of a very heavy demand

for iron continuing throughout the year. Toward the latter part of the year the filling of contracts and the gradual decrease in pig-iron production, which prevented additional orders, caused a falling off in imports, and during the last four months they were very light. The Cuban mines furnished a somewhat irregular supply, owing to the war. No new sources of foreign supply were developed.

PRODUCTION AND CONSUMPTION OF IRON ORE IN THE UNITED STATES.

Year.	Production.				Imports.				Consumption.				Tons Ore to Ton Pig Made.
	Long Tons.	Metric Tons.	Value.		Long Tons.	Metric Tons.	Value.		Long Tons.	Metric Tons.	Value.		
			Totals.	Per Ton.			Totals.	Per Ton.			Totals.	Per Ton.	
1894	11,880,000	12,070,080	$20,790,000	$1.75	168,541	171,238	$267,241	$1.59	12,048,541	12,241,318	$21,057,241	$1.75	1.81
1895	15,988,000	16,243,808	27,979,000	1.75	524,153	532,539	786,207	1.50	16,512,153	16,776,347	28,765,207	1.74	1.75
1896	14,650,000	14,884,400	28,567,500	1.95	682,806	693,731	1,092,489	1.60	15,332,806	15,578,131	29,659,989	1.93	1.78

As noted above, very nearly two-thirds of the ore supply came from the Lake Superior mines, which continue to be the chief source from which the furnaces of Illinois, Ohio, and Western Pennsylvania draw their raw material. The centers of northern iron production are yearly more and more dependent upon these great ore deposits, and the mining and transportation interests are yearly becoming more systematized, so that the ores are handled at a surprisingly low cost.

On the Mesabi range a good deal of exploration was carried on and some important new developments were made, but the shipments came chiefly from the mines already opened. This resulted from the fact that the range is now almost entirely under the control of the Lake Superior Consolidated Mines and of the Minnesota Iron Company. The developments made are for the future, and the new mines are held in reserve until their output is needed to supply the market.

Just at the close of the year a very important deal was announced by which the Lake Superior Consolidated Mines transferred its principal shipping properties, the Mountain Iron and the Rathbun, to the Carnegie Steel Company under a 50-year lease. The Lake Superior Consolidated, generally known as the Rockefeller interest, had in previous years acquired very extensive properties on the Mesabi range, but under the new deal it is apparently its intention to withdraw from the mining of ore. During 1895 and 1896 it built up a very large fleet of ore carriers on the lakes, but under the new arrangement it is understood that these will be at the service of the Carnegie Company at fixed rates, sufficient to return the owners an interest on their investment. The Carnegie Company will therefore have the advantage of securing its supplies of ore at the cost of mining and transportation, and will be entirely independent of whatever price for ores to outside buyers may be fixed.

There were practically no fluctuations in the prices of Lake Superior ores in 1896. In April a combination was made by the producers under the lead of the Lake Superior Consolidated Mines and the Minnesota Iron Company. The basis of the price was $4 per ton for standard Bessemer, Norrie, the well-known Gogebic mine, being taken as the standard, which was an advance of $1.10 over

the contract price for the season of 1895. On this basis the price for hard ores of the highest quality, such as Minnesota, from the Vermilion range, was as high as $4.25. Franklin and Biwabik were rated at $3.65, while the higher phosphorus ores of the Mesabi were fixed at $3.35. These quotations remained fairly steady throughout the season. The conditions of the iron market during the year prevented any increase in the area to which the Lake ores are supplied, and these were aided by the maintenance of high prices.

SHIPMENTS OF LAKE SUPERIOR IRON ORE BY PORTS FOR FIVE YEARS.

Year.	Marquette.	Escanaba.	Gladstone.	Ashland	Two Harbors	Duluth and Superior	Total.		Totals.
							Lake.	Rail.	
1892	1,026,338	4,010,085	115,886	2,223,683	1,165,076	4,245	8,545,313	528,930	9,074,243
1893	1,086,934	2,048,981	203,585	1,117,524	903,329	520,565	5,880,918	178,037	6,058,955
1894	1,424,409	1,657,240	79,109	1,731,703	1,373,344	1,367,286	7,633,091	115,841	7,748,932
1895	1,079,485	2,860,172	109,211	2,350,219	2,119,156	1,716,667	10,234,910	194,127	10,429,037
1896	1,564,813	2,321,931	220,887	1,566,236	1,813,992	2,156,179	9,644,036	290,410	9,934,446

SHIPMENTS OF LAKE SUPERIOR IRON ORE BY RANGES FOR FIVE YEARS.

Year.	Marquette.	Menominee	Gogebic.	Vermilion.	Mesabi.	Totals.
1892	2,666,856	2,261,499	2,973,993	1,167,650	4,245	9,074,243
1893	1,829,053	1,466,197	1,329,464	820,621	613,620	6,058,955
1894	2,058,683	1,139,273	1,810,290	948,514	1,792,172	7,748,932
1895	2,097,838	1,923,798	2,547,976	1,077,838	2,781,587	10,429,037
1896	2,603,839	1,560,467	1,799,971	1,088,090	2,882,079	9,934,446
Totals from opening of mines	11,256,209	8,351,234	10,461,694	5,102,713	8,073,703	43,245,613

RECEIPTS OF IRON ORE AT LAKE ERIE PORTS, 1895 AND 1896. (*a*)

Port.	1895.		1896.	
	Receipts.	Stock Dec. 1.	Receipts.	Stock Dec. 1.
Toledo	260,730	113,132	301,794	115,959
Sandusky	12,361	34,375	58,667	59,491
Huron	146,442	101,000	226,515	200,075
Lorain	214,219	224,264	191,445	231,288
Cleveland	2,312,370	1,200,792	2,313,170	1,419,311
Fairport	914,617	605,470	941,446	773,905
Ashtabula	2,474,791	1,301,302	2,272,822	1,441,666
Conneaut	244,967	292,460	327,623	275,800
Erie	811,989	335,718	847,849	355,222
Buffalo and Tonawanda	719,742	207,199	545,101	82,267
Totals	8,112,228	4,415,712	8,026,432	4,954,984

(*a*) Practically all of the Lake Superior ore goes to these ports; a relatively small amount goes to Chicago and vicinity. The statement of stocks includes only the amount on the docks for sale or contract delivery, and does not include stocks at Cleveland and Buffalo furnaces. The account of stocks at the receiving ports is always taken on December 1, the close of the shipping season, and not at the end of the year.

Pig Iron Production.—The year opened with the blast furnaces generally running at the high rate which they had reached during the boom of the latter half of 1895. At the close of that year there were 244 furnaces in blast, with an approximate capacity of about 1,000,000 tons a month. The reaction in demand and consequently in prices had already begun when January opened. Nevertheless, demand still continued large and the average for January was 240 furnaces, with a weekly capacity of 221,800 tons. In February there was a considerable contraction, the number of furnaces in blast falling to 215, with

a weekly capacity of 207,000 tons. For several months following, however, there was but little change, the output in March being practically the same as for February, when allowance is made for the three additional days in the month. April again showed small variation, but in May the number of furnaces fell to 195 with a weekly capacity of 199,600 tons. In June there was a further reduction, a number of furnaces going out of blast. In July, however, the output was kept up much better than had been expected, but in August there was another fall which continued through September and October. The lowest point was reached in the last-named month, when there were only 131 furnaces active and their weekly capacity was 116,550 tons, which was the lowest point recorded since the break after the panic of June, 1893. November and December showed a slight recovery.

PIG IRON PRODUCTION IN THE UNITED STATES.

Month.	1894.		1895.		1896.	
	Furnaces.	Production.	Furnaces.	Production.	Furnaces.	Production.
	Number.	Tons.	Number.	Tons.	Number.	Tons.
January	137	447,723	185	762,474	240	945,900
February	128	406,443	181	641,550	215	800,022
March	134	515,137	172	688,170	207	882,181
April	144	571,784	171	666,315	196	835,116
May	124	472,893	175	664,600	195	821,923
June	91	304,003	172	664,449	195	691,094
First half-year	.....	2,717,983	.....	4,087,558	.....	4,976,236
July	109	411,500	187	730,156	192	751,593
August	136	539,540	203	780,855	170	645,200
September	169	694,974	219	899,400	149	602,508
October	172	717,691	228	940,630	131	466,200
November	186	761,251	238	979,933	133	569,000
December	188	814,449	244	1,027,776	148	672,390
Second half-year	.....	3,939,405	.....	5,358,750	.....	3,646,891
Totals for year	.....	6,657,388	.....	9,446,308	.....	8,623,127

According to Fuel Used.	1894—Tons.	1895—Tons.	1896—Tons.
Anthracite	914,742	1,270,899	1,146,412
Coke	5,520,224	7,950,068	7,166,471
Charcoal	222,422	225,341	310,244
Totals	6,657,388	9,446,308	8,623,127

The decrease in 1896, as compared with 1895, was 823,181 tons, or 8.8%. The output for 1896, however, has been exceeded only three times before in our history—in 1895, in 1892, when it was 9,157,000 tons, and in 1890, when 9,202,702 tons were made.

Divided according to the class of iron, the production for two years was as follows, in long tons:

Kind of Iron.	1895.		1896.	
	Tons.	Per Cent.	Tons.	Per Cent.
Foundry and forge iron	3,650,889	38.7	3,499,899	40.6
Bessemer pig	5,623,695	59.5	4,654,955	54.0
Basic pig	.....	.....	336,403	3.9
Spiegeleisen and ferro-manganese	171,724	1.8	131,940	1.5
Totals	9,446,308	100.0	8,623,127	100.0

The decrease in foundry and forge iron was comparatively slight, the main loss being in Bessemer pig, the production of which was less by 17.2% in 1896 than in 1895. Basic pig was reported separately in 1896 for the first time. Pennsylvania furnished 219,863 tons of this iron; Virginia and Alabama, 73,604 tons; New York and New Jersey, 22,692 tons; Ohio and Wisconsin, 20,244 tons.

PRODUCTION OF PIG IRON IN THE UNITED STATES BY STATES. (IN LONG TONS.)

States.	1891.	1892.	1893.	1894.	1895.	1896.
Alabama	795,673	915,296	726,888	592,392	854,667	922,170
Colorado	18,116	32,441	45,555	73,669	58,508	45,104
Connecticut	21,811	17,107	12,478	7,416	5,615	10,187
Georgia	49,858	9,950	39,675	40,268	31,034	15,593
Indiana	7,729	7,700	5,567	.....	.....	.....
Illinois	669,202	949,450	405,261	604,795	1,006,091	925,239
Kentucky	44,844	56,548	47,501	33,854	63,780	70,660
Maryland	123,398	99,131	151,773	5,600	10,916	79,472
Massachusetts	8,990	7,946	7,853	156	4,710	1,873
Michigan	213,145	184,421	117,538	95,171	91,222	149,511
Minnesota	1,226	14,071	10,373	.....	.....	.....
Missouri	29,229	57,020	32,360	6,522	27,518	12,548
New Jersey	92,490	87,975	74,305	63,273	55,502	59,153
New York	315,112	310,395	191,115	175,185	181,702	206,075
North Carolina	3,217	2,908	2,843	.....	323	2,151
Ohio	1,035,013	1,221,913	875,265	900,029	1,463,789	1,196,326
Oregon	9,295	7,628	4,739	1,000	.....	.....
Pennsylvania	3,952,387	4,193,805	3,643,022	3,370,152	4,701,163	4,024,166
Tennessee	291,738	300,081	207,915	212,773	248,129	248,338
Texas	18,662	8,613	6,257	4,671	4,682	1,221
Virginia	295,292	342,847	302,856	298,086	346,589	386,277
West Virginia	86,283	154,793	81,591	80,781	141,968	108,569
Wisconsin	197,160	174,961	131,772	91,595	148,400	158,484
Totals	8,279,870	9,157,000	7,124,502	6,657,388	9,446,308	8,623,127

Classifying the production for two years more generally by districts we have the following statement in long tons:

District.	1895.		1896.	
	Tons.	Per Cent.	Tons.	Per Cent.
1. New England, New York, and New Jersey	247,529	2.6	277,298	3.2
2. Pennsylvania	4,701,163	49.8	4,024,166	46.7
3. Ohio	1,463,789	15.5	1,196,326	13.8
4. Illinois, Michigan, and Wisconsin	1,245,713	13.2	1,233,234	14.3
5. Missouri and west of Missouri River	90,708	0.9	58,873	5.7
6. Maryland, Virginia, West Virginia, and North Carolina	499,796	5.3	576,469	6.7
7. Kentucky, Tennessee, and Georgia	342,943	3.7	334,591	3.9
8. Alabama	854,667	9.0	922,170	10.7
Totals	9,446,308	100.0	8,623,127	100.0

We find from this table that the principal decrease in 1896 was in Pennsylvania and Ohio, the States which are the chief makers of Bessemer iron. Illinois also showed a very considerable decrease, but this was partly made up by a gain in the charcoal furnaces of the Lake Superior region. In New England, New York and New Jersey there was an increase. The New Jersey furnaces about held their own, and the gain came from the Lake Champlain furnaces and from the charcoal iron district which lies on the southern and eastern slopes of the Berkshire Hills in Massachusetts, New York and Connecticut. Iron making in Missouri has not flourished since the ore deposits of Iron Mountain and Pilot Knob were practically exhausted. Colorado furnished the larger part of the iron in this group, Texas making only a small contribu-

tion. No pig iron was made west of Colorado, the single furnace on the Pacific Coast—at Oswego in Oregon—having been idle since 1894.

During the year the Southern furnaces kept much more generally at work than those in other parts of the country. In Virginia, West Virginia and Alabama there was a marked increase, while Kentucky and Tennessee showed very little change. In Alabama especially the low price at which iron can be made enabled the furnaces to keep going when prices were too low for their Northern competitors, and in the latter part of the year sales of iron abroad assisted them materially. The Southern States furnished in 1896 21.3% of the total production of pig iron, against 18% in 1895.

The four leading States in 1896 were Pennsylvania, which made 46.7% of the total; Ohio, 13.9%; Illinois, 10.8%, and Alabama 10.7%. These four States furnished 82.1% of the total.

PRODUCTION AND CONSUMPTION OF PIG IRON IN THE UNITED STATES.

Year.	Production.	Imports.	Exports.	Stocks.	Consumption.	
					Totals.	Per Capita. Pounds.
1892.....	9,157,000	70,125	15,427	506,116	9,302,003	320
1893.....	7,124,502	54,394	24,570	662,068	6,998,374	237
1894.....	6,657,388	15,582	24,480	597,688	6,648,490	241
1895.....	9,446,308	53,232	26,164	444,332	9,626,732	315
1896.....	8,623,127	56,272	61,071	847,686	8,214,974	261

The increase in exports was chiefly in the later months of the year, and since the close of 1896 there has been a large further increase in the sales to foreign countries.

Steel Production.—The steel production of the United States for six years is given in the following table:

PRODUCTION OF STEEL IN THE UNITED STATES.

(In tons of 2240 lbs.)

Kinds	1891.	1892.	1893.	1894.	1895.	1896.
Bessemer.....	3,247,417	4,168,435	3,215,686	3,571,313	4,909,128	3,919,906
Open-hearth.....	579,753	669,889	737,890	784,936	1,137,182	1,298,700
Crucible.....	72,586	84,709	63,613	51,702	67,666	62,500
Miscellaneous....	4,484	4,548	2,806	4,081	858	900
Totals.....	3,904,240	4,927,581	4,019,995	4,412,032	6,114,834	5,582,606

The total production of steel thus showed a decrease of 832,228 tons, or 13.6%, as compared with 1895. This decrease was almost wholly in Bessemer steel, the falling off in that class of metal amounting to 989,222 tons, or 20.2%. In the output of open-hearth steel there was an actual increase of 161,518 tons, or 14.2%. Of the total production in 1896 therefore 74.4% was Bessemer steel, 24.4% open-hearth, and 1.2% crucible and other special steels. The figures show that the use of the open-hearth process is steadily gaining, as it has been for several years past. Had 1896 been a year of normal business prosperity, the gain would have been still more marked. Thus far the open-hearth metal finds its chief use in the making of structural material and

plates, very little of it being marketed in the form of rails, for which Bessemer steel continues to be chiefly used. The number of open-hearth steel plants under construction and projected exceeds that of Bessemer plants very largely. A considerable number of these new plants will use the basic process.

Of the Bessemer steel production in 1896 there was about 29% converted into rails, the total output of these having been 1,102,892 tons. The open-hearth production continues to grow, though to some extent checked by unfavorable trade conditions. In 1896 basic Bessemer steel was made in the United States for the first time, by the Troy Steel and Iron Company, at Troy, N. Y., the pig iron used being made from Lake Champlain ores.

Imports and Exports of Iron and Steel.—The imports and exports of iron and steel in the United States for two years, as reported by the Bureau of Statistics of the Treasury Department, were as follows:

IMPORTS AND EXPORTS OF IRON AND STEEL. (IN LONG TONS.)

Articles.	Imports.		Exports.	
	1895.	1896.	1895.	1896.
Pig iron	53,232	56,272	26,164	61,071
Scrap iron and steel	6,066	8,250	.....	.....
Bar iron	20,049	16,276	3,329	3,505
Rails	1,447	7,796	15,597	76,131
Cotton ties	3,485	7,157	.....	.....
Hoop and band iron	15	26	198	268
Steel ingots, billets, etc	26,255	20,356	3,148	8,855
Iron and steel plates and sheets	14,531	6,786	1,235	2,620
Tin plates	219,545	119,171	.....	.....
Wire rods	26,834	18,977	.....	.....
Wire and wire rope	5,773	6,775	29,674	38,043
Nails and spikes	.....	.....	10,228	14,854
Miscellaneous	978	759	.....	.....
Car wheels, number	.....	.....	14,106	15,842

The total value of all our imports of iron and steel, excluding iron ore but including articles not mentioned in the above table and which are not weighed at the custom-house, amounted to $19,506,576 in 1896, against $25,772,136 in 1895.

In addition to the articles named above there were also exported other manufactures of iron and steel, the weight and number of which are not given in the official records, including locomotives, machinery, hardware, sewing machines, saws and tools, firearms, scales and balances, miscellaneous castings, engines and boilers, cutlery, etc., the total value of all iron and steel exports in 1896 being $48,670,218, against $35,071,563 in 1895. To the amounts above given may also be added our exports of agricultural implements, which are largely composed of iron and steel, and which were valued at $4,643,729 in 1896 and at $5,319,885 in 1895. The value of the exports in 1896 largely exceeded those of any previous year.

IRON AND STEEL PRODUCTION OF THE WORLD.

The following table shows the iron and steel production of the world for the five years ending with 1896. In France, Germany, Belgium, and the United Kingdom the figures for 1896 are from the returns collected by the producers'

associations in their respective countries; for Russia the figures are from the estimate of the Ministry of Mines and Works; for Spain we are indebted to Señor Don Roman Oriol, and for Sweden to Prof. Richard Akermann. In all cases the figures are in metric tons.

PRODUCTION OF PIG IRON AND STEEL IN THE PRINCIPAL COUNTRIES. (IN METRIC TONS.)

Year.	Austria-Hungary.		Belgium.		Canada	France.		Germany.		Italy.		Russia.	
	Pig Iron	Steel.	Pig Iron.	Steel.	Pig Iron.	Pig Iron	Steel.	Pig Iron	Steel.	Pig Iron.	Steel.	Pig Iron	Steel.
1892	940,284	*a*480,000	753,268	260,037	38,514	2,057,300	682.000	4,937,461	*a*2,600,000	12,729	56,543	*a* 919,614	371,199
1893	982,707	*a*485,000	745,264	273,058	50,754	2,003,100	664,000	4,986,030	*a*2,600,000	8,038	71,380	1,160,737	390,000
1894	1,054,520	*a*490,000	810,940	396,914	45,327	2,077,647	663,264	5,559,322	*a*2,700,000	10,329	54,614	1,312,760	422,500
1895	1,075,000	*a*495,000	829,135	455,550	38,434	2,005,889	899,676	5,788,798	*a*3,941,300	*a*10,500	50,314	1,454,298	574,112
1896	1,130,000	*a*520,000	932,780	598,755	42,164	2,333,702	1,159,970	6,360,982	*a*4,195,000	11,287	59,500	1,629,810	*a*625,000

Year.	Spain.		Sweden.		United Kingdom.		United States.		All Other Countries.		Totals.	
	Pig Iron.	Steel.	Pig Iron	Steel.	Pig Iron	Steel.	Pig Iron	Steel.	Pig Iron	Steel.	Pig Iron.	Steel.
1892	247,329	56,490	485,664	158,978	6,817,274	2,966,522	9,304,428	5,001,494	*a*350,000	*a*250,000	26,863,865	12,883,263
1893	260,450	71 200	453,421	221,780	7,089,318	2,983,000	7.239.806	4.084,305	*a*350,000	*a*250,000	24,229,025	12,093,963
1894	260,000	70.000	459,132	205,865	7,364,745	3.050,000	6,757,248	4,482,592	*a*350,000	*a*250,000	26,058,970	12,785,749
1895	206,430	65,000	455.200	231,900	8,022,006	3,444,201	9,597,449	6,212,671	*a*375,000	*a*275.000	29,858,439	16,659,005
1896	246,326	104,577	466,400	250,600	8,700,220	4,306,211	8,761,197	5,366,518	395,000	290,000	31,009,831	17,581,131

(*a*) Estimated.

The gain in nearly all the European countries over 1895 was noteworthy. In all of them also the proportion of pig iron converted into steel continues to increase steadily. The use of steel is most marked in Germany; in Great Britain and France there is still a large production of wrought iron.

The increase last year in Great Britain carried that country very nearly to the first rank in pig-iron production, the United States exceeding it by only 60,977 metric tons. Four countries furnished 84.4% of the iron supply of the world: the United States having made 28.3%; the United Kingdom 28.1%; Germany 20.5%, and France 7.5%. The iron production for the first time in history exceeded 30,000,000 tons in one year. The large increase in the steel output in Great Britain in 1896 carried it in advance of Germany, which had held the lead for several years as between the two countries. Notwithstanding the decrease in the United States, this country still remained the largest producer of steel.

In North America the only producing country beside the United States is Canada, whose output is still small. Of the total output in 1896 the Nova Scotia furnaces made 31,091 tons, nearly all the rest coming from a single furnace at Hamilton in Ontario. The iron resources of the country are still only partially developed.

No South American country is at present an iron producer, though efforts are being made to establish the industry in Chile, where iron ores exist in sufficient quantities. In Asia iron is made in China and Korea, but chiefly by primitive methods, and in the absence of statistics the quantities can only be roughly estimated. Japan is carefully cultivating an iron industry which

promises growth in the future, as great care is being exercised to adopt the methods most approved by experience in the United States and Europe. In Africa a little iron is made here and there by native tribes, but its quantity is unknown. The only country where iron mines are worked by civilized methods is the French colony of Algeria, where there is a considerable production of iron ores, which are wholly exported.

Austria-Hungary.—There was less change in 1896 than in any other country in Europe. In Austria some advances have been made at the Bohemian steel works. The iron industry of Hungary is making slow but steady progress.

Belgium.—The iron works of Belgium were active, the pig iron production showing an increase of 103,645 tons, or 12.5%, and the steel works of 143,205 tons, or 33.8%. In Belgium, as in Germany, steel is taking the place of wrought iron.

France.—This country shows a clear gain of 16.3% in pig-iron and of 28.9% in steel. The total production of pig iron in 1896 was 2,333,702 tons, of which 493,361 tons were foundry iron, and 1,840,341 tons were steel pig and forge iron. The production of wrought iron, including forgings made from scrap, was 814,643 tons. The production of steel ingots included 726,848 tons of Bessemer, 401,921 tons of open-hearth and 31,201 tons of crucible and other special steels. There was, as compared with 1895, a gain of 45.4% in Bessemer, but only 6.8% in open-hearth steel.

Germany.—The statistics continue to show a steady and substantial growth, an active home demand and an increasing export trade absorbing the output. The steel production is very large and no other country converts so large a proportion of its pig iron into steel. This is chiefly due to the introduction of the basic process, which is specially adapted to German pig iron.

Russia.—The iron industry shows a substantial growth, the chief gain being in South Russia. The production of 99,500,000 pouds reported in 1896 is still considerably below the requirements of the country. The future growth of the industry is chiefly a question of transportation.

Spain is an extensive miner of iron ores, which are largely exported, its metallurgical industry being small.

Sweden.—We are indebted to Prof. Richard Akermann for the statistics of the production of iron and steel in Sweden during the year 1896. The Swedish blast furnaces, all of which use charcoal as fuel, turned out during the year 488,400 metric tons of pig iron, showing an increase of 33,200 tons as compared with 1895. The quantity of blooms produced from pig iron in charcoal hearths was 185,500 tons, the larger part of which was exported. The steel makers were active during the year, the manufacture of steel continuing to grow somewhat at the expense of the iron production. There are reported in 1896, 115,300 tons of Bessemer steel ingots and castings, and 135,300 tons of open-hearth ingots and castings, making a total steel production of 250,600 tons. The open-hearth process is increasing more rapidly than the Bessemer. The exports of bar iron and steel in 1896 reached a total of 180,515 metric tons, showing a gain of 2429 tons as compared with 1895. The export of iron ores is growing very rapidly, owing to the demand for the British furnaces, and, to a somewhat less extent, for the German furnaces. The ores sent abroad are

chiefly from the Gellivara and other deposits in Northern Sweden. The total shipments in 1896 were 1,150,695 metric tons, showing the large increase of 350,243 tons, or over 43%, as compared with 1895.

United Kingdom.—The total output of pig iron in 1896 was 8,563,209 long tons, an increase of 7.8% over 1895, and very nearly the highest point ever reached. About 50% of the product was converted into steel, and about 15% into wrought iron, the rest being used in the foundry or exported as pig iron. The production of steel in 1895 and 1896 was as follows, in long tons:

Year.	Bessemer.		Open-Hearth.		Crucible, etc.	Totals.
	Acid.	Basic.	Acid.	Basic.		
1895.....	1,093,675	441,550	1,564,868	159,869	100,000	3,389,962
1896.....	1,357,580	458,262	2,145,268	172,287	105,000	4,238,397

In Great Britain the open-hearth process has reached its greatest development, furnishing last year 54.7% of the total product. The basic process gains ground but slowly.

TECHNICAL PROGRESS IN IRON AND STEEL.

While no remarkable changes are reported there is to be noted a continued tendency toward the concentration of manufacture and to the development of production on a very large scale. The first of the great Duquesne furnaces, of the Carnegie Steel Company, at Pittsburg, has been put in operation. Two of these furnaces are now at work. One of these has recorded an output of 690 tons of pig iron in one day; 4294 tons in a week, and 17,182 tons in a month. Two more furnaces are nearly completed and are expected to produce nearly 1000 tons a day each. This far surpasses any furnace work previously done anywhere in the world. In England the highest average shown by a group of furnaces has been in South Wales—36,675 tons a year. In Germany the highest point reached is at the Horde furnaces in Westphalia, where four stacks have produced 178,762 tons of basic pig in a year, an average of 44,690 tons per furnace.

Technically a great deal of attention has been given to the study of the composition of iron and steel, and a number of papers and discussions have been published. The most important group of these was presented at the meeting of the American Institute of Mining Engineers in Chicago.

There were no new developments made public in the application of electricity to the metallurgy of iron and steel in 1896. It has been stated that works are soon to be erected in Sweden, where large water-power is available, to make iron by the De Leval electric process; but no details with regard to these works can be ascertained, nor has any iron been made by this process on a commercial scale.

The Iron Markets in 1896.

The year 1896 was an eventful one throughout for the iron market in the United States, and there have been but few in its history—if we except the panic years of 1873 and 1893—which have shown such important changes and fluctuations. While production remained at a comparatively high level for a considerable part of the year it exceeded the demand sufficiently to weaken prices and to leave the market generally in a most unsatisfactory condition. Furthermore, there has been no year on record in which pools and combinations had so strongly marked and unfavorable an effect as during 1896.

Something of the general course of the markets is indicated by that of the production of raw material as given in the preceding tables. We find that at the opening of the year there had been some recovery from the reaction which, toward the close of 1895, had set in from the activity caused by the December boom, and which had been accentuated by the scare which followed the issue of the President's Venezuelan message in December. During the month of January this recovery was generally well maintained, and was supported by the success of the bond issue, which replenished the gold reserve. In the second half of February, however, there was a halt in the market, and for several weeks it remained in a somewhat dull condition, and throughout the month of March sales began to decrease in amount, and the markets were in the state generally described as waiting or expectant. Not much weakness was developed, however, until April was reached, when orders began to fall off and the markets gradually weakened. At the opening of April a considerable excitement was caused by the meeting of the steel producers, and the preliminary arrangements for the formation of the pool, which later crystallized into shape and took the form with which we have since been so familiar. This pool included practically all the makers of Bessemer steel billets in the country, whether their billets as made were used up in finished material in the makers' own mills or were sold to the smaller mills in billet form. This pool at once adopted the policy of steadying the market by fixing a high price, the rate being established at $20.25 per ton at Pittsburg, and a penalty being imposed on all sales made below the established rate. The immediate effect of this organization was to stir up the opposition of the various interests affected, including chiefly the wiremen and the tin-plate manufacturers, whose business would be very seriously affected by the increase in price. This increase did not really take effect at once, however, as there were in existence a number of contracts for billets at a lower price which were still unfilled, and as there was a considerable stock of steel in the hands of brokers and middlemen, so that the combination price could not rule until these supplies were worked off, practically the result was that, until August, or in fact almost the first of September, the sales of billets at the pool price were exceedingly small.

The continued high rate of production of pig iron, combined with the decrease in demand, had an injurious effect upon the market, and while a strong effort was made by those concerned to keep up prices there was a gradual cutting and shading which reduced the actual selling prices. The reduction of output in June and the following month checked this to some

degree, but prices both for foundry and Bessemer pig continued to be somewhat irregular, and a pressure to sell began to be evident in most of the markets. In July certain Philadelphia representatives of the blast furnaces made an effort to effect a combination of the furnacemen, but two or three preliminary conferences made it very clear that there were too many interests to be harmonized to offer any prospects of success, and the matter was afterward dropped.

As the year advanced, also, the election began to be a factor in the market. It is an old saying that election year is always a dull year in the iron trade, but in 1896 this was emphasized by the fact that the issues as made up involved the whole monetary policy of the country, and consequently affected trade in a peculiar way. After the national conventions had been held the iron markets settled down into a dull condition; no one felt confident as to the future; no one was disposed to enter into any new enterprise or to accumulate stocks of finished material. Buying of raw material was done simply to meet immediate necessities, and prices were determined rather by the necessities of the seller than those of the buyer. The steel pool held several meetings in August and September, but did nothing beyond resolving to maintain the price at which it was already manifest that little or no business could be transacted. The reduction in pig iron output did not stimulate prices, as it was evident that it had not been made on the same scale as the decline in consumption, and that there was still a surplus of output beyond the immediate needs of the market. A little spurt was caused in September by the letting of some contracts by the Navy Department, but it affected only a few makers of plates, beams, and similar material.

As election time approached the iron market was more and more dull and depressed, and it was still more evident that business had been limited to immediate necessities. A few orders for rails for export were the subject of considerable interest, but were not sufficient in amount really to affect the trade. The fact that they had been taken at prices variously estimated at $6 to $8 per ton below the price charged for rails for home delivery caused a great deal of comment. Meantime the furnacemen began to complain in no uncertain tones of the high prices maintained for ore and coke, which continued at the rates fixed in the spring without any regard to the falling quotations and lower demand for pig iron. The furnace operators certainly had a difficult time of it, and were not by any means to be envied.

As election approached and the conviction grew stronger of the probabilities of success of the anti-silver party, a little more confidence was expressed, and while this did not cause much increase in actual commercial sales, it gave rise to a considerable number of speculative purchases, and a good many thousand tons, both of foundry iron and Bessemer pig, were bought by parties who intended to hold the iron for a rise. This helped to give steadiness to the pig iron market, and to stop the shading of prices, and in so far had a favorable effect. The actual result of the election in November was felt at once in the iron market, and for a week or two it seemed almost as if another boom was coming. The excitement quickly subsided, however, and while there was a real improvement in business, it was manifested in the confirming of contracts

already made, and in a willingness to close contracts for 1897 rather than in the immediate volume of sales. The realization of the fact that the election was but one step forward and that a long and tedious work remained to be done before the country could be put upon a sound financial basis, with some uncertainty as to the final result, checked improvement and made the feeling still one of doubt as to the future.

During the closing months of the year the iron markets were affected principally by the course of the different pools and combinations. The most important of these, of course, was the steel-billet pool, to which reference has already been made. As was said above, the old contracts and the stock in the hands of middlemen at the time the pool began operations practically prevented its prices from becoming operative for several months after its first organization. When these contracts and stocks were exhausted, business had fallen into a very dull condition, and the demand for finished material was so light that buyers of billets did not feel disposed to pay the pool prices, and preferred for the most part to shut down their works rather than to accumulate goods which they must sell at a loss, if at all. In this condition of affairs the trade was enabled to realize fully the great advantage which the pool gave to those steel-makers who owned their own mills and put their raw material into finished form in their own works. A meeting held, nominally to reorganize the pool, did little beyond including in it the open-hearth billets which had heretofore not been considered, and the price of open-hearth steel was fixed at $2 per ton above that of Bessemer—an action which was deprecated as injudicious by many of the owners of Bessemer steel works. If rumors are to be believed, the dull condition of business and the anxiety for work were two great temptations for certain members of the pool, and charges were very freely made to the effect that some of them had violated the agreement and were selling at prices below that fixed by the combination. While no evidence was furnished to prove these charges, there seems to have been some foundation for them. A meeting was held in December to settle the troubles, which was of a stormy nature, but which ended in the extraordinary announcement that the pool had been fully maintained, but that members were left entirely free with regard to prices. This action, which to many seemed like a practical dissolution of the combination, was followed by a general break in quotations. Heavy sales were made chiefly for future deliveries, and the price fell from the pool rate of $21.25 per ton at Pittsburg to $19, $18, $17, and even to $15.50, at which price a good many tons were contracted for.

It is understood that many Eastern mills covered their probable requirements for the first half of 1897 at prices ranging from $17 to $18.50 at tidewater. This fact rendered impossible the reinstatement of pool prices until 1897 should be well advanced, and early in 1897 the pool was practically dissolved.

Following the break-up of the pool—and perhaps partly explaining it—announcement was made of the deal between the Carnegie Steel Company and the Rockefeller iron-ore interests, to which reference has been made on a preceding page. The situation in the steel market is now practically that the Carnegie Company controls its own supply of ore; controls at a fixed rate all

the Lake transportation which it needs; has well advanced toward completion a railroad line of its own from the nearest Lake port to its Pittsburg mills, on which the cost of transportation will be reduced to a minimum, and finally, through its auxiliary, the H. C. Frick Company, controls the greater part of the output of the Connellsville coke region. With its enormous plant, to which all the latest improvements and economies have been added, and with its raw materials practically furnished at bare cost, this company is not only in a position to make steel cheaper than any other producer, but it is so situated as to be absolutely in control of the market, and make the prices of steel what it will. We have had pools and combinations without number in the iron market in this country, but never before a position like this, in which a single company could absolutely dominate the trade, and make any combination which can be formed simply the register of its own wishes. The situation is not altogether a comfortable one, and many are looking anxiously for the result.

To go back a little in the order of time and to take up another pool which made considerable stir, we find that the nail pool, which controlled the output at first of steel-wire nails and afterward of cut nails also, commenced to develop trouble during the dull months of the later summer. This pool, which had been in existence over two years and had been very thoroughly organized, occupied a good deal of space in the public attention from the beginning, since no other pool which had been, or possibly could be, organized in the iron trade came into such close and immediate relations with the public. The nail trade deals directly with the retail hardware trade, which goes into every village in the country, and it deals in an article of universal consumption and necessity in which any change in prices is felt by men of almost every business and every condition in life. This pool, taking hold of the market at a time when it was demoralized by excessive competition, succeeded in more than doubling the wholesale selling price of both wire and cut nails. Throughout the greater part of the year that price was maintained on the basis of $2.55 per keg for carloads lots in Pittsburg for wire nails and $2.40 per keg for carload lots in Pittsburg for cut nails. In a time of general reduction of prices the hardware trade resented the maintenance of these high rates, but for the most part their protests were unavailing. The managers of the pool had accumulated a large fund, which was used to buy off competition and to induce the few manufacturers who remained outside the association to retire from business, either by buying their works outright or by paying them a subsidy. As this course rather encouraged the industry of starting opposition works, the association added to it a regular system of boycotting, intended to prevent the manufacturers of nail machines from selling them to any one not in the pool. This was somewhat dangerous ground, however, and a test case made up in Indiana was decided in a very emphatic way by the United States Circuit Court, which held any contract in restraint of business of this kind to be against public policy and consequently void. Moreover, business began to fall to a very low point, retailers generally buying only what they were absolutely forced to, and a powerful public opinion was brought to bear upon the pool methods, which were widely published and commented upon. Moreover, the tax upon manufacturers for maintaining the pool and buying off outsiders grew to be onerous,

and many makers began to realize that it amounted to almost as much as the increase in the price of nails, and that they were simply incurring public odium and opposition without obtaining any corresponding money profit. Under these circumstances it is not surprising that a meeting was called early in December at which it was formally voted to disband the pool and to leave the trade free to open competition. The immediate result of this action was a drop in prices from $2.55 for wire nails and $2.40 for cut nails to somewhere about $1.40 and $1.25 respectively; and although natural conditions later induced a slight reaction from the lowest point, the year closed with nails selling on the basis of $1.50 to $1.55 at mill for wire and $1.35 to $1.45 at mill for cut nails. Another curious result was the cancelling of a number of orders for nail machines which had been placed with the manufacturers by parties who intended to go into the business in order to secure a subsidy from the pool; thus showing that its dissolution put an end to a profitable, if hardly a legitimate, branch of industry.

The steel-rail pool, which is composed of a few large concerns and is of a more stable character than any of the others, maintained the price of steel rails, standard sections, at $28 per ton at mill throughout the year. One event of 1896, the taking of several orders, amounting in all to 45,000 or 50,000 tons for export, principally to Japan, at a much lower price, created unfavorable comment but did not seem to affect the pool at all. In December a meeting was held at which it was resolved to put the price of rails at $25 per ton at mill for 1897, a reduction of $3 per ton. Railroad men are not inclined to accept this gratefully, however, and the opinion has been very freely expressed that in view of the present prices of raw materials and the conditions of trade $22 per ton would be a sufficient price. It is, moreover, the opinion that at $22 a much larger business would accrue to the company.

Since the close of 1896—in February, 1897—the rail combination has also broken up under conditions which somewhat resembled those of the end of the billet pool. The price of rails fell from the $25 fixed by the pool to $18 or $20 per ton. Large orders were taken even at $17. The result was the placing of very heavy orders, including large ones for export.

The demand for rails was light throughout the year; the unfavorable conditions of business forcing the railroad companies to keep their expenditures down to as low a point as possible, repairs were postponed wherever possible. The steel rails laid years ago to replace iron are now showing their wearing qualities, and in many cases it is possible to continue a steel rail in use for another year or two in a way which could not have been permitted when iron rails were in common use. The mileage of new railroads built during 1896 was light. Conditions did not favor new enterprises, and the older companies did not feel able to make any great extension of their branches or subsidiary lines. The estimates of the *Railway Age*, of Chicago, and the *Railroad Gazette*, of New York, the two principal authorities on this point, put the total new mileage for 1896 at about 1800 miles only, the lowest report made for a number of years. The building of electric railroads was also very unfavorably affected by financial conditions. Many new enterprises were projected, but very few comparatively reached the stage of actual construction. The

difficulty of securing money and high prices of rails combined to check new construction, and the results are seen in the diminished sales of material.

Exports of Iron and Steel.—Exports of rails have been already referred to. The conditions prevailing here, the possibilities of cheap production which have been developed in this country, combined with the great activity of trade abroad and the demand for iron, have led to the somewhat unusual condition of considerable exports of pig iron to England. Some of our Alabama furnacemen found that their iron could be sold in Liverpool at about 38s. per ton. At this price they could net at furnace almost if not quite as much as they were obtaining in the Northern markets, and advantage was accordingly taken of the opportunity. It is understood that 40,000 tons at least, chiefly of No. 2 foundry, with some forge iron, went abroad to Great Britain, while smaller orders were booked for Italy and other European countries. Efforts are being made by the Tennessee Coal and Iron Company to develop this trade with the hope of making it a permanent feature in the Southern iron business. Whether more than a temporary success can be secured remains to be seen.

Prices of Coke.—The price of Connellsville coke was maintained throughout the year at $2 per ton at furnace. There was talk at one time of a further increase, but conditions did not favor it, the comparatively light demand and the competition of West Virginia coke being sufficient to check the proposed movement. Most furnacemen found the combination of $2 coke with $4 Lake ore quite as much as they were able, and a good deal more than they were willing, to carry.

General Summary.—The fact that the large production of pig iron was generally absorbed and that the year closed with unsold stocks very little larger than they were at its commencement, emphasizes the fact that the margin between a dull and an active year is much narrower than is generally suspected. A comparatively small surplus production is quite sufficient to induce a fall in prices and to affect the markets unfavorably. That a year which could carry a consumption of raw iron only a few hundred thousand tons less than the most prosperous one on record should nevertheless be characterized universally as a year of dullness and depression is a remarkable fact, and leads us to believe that the dullness was due not so much to any actual decrease in demand as to the fact that the increase in demand was not proportionate to that in population and the general growth of the country.

The Local Markets.—We give below notes on the course of the leading local markets of the country.

CHICAGO.

THE year 1896 was an unfavorable one in the iron trade at this center. Throughout the entire year there was displayed a great lack of confidence, and consequently business transacted was but little better, if any, than the preceding year. Low prices in all lines prevailed, and in some instances they were lower than ever before quoted. Competition was unusually sharp, and therefore the consumer bought his material at a price lower, probably, than he ever will find again. The buying throughout the entire year was almost entirely confined to quantities for temporary use, and but few contracts of any

importance were closed. The year opened up with no increase in trade, and with prospects for a continuance of such conditions. The great mills of the Illinois Steel Company, at South Chicago, were closed down the entire month of January, and through the year this company was compelled to shut down a number of times, thus showing the bad industrial conditions.

In pig iron sales there was a slight increase in the latter part of January, and in February business had become quite brisk. Some large buying of Southern iron was done, chiefly by the car builders, who bought heavily; it was estimated that more iron had been bought by this class of consumers during the month of February than the aggregate of the entire twelve months of 1895. Fully 40,000 tons of iron were placed in February through this market. Despite the increased business, prices fell off 50c. per ton over opening quotations. The months of March, April, and May were fairly good in point of sales of pig iron, but the remaining months of the year, with the exception of a few weeks in November after election, business was very dull. Pig iron prices fluctuated greatly. Northern foundry No. 2 opened the year at $14, and in October it reached the extremely low price of $11, recovering at the end of the year to $11.75. Southern No. 2 foundry iron opened at $12.85, and reached its lowest point, $10.85, in September, closing the year at $11.90.

Steel rails were in very poor demand the entire year, the railroads having kept out of the market with great persistence. March and April were the best months of the year. Rail prices held at $29 per ton, Chicago delivery, the entire year.

In billets and rods business was poor most of the year, and it picked up in November and December only through the collapse of the nail trust. Billet prices fluctuated somewhat the first four months, the January price being $21.25; it fell to $19.50 in April, but improved to $21.25 in June, where it stopped the rest of the year, in accordance with the orders of the pool.

In bar iron there was some increase in demand from the railroad and agricultural implement manufacturers, but buying by these concerns was still comparatively light, taking into consideration their usual demand. Some of the rolling mills closed entirely during the summer, while others simply ran on from one-third to one-half capacity. Through the influence of the Bar Iron Association the price of bars was upheld most of the year, but for some reason fell off in December to 1.20c., the quotation from January to November being 1.35c.

In structural material the demand for bridge iron was larger than 1895, but in building shapes there was but little improvement. There was considerable iron used in the erection of elevated roads, but contracts for such material were placed a couple of years ago. The building laws confining the height of buildings in Chicago to 12 stories have injured business. Prices in structural material fluctuated much and but little strength was noticed the entire year.

In merchant steel business improved but little over 1895. The principal buyers in this line are the agricultural implement, carriage and wagon builders. They were by no means up to their usual demand. Prices were maintained fairly well.

The following table shows the range of prices for iron in Chicago during the year:

MONTHLY RANGE OF IRON PRICES, CHICAGO, 1896.

Month.	Pig Iron.			Steel Rails.	Steel Billets.	Bars.	Beams.
	Lake Superior Charcoal.	Northern No. 2 Foundry.	Southern No. 2 Foundry				
January	$15.00	$14.00	$12.85	$29.00	$21.25	1.30c.	1.80c.
February	15.00	13.50	12.35	29.00	19.50	1.30c	1.70c.
March	14.00	12.75	12.60	29.00	20.00	1.35c.	1.65c.
April	14.00	12.50	11.85	29.00	19.50	1.25c.	1.70c.
May	14.00	12.00	11.85	29.00	21.25	1.35c.	1.70c.
June	14.00	12.00	11.85	29.00	21.25	1.35c.	1.80c.
July	14.00	11.75	11.85	29.00	21.25	1.35c.	1.80c.
August	14.00	11.75	11.85	29.00	21.25	1.35c.	1.80c.
September	14.00	11.50	10.85	29.00	21.25	1.35c.	1.75c.
October	14.00	11.25	10.90	29.00	21.25	1.35c.	1.75c.
November	14.00	11.25	11.10	29.00	21.25	1.30c.	1.75c.
December	14.00	11.75	11.90	29.00	21.25	1.20c.	1.65c.

The steadiness shown by the pooled products explains itself. In pig iron Lake Superior charcoal showed few changes, because there is a special demand for that grade for certain purposes.

CLEVELAND.

Iron Ore.—Notwithstanding the depression of business in the iron trade during the year 1896, the season was a close second to the preceding year, when all former records of ore shipments were broken. The total Lake shipments of ore in 1896 were 9,657,921 gross tons, while those of 1895 were 10,233,910 tons. The rail shipments bring up the total for 1896 to 9,950,000 tons. The receipts at the Lake Erie ports were 8,026,432 tons against 8,112,228 tons in 1895. There was on the docks at the Lake Erie ports at the close of the shipping season 4,954,984 gross tons, against 1,949,698 tons at the beginning of the season. These figures indicate that the shipments to the furnaces since May have been 5,021,146 gross tons, against 6,339,496 gross tons for the corresponding period in 1895. During the season the receipts at the port of Cleveland were 2,313,170 gross tons, thus placing this port again in the lead as an ore receiving point, a distinction it lost to Ashtabula in 1888.

While the shipments of ore were heavy from Lake Superior, it was not a satisfactory season for the iron ore interests. When the season opened, it was calculated that 9,700,000 tons of Bessemer would be shipped, including some sold in the Mesabi range in 1895, but held over on account of the high lake freight rates. The lack of demand, consequent on a depressed business condition, made it necessary to cut the estimate down to 7,500,000 tons. A few months later it became apparent that there was not sufficient demand to warrant even that output. The sales of Bessemer were 5,000,000 tons, it is estimated, while the shipments were 1,000,000 tons more, which was held over from 1895. The price of Bessemers remained practically stable during the year, the dealers not deviating from the standard scales fixed by the ore association at the beginning of the season, which were $4.50@$5 for hard speculars and $4@$4.25 for Bessemer hematites.

There were more non-Bessemers sold during the year than was anticipated

when the season opened. During the latter part of the season especially, the demand for non-Bessemers was fairly good, and the prices on these ores fluctuated somewhat. Early in the season they brought $2.75@$2.85 at some of the Lake Erie ports. During the summer the price had fallen from 30c. to 40c., but there was a recovery under the demand which marked the close of the season.

Lake Freight Rates on Ores.—The year 1896 differed widely from its immediate predecessor, so far as lake freight rates were concerned, for the tables were turned and the shippers had their inning. Instead of being at the mercy of the vessel-owners, as they were during the latter part of the season of 1895, they practically dictated their own terms during the last few months of 1896. In 1895 the average "wild" rates, or rates ruling from day to day, were 50% higher than the contract price. Many new vessels were put into commission, and the result was that the order was reversed in 1896, the "wild" rates being about 60% less than the contract rates made before the season opened. Season ore contracts were made last spring from Ashland at $1.05, from Marquette 95c., and from Escanaba 70c. The average "wild" rates were: Ashland, 77c.; Marquette, 66c.; Escanaba, 52c.

Pig Iron.—Considering all the circumstances, pig iron held its own fairly well in this market during the year. The highest quotation for Bessemer pig in last January was $13 and it was the same in December. On the whole, the market seems to have improved, as the lowest quotation in December was $11.75, against $10.50 in January. The high-water mark was reached during the latter part of May and the first week in June, when it brought $14. It came very near touching the low mark again in October, when it sold at $10.75, only 25c. advance over the low price of last January. The following table will show the range of the price of pig during the year, with averages for each month:

PRICES OF BESSEMER PIG IRON AT CLEVELAND, 1896.

Month.	Highest.	Lowest.	Average.	Month.	Highest.	Lowest.	Average.
January	$13.00	$10.50	$11.75	July	$12.75	$12.25	$12 50
February	13.00	13.00	13.00	August	12.25	12.25	12.25
March	12.75	12.75	12.75	September	12.25	12.25	12.25
April	13.75	13.75	13.75	October	12.00	10.75	11.31
May	14.00	13.75	13.81	November	13.00	11.50	12.31
June	14.00	12.75	13.12	December	13.00	11.75	12.38

The sales during the early part of the season were fairly good, and up to May and June the business was considered good. About that time, however, the agitation of political questions began to attract the attention of the public, and in July there was a perceptible drop, a part of which was due to the usual shut-downs of some of the furnaces. The shut-downs were of longer duration than ordinarily, however, the furnacemen preferring to wait until political questions were settled. When the election was over furnaces were put in blast every week, until the first of December, when out of the 28 furnaces in this section 13 were in blast, with a weekly capacity of 1184 tons. The other 15, with a weekly capacity of 18,558 tons, were out of blast, but ready for an early start.

NEW YORK.

New York is not in any sense a primary iron market, nor an important distributing center in the iron trade. It is chiefly an agency market from which a certain district is supplied. The course of the local market in 1896 followed quite closely that of the general trade, and no unusual changes are to be noted. For the greater part of the year dullness and depression prevailed, resulting from the financial conditions. Sales were limited by the smaller needs of the shops and foundries, and there was a general tendency throughout the year to limit buying to immediate necessities and to avoid carrying large stocks.

The structural iron market, usually an important section in the city, felt the effects of the depression. Some large buildings were begun, but other projects were held back on account of the difficulty of financing them, and the building record was rather a limited one. At the beginning of the year a good deal of local cable and electric street railroad work was promised, but it failed to come out, on account of financial difficulties, and most of it was postponed to 1897.

The features of the year, if they were of sufficient importance to be called features, were the increased sales of Southern pig iron in this district, and the low prices of most kinds of merchant iron and steel. For a large part of the year there was so much shading and allowance of commissions, etc., that it was difficult to make reliable quotations.

Upon the whole local dealers and brokers can hardly recall a year in which they did so much work for so small a margin of profit.

PHILADELPHIA.

While Philadelphia does not equal Pittsburg as a market, it comes second to that city as a distributing point, and controls the output of a large and important district. The leadership passed years ago from Eastern to Western Pennsylvania, but the former is still an active iron country, and Philadelphia is its leading point. The city itself is a very large consumer of iron and steel, its varied industries demanding a large supply of material.

The course of business during 1896 followed closely the general conditions of the iron market. The activity at the opening of the year gradually gave way to dullness, and for the last half of 1896 transactions were limited in their range and prices were weak. Up to June there was a fair amount of trade, though prices were not maintained as a rule; but from the opening of July to November buying was on a limited scale. After the election there was a slight spurt, but it did not last long, and the market soon subsided into a quiet condition again, every one looking forward to the future, but hesitating about the present.

The anthracite blast furnaces of Eastern Pennsylvania were less able to stand competition than the Western coke furnaces, and many of them went out of blast. The rolling mills and iron works were affected by the high price of steel and there was less work done in them, as a rule, than at any time since the months immediately following the panic of 1893. Only those which had contracted early for their supplies of raw material were able to keep up the

competition. Many of their managers have been doing a good deal of thinking, and the results may be apparent in new basic steel plants before long.

Locally the great locomotive shops and other industries depending on railroad work were very quiet and took less than their usual supplies. The city foundries were generally short of orders during the second half of the year. For structural and bridge material demand was a little better, but still below normal proportions. The most active work was found in the ship-building yards, where naval contracts helped to make up a very fair year's business. Nearly all, however, had a struggle to get through the year, and were not sorry when it ended.

PITTSBURG.

In the iron and steel trade 1896 will long be remembered as a remarkable year, showing the wonderful workings of the iron and steel products and the large increase in the mammoth plants in various parts of the country; works that a few years ago were considered of large capacity appear very insignificant at the present time. The year 1896 was not a very favorable one compared with the preceding one, the volume of business showing a large falling off. The Bessemer sales in Pittsburg for 1895 reached the large amount of 1,915,545 tons. The total sales for 1896 of all descriptions were 3,597,323 tons, being largely in excess of any year in the history of the United States. The sales of Bessemer from January to June 30, 1896, was 574,871 tons; billets, 268,200 tons; all descriptions reached 1,295,764 tons; but the second half of the year showed a very great falling off. Prices for all kinds of products have been irregular and uncertain. Pig iron is lower than in England. The South has been making large shipments to European markets. This is a great change in the situation and shows how this country is progressing in the way of manufacturing.

The following table shows the sales of raw iron and steel reported in Pittsburg for the year:

SALES OF BESSEMER PIG IRON, STEEL BILLETS, AND TOTAL SALES AT PITTSBURG.

	Bessemer Pig.	Steel Billets.	All Kinds.
First six months.....	574,871	268,200	1,295,764
Second six months.....	341,350	108,700	767,129
Total.....	916,221	376,900	2,062,893

It will be seen that of the transactions the first half of the year witnessed 64% of the sales of Bessemer pig, 71% of those of billets, and 65% of those of all kinds. The proportion would have been still greater if some heavy purchases of pig iron for speculation had not come in November and December.

As Bessemer pig iron and steel billets are regarded as leaders of the market, we give on the following page the weekly prices of those descriptions of material throughout the year:

WEEKLY PRICES OF BESSEMER PIG IRON AND STEEL BILLETS IN PITTSBURG FOR 1896.

Week.	Price Per Ton.		Week.	Price Per Ton.	
	Bessemer Pig	Steel Billets.		Bessemer Pig	Steel Billets.
January 4	$10.85@11.50	$16.00@16.50	July 4	$12.40@12.65	$19.75@20.25
" 11	11.25@11.50	16.10@16.50	" 11	12.25@12.60	19.00@19.75
" 18	11.75@12.60	16.50@17.50	" 18	12.20@12.40	19.35@19.75
" 25	12.75@13.35	17.25@18.60	" 25	11.70@12.00	19.10@20.00
February 1	13.25@13.50	17.75@18.25	August 1	11.25@11.50	19.50@19.80
" 8	12.75@13.15	17.50@18.25	" 8	11.25@11.75	18.90@20.25
" 15	13.00@13.50	17.35@18.25	" 15	10.75@11.25	19.25@20.25
" 22	12.85@13.50	17.50@18.00	" 22	10.90@11.15	19.75@20.25
" 29	12.40@12.50	17.50@17.85	" 29	10.75@11.20	18.75@20.25
March 7	12.50@12.60	17.40@17.65	September 5	11.00@11.75	19.50@20.25
" 14	12.40@12.75	17.30@17.75	" 12	11.75@12.00	18.75@20.25
" 21	11.85@12.25	17.15@17.50	" 19	11.50@11.85	19.50@20.25
" 28	12.35@12.50	17.25@17.85	" 26	11.50@12.00	19.50@20.25
April 4	12.50@13.75	17.40@20.00	October 3	11.50@12.00	19.50@20.25
" 11	13.60@13.85	19.85@20.25	" 10	11.40@11.50	19.40@20.25
" 18	13.25@13.75	19.50@20.00	" 17	11.85@12.00	19.80@20.25
" 25	13.40@13.60	19.40@20.25	" 24	11.90@12.10	19.90@20.25
May 2	13.00@13.40	19.35@20.25	" 31	11.75@12.25	19.60@20.25
" 9	12.75@13.00	19.50@20.00	November 7	12.25@12.75	19.85@20.25
" 16	12.65@13.00	19.50@20.25	" 14	12.40@12.60	20.00@20.25
" 23	13.00@13.25	19.65@20.25	" 21	12.35@12.75	19.60@20.25
" 30	12.50@13.25	19.25@20.25	" 28	12.10@12.30	19.50@20.25
June 6	12.50@13.35	18.75@20.25	December 5	12.00@12.20	19.50@20.25
" 13	12.30@12.75	19.35@20.25	" 12	11.75@12.00	19.50@20.00
" 20	12.50@12.75	19.00@20.25	" 19	11.75@12.00	16.50@17.00
" 27	12.40@12.50	19.50@20.25	" 26	11.50@11.75	15.00@15.50

Values show a wide range. For billets the comparative steadiness from April on shows the working of the steel pool; the sudden break in December was the result of the withdrawal of the pool prices. The highest price quoted on Bessemer pig was $13.85, the lowest $10.75, a difference of $3.10. The closing prices were better by $1 a ton than the low quotations of August and September.

We give below a table showing the monthly average prices of all the important kinds of iron and steel in Pittsburg during 1896:

PRICES OF IRON AND STEEL IN PITTSBURG IN 1896.

Month.	Pig Iron.			Ferro-manganese.	Steel Billets.	Wire Rods.	Sheet Bars.	Muck Bars.
	Bessemer.	No. 1 Foundry.	Gray Forge.					
January	$10.85@13.35	$12.75@13.75	$10.50@12.25	$52.32	$16.00@18.60	$23.79	$18.19	$19.91
February	12.40@13.50	12.50@13.25	11.00@11.50	51.79	17.50@18.25	23.53	19.73	21.00
March	11.85@12.75	13.00@13.75	10.70@11.25	49.85	17.15@17.85	22.59	19.84	20.73
April	12.50@13.85	13.00@13.50	10.90@11.50	49.95	17.40@20.25	25.94	22.38	21.22
May	12.50@13.40	12.25@13.25	10.75@11.15	50.33	19.25@20.25	27.63	22.21	21.18
June	12.30@13.35	12.40@13.30	10.60@11.00	50.10	18.75@20.25	26.41	22.36	20.73
July	11.70@12.65	12.00@13.00	9.75@10.75	49.33	19.00@20.25	25.68	22.50	20.38
August	10.75@11.75	11.75@12.50	9.50@10.00	49.08	18.75@20.25	25.13	22.23	19.63
September	11.00@12.00	11.65@12.20	9.50@10.16	49.41	18.75@20.25	25.50	22.58	19.79
October	11.40@12.25	11.60@12.25	9.75@10.25	49.33	19.40@20.25	24.29	22.19	19.90
November	12.10@12.75	11.75@12.25	10.20@10.80	49.42	19.50@20.25	24.75	22.39	19.99
December	11.75@12.20	12.00@12.25	10.00@10.60	49.19	15.00@20.25	24.19	22.50	19.91
Year	$10.75@13.85	$11.60@13.75	$9.50@12.25	$50.00	$18.04@19.75	$24.95	$21.59	$20.36

The range for No. 1 foundry iron was from $11.60 to $13.75, a difference of $2.15. Gray forge, which was quoted at $12.25 in January sold down to $9.50 in August and September. The price recovered a little before the end of the year.

For purposes of comparison the following table will show the prices of iron

and steel in December for five years. The quotations given are of the second and third weeks in the month.

DECEMBER PRICES OF IRON AND STEEL AT PITTSBURG FOR LAST FIVE YEARS.

	1892.	1893.	1894.	1895.	1896.
Bessemer pig	$13.75@13.80	$11.00	$10.25	$12.75	$11.80
No. 1 foundry	14.25@14.50	12.25	11.40	14.25	12 25
No. 2 foundry	13.25@13.50	11.25	10.40	13.50	11.50
No. 3 gray forge	12.60@12.65	10.00	9.50	12.00	10.40
No. 4 gray forge	12.50@12.60	10.00	9.25	11.75	10.25
White iron	12.00@12.25	10.60	9.00	9.00	9.75
Mottled	12.00@12.30	10.25	9.00	9.00	9.75
Silvery No. 1	10.00@10.25	13.50	13.60	14.80	9 80
Ferromanganese 80%	60.00@62.00	52.50	49.50	54.00	49.25
Charcoal pig—					
No. 1 foundry	20.00@20.50	18.50	17.50	17.80	17.50
No. 2 foundry	19.00@19.50	18.00	16.25	17.00	16.00
Cold blast	25.00@26.50	25.00	23.50	23.00	23.25
Warm blast	19.00@19.50	18.00	16.50	16.00	16.00
Spiegel 20%	28.50@29.00	23.75	18.50	18.00	17.00
Muck bar	24.50@24.75	21.00	18.40	21.00	20.00
Steel billets	22.00@23.00	16.75	15.15	16.50	20 00
Steel rail ends	15.50@16.00	11.50	10.50	14.00	13.15
Steel bloom ends	16.00@16.25	12.00	10.45	14.00	13.00
Old iron rails	20.00@21.00	14.50	12.50	15.00	16.00
Old steel rails	14.50@15.00	10.50	10.00	13.00	13.00
No. 1 wire scrap	20.00@21.00	16.25	10.00	14.00	13.00
No. 2 wire scrap	15.00@15.50	15.00	9.50	12.50	11.00
New steel rails	30.00@31.00	24.00	22.00	28.00	28.00
Bar iron	1.60@ 1.65	1.45	1.15	1.25	1.20
Iron nails	1.55@ 1.60	1.10	1.00	1.25	1.50
Steel nails	1.55@ 1.40	1.10	1.00	1.25	1.50
Wire nails	1.55@ 1.60	1.20	1.00	1.30	1.65
Coke at furnace	1.90@ 1.95	1.15	1.00	2.00	2.00

The prices of 1896, though above the range of 1893 and 1894, were lower than had ever ruled before 1893. Unfortunately their low range did not much tempt buyers at the time.

Sales and Prices of Iron Ore.—The following figures show the sales and prices of iron ore for the past five years. The figures for 1896 show a material advance over the three preceding years. In 1892 the sales were 5,752,000 tons at prices ranging from $4 to $5 per ton. Sales were all made deliverable on the docks at Cleveland and Lake Erie; this will be continued until the canal is completed between Pittsburg and Cleveland, when our mills will be supplied direct from the mines at much less than the present cost.

SALES OF IRON ORE AT PITTSBURG.

Date.	Tons.	Quality.	Price.
1896	3,300,000	Bessemer	$3.75@4.10
1895	4,300,000	Bessemer	2.90@3.75
1894	2,950,000	Bessemer	2.25@3.25
1893	4,010,000	Bessemer	2.75@3.00
1892	5,752,000	Bessemer	4.00@5.00

The high prices of 1896 were due to the ore pool, which succeeded in maintaining its quotations, though the furnacemen protested very strongly against it.

Coke Prices.—Coke prices through the year have been on the basis of $2 per ton, f. o. b. at ovens, for furnace coke. Dealers, however, have made contracts right along at $1.75 per ton, and even at lower rates. Large consumers have made their own rates also.

Southern Iron Ore and Coal.

By William M. Brewer.

It is almost necessary, especially in the Southern States, to consider the industries of mining iron ore and coal under one heading, because with the exception of the limited supply of coal mined exclusively for domestic purposes, the remainder is employed for the manufacture of coke to be used in the blast furnaces. During the past year there was a falling off of the production as it was returned for 1895. Owing to the depressed condition of business and the agitation in politics during the twelve months it is only natural that such a decrease should have occurred.

The production of pig iron during 1896 very closely approached that for 1895, when the Birmingham district produced a larger tonnage than had been reported for that district in any one year before. This production has not resulted from the fact of any increase of the number of furnaces in blast, but rather from the greatly improved practice in metallurgy. To-day there are furnaces with a guaranteed capacity of 125 tons of pig iron per day, which are actually producing from 180 to 200 tons per day. While the production of pig iron has so closely approached that during 1895, yet the output from the iron-ore mines, so far as the limonite or brown ore is concerned, has considerably decreased, while the output of red hematite from the Birmingham district has increased in proportion.

As is generally known, the bulk of the limonite ores mined in Alabama and Georgia comes from Barton and Polk Counties in Georgia, and Cherokee, Calhoun, and Talladega Counties in Alabama. Reports from all these counties show a considerable decrease in the output during 1896, as compared with that during 1895. The main cause for this has been the lack of water supply during the past summer. In consequence of this many of the ore washers were run on considerably less than full time and some were shut down entirely. This condition is being met by some of the companies in the most practical way possible—that is, by boring deep wells with the expectation of obtaining a sufficient supply in future to overcome any deficiency caused by such an extremely dry season as the past has been.

The production of charcoal iron in Alabama and Georgia was practically very light. One furnace, that at Rock Run, Ala., was in blast continuously during the year, and the Shelby furnace a portion of the time. The production from these two represents the manufacture of charcoal pig iron in the two States for 1896, and the quantity produced aggregated less than probably during any year since the industry was started. The reason for this condition has been the lack of demand for this grade of iron. In the past only charcoal iron was used in the manufacture of car wheels, but to-day a mixture of charcoal and coke iron is almost universally used by the car-wheel manufacturers, and it is freely asserted that some are manufacturing wheels from all-coke iron. Of course while such conditions exist the charcoal furnaces will remain out of blast, and many companies will adopt the same policy as has been adopted by the Clifton Iron Company at Ironaton, Ala., which has rebuilt one of its charcoal furnaces and so altered its construction as to make it one of the most

modern types of coke furnaces, with a guaranteed capacity of 125 tons of pig iron per day, and probably an actual capacity of from 175 to 200 tons per day; which has been demonstrated to be possible by the practice in the Birmingham district.

Considerable interest is being manifested by the iron-masters with regard to the adaptability of Alabama coal for the manufacture of coke in by-product ovens. Up to the present time no attempt has been made to construct any of these in the State, but the most important companies are thoroughly investigating the subject.

The German Iron Market in 1896.

By E. Schrödter.

During the year 1896 the activity which began in 1895 continued. The mills and furnaces were everywhere busy, and the results of the year were generally satisfactory to the producers.

The demand for pig iron was so great that the furnaces, in spite of an increased capacity and working with full force, were unable to supply the steadily increasing demand, and a scarcity of pig iron was frequently felt. As early as March the total production of the blast furnaces was disposed of for the second and third quarters, and that for the fourth quarter was in large part contracted for. There were no stocks of pig iron at the furnaces after the beginning of summer. The pig-iron syndicate, which was re-established during the course of the year upon an improved basis, raised its quotations to correspond to the advanced prices of all raw materials, and found ready takers.

The production of pig-iron during the year 1896 amounted to 6,360,982 metric tons, of which about one-half came from the Rhenish-Westphalia district; the increase over 1895 was 488,194 tons, or 8.8%. The prices for pig-iron were as follows:

	January.	April.	July.	October.	December.
	Marks.	Marks.	Marks.	Marks.	Marks.
Foundry No. 1 and hematite	65.00	65.00	65.00	66.00	67.00
Foundry No. 3	56.00	57.00	57.00	58.00	60.00
Forge pig No. 1	51.00	52.53	54.00	57.58	58.59
Luxemburg forge pig iron	44.80	44.80	44.80	47.20	48.80
Thomas iron	52.00	56.00	56.00	57.20	59.80
Spiegeleisen 10 to 12% m	55.00	59.00	63.00	64.00	65.00

The following shows the number of blast furnaces in operation during the year: Upper Silesia, 29; Wurtemberg, 1; Bavaria, 3; Central Germany, 8 (including charcoal); Rhenish Westphalia, 52; Central Rhine, 9; Siegerland, 28; Dill and Lahn district, 6; Saar district, 19; Lorraine, 29—total, 184. There were 44 reported out of blast.

In the Rhenish Westphalia district there are four new blast furnaces of large capacity in the course of construction, two by the Deutscher Kaiser Company, which are approaching completion, and two by the firm of Fried. Krupp, of Essen, which are to be in operation by October 1, 1897. Both these

will be run in connection with basic steel plants. In Stettin a blast furnace for Swedish ores and English coal is in course of construction.

For blooms and billets the conditions of the market were the same as for pig iron; the products of the steel plants met with rapid sale, and while only a short time ago they had great difficulty in disposing of their enormously increased production, in 1896 consumers were compelled to hurry in their orders. Sales of future supplies extend far into 1897.

Since the demand for billets in the German market could not be met at times, and since a want of material was frequently experienced by the rolling mills, it is but natural that the exports should have decreased somewhat, and the total loss was about 20%, as against the preceding year.

The finished products of the iron industry did not fully participate in the general increase; the situation of the wire mills especially was not what it should have been. Business was throughout fairly active at advanced prices for wire, but was less favorable for wire products. For plates, however, business was very good during the whole year at advanced prices, and the rolling mills will be busy for some time to come. Quotations were as follows:

	January.	April.	July.	Oct.–Dec.
	Marks.	Marks.	Marks.	Marks.
Boiler plate, wrought-iron	160.00@175.00	165.00@175.00	175.00	177.50
Boiler plate, steel	125.00	140.00	150.00	152.50
Fine plate	130.00@140.00	145.00@150.00	145.00@150.00	145.00@155.00

The bar iron market for all sorts was exceedingly brisk, and activity increased from month to month, until late into the fall, so that the rolling mills were compelled to extend their time of delivery.

There was a lively demand also for all other kinds of bar steel, as well as wrought iron. The price for ordinary bar iron, which at the beginning of the year was 108 marks, raised by April to 120 marks, and has since attained to 131 marks per ton.

Railroad material of all kinds was in brisk demand. Contracts for large quantities of rails and wheels for South Africa and the Dutch East Indies were placed at the beginning of the year, to which were added in the spring the very important quantities required by the State roads. Beside these, builders of railroads gave out contracts, so that the mills had plenty to do during the whole year. The exports of rails showed a gain of 13%.

The iron foundries and machine shops had plenty of work throughout the whole year, and part of them were compelled to stipulate for late term deliveries. The demand is still very brisk, and nearly all the foundries have plenty of orders on their books for a long time to come.

Iron Ore.—Inquiry in the ore market was extremely brisk, especially for spathic iron ore, so that almost the whole of the 1897 production was disposed of before the end of 1896. In consequence of the great demand, prices advanced from 9.50@10.50 marks in January to 13.00@13.50 marks in April, and 14.40@16.00 marks in July. The latter price held until the close of the year. The demand for red hematite ores was also very lively.

The Manufacture of Basic Iron in Alabama.

By William B. Phillips.

The production of basic pig iron for the open-hearth steel furnace has become an important industry in the United States. It has had a rapid growth in the northern and north central iron and steel districts, and is recognized as a competitor of the Bessemer and acid open-hearth processes. The competition will probably become in the immediate future still more formidable. The discovery in Minnesota of enormous deposits of non-Bessemer iron ore, which can be cheaply mined and transported, must lead to the establishment of basic pig and steel plants in the Northwest, because the exhaustion of the more accessible Bessemer ores of the Lake regions steadily and rapidly proceeds by the increased demand made upon them. Indeed, it is an open question if we have not seen the high-water mark of the output of Bessemer and other acid steels.

So long as an abundant supply of Bessemer ore was assured at fair prices, the attention of the steel-makers was, in a measure, restricted to ores which would yield pig iron containing not more than 0.1% phosphorus. The extraordinary development of the demand for steel rails, bridge and structural steel of great strength and ductility, and above all the growing disuse of wrought iron, have combined to stimulate the production of Bessemer steel. But it has now been conclusively proved that this old favorite can no longer hold the field it once occupied.

It is no longer a *sine quâ non* for the making of good steel that ore of less than 0.05 parts of phosphorus per 50 parts of iron shall be used, and consequently it is toward the more phosphoritic ores that attention is being directed. This is well, for assuredly it would not be wise to wait until the increasing price of Bessemer steel, due to the increasing scarcity of Bessemer ores, should bring us into an awkward situation. Hence more interest is manifested in the manufacture and use of basic steel than ever before, and by far the greater tonnage of steel works built during the last 3 years consists of basic open-hearth.

Some may argue that the investments in the non-Bessemer Lake ores are compelling capitalists to provide an outlet for their product. There may be a modicum of truth in this, but the non-Bessemer ores could not be sold to steel-makers at any price if it were not perfectly feasible to utilize them, not merely as a mix, but of and for themselves.

Two classes of ore represent the extremes of chemical composition in respect to their applicability to steel making—the high-grade Bessemer ores with phosphorus below 0.05%, and the highly phosphoritic ores with phosphorus not below 1%. The first find their adaptation in the acid Bessemer and the acid open-hearth processes, the latter for the most part in the basic Bessemer or Thomas process. Between them lie fully three-fourths of the iron ore deposits of the world—ores which yield pig iron carrying from 0.1% to 1% of phosphorus. In this country the Thomas process, based on the use of pig iron containing from 2 to 3% of phosphorus, has had a very limited application. The Pottstown Iron Company, Pottstown, Pa., was until this year, when the Troy Steel and Iron Company began operations, the only large establishment that attempted its manufacture. The quality of the steel made was excellent, and there was a fair market for the

slag as a phosphatic fertilizer, but the use of this process has not extended, and practically all the steel made in the United States, over 6,000,000 tons annually, has been made from pig iron of less than 0.1%, or not more than 1% phosphorus. Unfortunately the exact statistics cannot be secured, since the production of acid open-hearth and basic open-hearth steel are not given separately, but it is well known that a very large percentage of the increase is to be credited to basic metal.

PRODUCTION OF OPEN-HEARTH STEEL INGOTS IN THE UNITED STATES.

(In tons of 2240 lbs.)

	1890.	1891.	1892.	1893.	1894.	1895.	1896.
Bessemer.....	3,688,870	3,247,417	4,168,435	3,215,686	3,571,313	4,909,128	3,919,906
Open-hearth...	513,232	579,753	669,889	737,890	784,936	1,137,182	1,298,700

During these 6 years the production of open-hearth steel has increased by 121%. It is probable that at least 50% of the increase was in basic metal. During the same period the production of Bessemer steel increased 33.1%, and it is likely that the production of open-hearth steel, especially basic open-hearth, will increase at a much faster rate within the next 6 years.

The purpose of the present paper is to direct attention to the manufacture of pig iron suitable for the basic open-hearth furnace from materials not hitherto considered as very promising, viz., the ores, fluxes, and fuels of Alabama. It has also been thought best to restrict these remarks to the Birmingham district in Alabama, because it is only here that the output of basic iron has attained large proportions, and that a great number of analyses of stock and product have been made every day for a year. While it is true that furnaces elsewhere in the South, especially in Virginia, have made basic iron, and can do so successfully, it is believed that the results will not differ essentially, except in respect of cost, from those on record here. The analysis of the tables of production and cost accounts covering 75,000 tons of basic iron in the Birmingham district will represent the industry under conditions as favorable as can be expected anywhere in the South. It is assumed that the vital difference between any two districts in the South would be in respect of cost, and not of quality. The quality of the iron made would depend to a great extent upon the specifications of the contract, for it is obvious that if these are severe the risk of increased percentages of casts not suitable for shipment, and the cost of production, become greater.

Generally speaking, Southern iron men at present, and Southern steel men in the future, must work with ores that put from 0.3 to 0.8% of phosphorus in the iron. The ores are far too high in phosphorus for Bessemer metal, and far too low for Thomas metal. The Southern iron trade has been marvelously developed during the last 10 or 15 years, and costs of production have been forced to a point not anticipated by the keenest observers, but it has been built up on material not intended for the steel-maker, but for ordinary foundry, mill and pipe irons. It is to the same iron that we must look if we are now to make steel. There must be less silicon and sulphur in the iron, elements within easy control of an experienced furnaceman, but it will otherwise be the same as that turned out at present every day.

Under specifications likely to continue, the maximum silicon must be 1%, the maximum sulphur 0.05%, while phosphorus will remain about 0.75%, though it is possible to lower it by perhaps 0.05%, and under exceptional circumstances by 0.1%. For the most part it will vary from 0.7% to 0.85%. A large number of analyses of basic iron made under many different burdens has shown that while a maximum phosphorus of 0.75% may be attained, yet it is not thought that the extra 0.1% constitutes a source of serious danger to the maker of basic steel, though some steel men may be inclined to insist upon a maximum phosphorus of 0.75%, more especially if they are using the basic pig as a mix for very low phosphorus iron. The amount of phosphorus allowable in basic stock is to some extent controlled by the exigencies of the trade, and is subject to greater variation than silicon or sulphur. In many cases it may reach 1%, and considerable shipments have been made with phosphorus that high, while on the other hand large shipments have been made with maximum phosphorus 0.75%. If concentrates or other rich ores were used in the furnaces it would, of course, diminish the phosphorus content in the pig iron, not because the phosphorus is removed from the ore but because the amount of ore required per ton of iron is lessened. For instance, if it requires on the average 2.30 tons of ore to make a ton of iron, and the ore contains 0.32% of phosphorus, we would expect to find in the iron 0.73%. But if the amount of ore per ton of iron were 2 tons, and the phosphorus in the ore was 0.32%, as above, we could expect to have 0.64% in the iron.

It is a well known fact that Alabama ores, so far as explored, cannot be used in the production of Bessemer pig iron. Isolated bodies of brown ore (limonite) suitable for this purpose may be found here and there, but no one who has had experience with such ores would think of founding upon them an industry of this kind, for they are too unreliable in composition. They are generally richer in iron and almost always lower in silica than the hematites. In the production of basic iron they play a very important part, as will hereafter appear. Alabama is devoid of Bessemer ore, so far as now known, and it is also not to be considered as a source of ore suitable for Thomas pig, unless certain deposits of fossiliferous and brown ore, not fully explored, should be found to be workable. These fossiliferous ores contain from 1.5 to 5% of phosphorus, and would be suitable for the production of Thomas pig, though they carry nearly 40% of sand, which, however, can be easily and cheaply removed by the Wetherill process, and the iron brought up to 52 to 55% of iron.

Such ores as Alabama can be depended upon to yield for years to come, so far as is positively known, can be made into steel only by the basic open-hearth process. Even the duplex process, with its more or less successful conjoining of the Bessemer and the open-hearth, must be excluded in the light of the experience of the last 12 months. We may even say that we shall not need to desiliconize the pig iron after it has come from the blast furnace. This operation can now be conducted in the blast furnace itself with a certainty and a regularity that renders any further experiments with desiliconizing processes wholly unnecessary. In the interval between the production of the first Birmingham basic iron on regular order in 1888, and the fall of 1895, when very large orders were taken under stringent specifications, there was more or less doubt among even the best informed and most progressive iron men of the possibility of producing first-class

basic iron from local materials. There was always the reservation of being obliged to use some desiliconizing process to lessen the silicon, and the sulphur also, to a great extent in the pig iron after it had left the blast furnace. It was with some misgivings that it was finally decided to attempt the production of basic iron in the blast furnace itself with local ore, flux, and fuel. It is true that some basic iron was made here in the spring of 1888 for use in the Henderson open-hearth furnace at North Birmingham; and again in the winter of 1892, and the spring of 1893, for the Jefferson Steel Company, successors to the Henderson Steel Company; but the orders were not large, and the iron was made almost exclusively from brown ore. When it became possible to secure large orders it was recognized that the iron could not be made from brown ore, because an adequate supply was not obtainable, and even if it had been, the cost of production would have left a very narrow margin, or none at all, of profit. If pig iron suitable for conversion into steel cannot be made from local hematite ore, the manufacture of steel at Birmingham will never become a commercial fact. There may be places in the South where brown ores can be secured in quantity, quality, and price that would warrant the production of basic iron from them exclusively, but this is not true of the immediate vicinity of Birmingham, and it is here that the cheapest iron will be made for many years to come.

The true significance of the production of the 75,000 tons of basic iron made here during the last year lies not in the fact that it was made at a cost that allowed it to be sent to distant markets, but in the fact that it was made of ores that can be put into the stockhouse for 60c. and $1 per ton. The production of these 75,000 tons of basic iron is therefore one of the noteworthy occurrences in the development of the iron and steel industry of the United States.

Perhaps the commonest criticism of Alabama iron was that it carried too much silicon rather than too little. With the silvery irons showing over 5% of this element, and the foundry grades at times ranging from 2.75 to 3%, the tendency was toward high-grade softeners, with sulphur from 0.03 to 0.05%. When the furnaces were working cold, and mill and mottled irons were made, the difficulty was to keep the sulphur down. When the silicon fell to less than 1.5% the sulphur increased, and when the silicon was below 1% the sulphur was generally above 0.1%.

This was the situation when it was asked if large orders for basic iron could be taken under the following specifications: Maximum silicon, 1%; maximum sulphur, 0.05%; maximum phosphorus, 1% in some cases, 0.85% in others, and 0.75% in still others. After careful consideration the orders were taken, and during the last 13 months (September, 1895, to September, 1896, inclusive) every cast was analyzed for silicon, sulphur, and phosphorus; very large shipments were made, and not a single carload was rejected. Considering the nature of the ores used, the irregularity of the stock, and the inevitable mishaps attending the prosecution of a new business, the success attained is certainly remarkable. Excluding the relatively small percentage of casts that showed either too much silicon or too much sulphur, the average silicon, sulphur and phosphorus in 1188 casts was as follows: Silicon, 0.51%; sulphur, 0.032%; phosphorus, 0.72%. In manganese the metal carried about 0.5%, graphitic carbon 2.75 to 3%, and combined carbon 0.6 to 0.8%.

The ore used was of three kinds, no single one being employed, and for the

most part the three together, viz.: Hard or limey ore, soft or lime-free ore—these two being hematites—and brown ore or limonite.

The Hard or Limey Ores.—These are red fossiliferous hematites of the Clinton formation. They carry on the average:*

	Per Cent.
Water	0.50
Metallic iron	37.00
Silica	13.44
Lime	16.20
Alumina	3.18
Phosphorus	0.37
Sulphur	0.07
Carbonic acid	12.24

For the most part the hard ore carries enough lime to flux its own silica, although it not infrequently happens that there is a deficiency of lime. If the ore were always self-fluxing it would enable the furnaceman to use much less additional lime. It is necessary to keep a close watch over the hard ore, even from the same mine, on account of the lime-silica ratio, and because the iron decreases with an increase of lime. Some hard ores carry from 3 to 6% more lime than silica and alumina, and it is necessary to know the composition to proportion the added flux. This is supremely important, because the desiliconizing and desulphurizing action maintained within the blast furnace is dependent for its success upon the basicity of the slag, and this in turn conditions the fusibility of the slag. The action of the very basic slag in the furnace upon the iron that trickles down through it may be compared to certain desiliconizing and desulphurizing processes for the treatment of high-silicon, high-sulphur pig iron on its way from the blast furnace to the steel furnace. Instead of pouring molten pig iron, whether taken direct from the furnace, or remelted, through a molten bath of very basic material, and removing the silicon and sulphur in this manner, the process is carried on in the blast furnace, and there is no longer a necessity for an intermediate process; though it must be confessed that the methods proposed and indeed tried in the Birmingham district for improving the quality of the iron intended for the basic open hearth after it had left the furnace were very ingenious and effective. We do not now speak of the improvement in the iron due to the substitution of cast-iron moulds for sand moulds, but only of essential changes within the body of the iron due to the elimination of certain impurities.

It is in respect of the basicity of the slag that the intelligent use of hard ore becomes so important. The better grades of this are an ideal ore in one particular; it contains the flux in a state of perfect admixture with the material to be fluxed. No mixture of lime and oxide of iron, prepared artificially, could be any better than the ore which nature has prepared.

The Soft or Lime-Free Ores.—An average analysis of stockhouse ore is as follows:†

	Per Cent.
Water	7.00
Metallic iron	47.24
Silica	17.20
Alumina	3.35
Lime	1.12
Phosphorus	0.30
Sulphur	0.06

* This analysis is taken from the writer's *Iron Making in Alabama*, 1896, p. 40. † *Ibid.*, p. 32.

By far the greater part of the coke iron made in Alabama has been produced from ore mixtures carrying large amounts of soft ore. There was a time when but little hard ore was used, and the value of the limey ore was not recognized. Indeed, when it was encountered, in sinking a slope on the soft ore, the mine operators were disposed to abandon the property, thinking that the ore had given out.

The cheapness of the soft ore, 10c. per ton, as compared with hard ore at 40c., and the fact that it carries from 10 to 12% more iron than the hard ore, have together caused its general employment. It is quarried rather than mined, as nearly all of the workings are in open cut, by the bench system. At some places regular mining operations are conducted underground, and both the soft and the hard ore are won by drift, slope, pillar, and room. But the quantity of soft ore obtained under cover is trifling as compared with what is in effect quarried. The seams vary in thickness from 4 to 20 ft., the thinner seams being worked only when the iron is above 50%. The present practice is to remove the overburden of soil, slate, sandstone, and thin seams of poor ore, that is, from 10 ft. to 30 ft., and to mine from the seam such ore as is suitable. The big, or Ishkooda seam, is about 20 ft. thick, and the upper 10 ft. of it carries about 47% of iron. Below this the seam loses in iron at the rate of about 0.5% per ft., and the silica increases rapidly, although there is no regular rule. It has not been found advisable to take more than 10 ft. of this seam. Under cover the soft ore gradually changes into hard ore by mixture with carbonate of lime, and when the plane of atmospheric decomposition is passed, which is at depths along the dip of from a few feet to 300 ft., the seam is known as hard ore. The soft ore is therefore merely the upper part of the hard ore from which the lime has been removed by leaching.

From the very complete records at my disposal it must be said that the best results in the production of basic iron have been attained by the use of the trinity of ores—hard, soft, and brown. No attempt has been made to produce it from hard ore exclusively on a regular commercial scale, or from hard and brown, or from soft and brown ore. We have the results from the use of hard, soft, and brown, and from hard and soft, with no brown ore. There is very little use in discussing the adaptability of brown ore alone for this purpose, as it is already known that it is suitable, and also that the cost would probably be prohibitory, even if it could be obtained in sufficient quantity. The best practice will therefore be to continue the use of all three of these ores for the present, while striving to increase the proportion of hard ore.

The low cost of basic iron in the Birmingham district is the strongest argument for the establishment of steel works there. So long as ore suitable for producing this kind of iron can be laid down in the stockhouse for 1½ to 2½c. per unit of iron, the manufacture of basic iron, and consequently the manufacture of basic steel, will be commercially profitable. This can be done from ore now in sight for many years to come. If the practice in the future is to be what it has been during the last year (it will probably be much better as experience is accumulated), there is enough brown ore to allow of the use of 20% for an equally long period.

Brown Ore or Limonite.—The brown ores (limonites) used in the production of

basic iron are those usually employed in the production of ordinary iron. An average analysis of stockhouse ore is as follows:

	Per Cent.
Hygroscopic water	7.00
Combined water	6.00
Metallic iron	48.54
Silica	11.22
Alumina	3.61
Lime	0.84
Phosphorus	0.38
Sulphur	0.09

If care be taken in the mining and washing of brown ore it is of a fairly uniform composition. No calcined brown ore has been used in making basic iron. Some basic iron was made in 1892–1893 from brown ore exclusively, but during the last twelve months it has been used to the extent of about 20% of the ore burden.

Good basic iron has been and can be made without using brown ore, but if it be omitted there is an increased risk of an excess of both silicon and sulphur. On this account, and when working under severe specifications, its use becomes a necessity. For instance, it was found that the best results were obtained by using an ore burden containing 20% of brown ore irrespective of the percentages of hard and soft, which may vary within wide limits. So far as concerns the ore burden the records cover a considerable range, from 36.1% hard, 42.6% soft, and 21.3% brown, to 64% hard, 36% soft, and no brown. Of 30,222 tons of iron specially examined with reference to the ore burdens on which it was made, the brown ore showed the following percentages, viz.: 0, 8.9, 10.6, 14.5, 19.1, 20.0, 20.1, 20.3, 21.1, 21.3. When running exclusively on hard and soft ore the average silicon was 0.68%, the average sulphur 0.043%, and the average phosphorus 0.7%. With an ore burden of 52.3% hard, 27.4% soft, and 20.3% brown, the average silicon was 0.47%, the average sulphur 0.033%, and the average phosphorus 0.7%. Important as these differences between the silicon and the sulphur are, they become even more so when it is stated that the chances of running over 1% silicon with an ore burden containing no brown ore are nearly four times greater than when 20% of brown ore is used, and the chances of exceeding 0.05% sulphur are more than twice as great. Furthermore, the range of both silicon and sulphur is wider when brown ore is omitted than when 20% of it is used; lastly, the average consumption of coke per ton of iron with no brown ore was 1.53 tons, and with 20% of brown 1.19 tons. The saving of flux with increase of hard ore is a partial offset to these unfavorable results, but even after deducting this the balance is largely in favor of the use of brown ore.

The Fluxes.—The basic iron of 1892–93 was made with limestone as flux, but during the last 12 months dolomite has been exclusively employed. An average analysis of stockhouse samples is as follows:

	Per Cent.
Silica	1.50
Oxide of iron and alumina	1.00
Carbonate of lime	54.00—Lime..... 30.31
Carbonate of magnesia	43.00—Magnesia, 20.71

The experience with limestone was not favorable. It carries 4% silica and 53% lime, with 1% oxide of iron and alumina, on the average, but is subject to wider

variations in composition than the dolomite. The value of magnesia as a desiliconizer and a desulphurizer in the blast furnace is still somewhat in dispute, but the experience with dolomite here has proved beyond any question that it can, under our local conditions, be used with great advantage. Limestone of 1.5 to 2% silica cannot now be obtained here, and it is useless to discuss the question of its application to the production of any kind of iron. Dolomite has supplanted limestone to a very considerable extent in the Birmingham district within the past year. The amount of dolomite consumed, per ton of iron, varies, of course, with the amount of hard ore used. For basic iron the variation is from 0.12 ton with 81.2% hard and 18.8% soft ore to 1.08 tons with 36.2% hard, 53.2% soft, and 10.6% brown ore. When 0.12 ton was used the consumption of ore, per ton of iron, was 2.64 tons, and of coke 1.55 tons, the make of iron being 520 tons. When 1.08 tons were used the consumption of ore was 2.25 tons, and of coke 1.32 tons, the make of iron being 2068 tons. In making 7424 tons of basic iron the consumption, per ton of iron, was in tons: Ore 2.10 tons, dolomite 0.92 tons, coke 1.23 tons, the ore burden being composed, by percentages, as follows: Hard 36.1%, soft 42.6%, brown 21.3%, and the total burden: Hard ore 17.8%, soft 21.0%, brown 10.5%, dolomite 22%, coke 28.7%.

Fuel.—All of the basic iron made here is coke iron, no charcoal or splint coal being employed. During the last year most of the coke used was 48-hour "beehive," made of washed slack-coal. An average analysis, stockhouse sample, is as follows:

Coke.	Per Cent.
Moisture	0.75
Volatile and combustible matter	0.75
Fixed carbon	89.00
Ash	9.50
Sulphur	1.00

Ash of Coke.	Per Cent.
Silica	45.10
Oxide of iron	12.32
Alumina	31.60
Lime	1.50
Magnesia	Trace.
Sulphur	0.10

The ultimate strength of the coke is about 2000 lbs. for a 1-in. cube, and the compressive strain about 500 lbs. per cu. in. The apparent sp. gr. is 0.80, the true sp. gr. 1.80, the percentage of cells by volume about 45, and the volume of cells in 100 parts by weight about 47. In structure this coke is generally fine-grained and close, and breaks into lumps rather than "fingers." It is a small-celled coke with strong walls, and carries a good burden, 1 lb. carrying as much as 2.54 lbs.

The consumption of coke in tons per ton of iron varies from 1.56 when using 64.6% hard, 35.4% soft, and no brown ore, to 1.05 when using 60% hard, 20% soft, and 20% brown ore, the calculations being based, respectively, on 1221 and 2934 tons of iron. It is much better to state it in this way than to give the average over a longer period, during which the burden may be changed repeatedly. When no brown ore is used the consumption of coke is higher. For instance, with 64% hard and 36% soft ore, the make was 3521 tons, and the consumption of coke per ton of iron was 1.52 tons. With 81.2% hard, and 18.8% soft ore, it was 1.55, the make of iron being 520 tons. With 64.6% hard, 26.5% soft, and 8.9% brown ore, the make of iron was 1140 tons, and the consumption of coke was 1.24 tons. And, lastly, with 36.1% hard, 42.6% soft, and 21.3% brown ore, the make was 7424 tons, and the consumption of coke was 1.23 tons. Many other instances might be given, but these are sufficient for the present purpose.

Coke of this quality can be secured here in very large and regular shipments. Very great improvements have been carried out during the last few years, and the best coke now made here will compare favorably with the best made anywhere else in the United States, as has been substantiated by the most careful chemical and physical tests, embracing all kinds of coke and covering hundreds of thousands of tons, and by actual furnace records in making basic iron. We have the records of the production of more than 22,000 tons, in which the average consumption of coke in tons per ton of iron was 1.26. The coke is good and the furnace practice is good.

Furnace Burdens.—Let us now examine somewhat closely into the furnace burdens, and their effect upon the quality of the iron. The following table gives the results of the examination of the conditions attending the production of 30,222 tons of basic iron made during the year ending September 9, 1896. More than twice as much was made, but it is not necessary for a correct understanding of the matter to examine the returns from every ton of iron made. The examples selected are in nowise exceptional, but were chosen for the sole purpose of exhibiting certain types of furnace burdens. Every cast of iron made during the year was analyzed for silicon, sulphur, and phosphorus, and many also for manganese and the two carbons. The inspection of the iron was so rigid that, as already remarked, not a single shipment was rejected. For the first time in the history of iron-making in Alabama a critical chemical examination of every cast of iron was insisted upon, and it was faithfully carried out. The burden for each day was known, and entered upon the analysis sheet, as also the number of charges, the tons of iron made, the revolutions of the blowing engines, and the pressure and temperature of the blast.

ORE BURDENS OF HARD AND SOFT ORE.

(In tons of 2240 lbs.)

Reference No.	No. of Charges.	Percentages Of Ore Burden.			Percentages Of Total Burden.					Tons of Iron Made.	Consumption. Tons Per Ton of Iron.				1 lb. Coke Carries lbs. of Burden.	Silicon. Cast No.				Sulphur. Cast No.				Phosphorus. Cast No.			
		Hard.	Soft.	Brown.	Hard.	Soft.	Brown.	Dolo.	Coke.		Ore.	Dolo.	Coke.	Total.		1.	2.	3.	4.	1.	2.	3.	4.	1.	2.	3.	4.
1	1848	64.0	36.0	....	33.1	18.6	....	14.5	33.8	3521	2 34	0.66	1.52	4.52	1.97	.58	.68	.65	.69	.044	.038	.035	.044	.73	.73	.74	.74
2	659	64.6	35.4	....	33.3	18.1	....	15.0	33.6	1221	2.38	0.69	1.56	4.63	1.85	.54	.81	.86	.65	.043	.035	.037	.046	.69	.69	.70	.69
3	569	61.5	38.5	....	33.1	20.7	....	12.4	33.8	1122	2.30	0.54	1.48	4.34	1.96	.49	.54	.48	.46	.050	.042	.041	.049	.67	.67	.67	.68
4	279	81.2	18.8	....	49.2	11.1	....	5.3	34.4	520	2.64	0.12	1.55	4.31	1.91	.86	.74	.95	.94	.056	.046	.045	.042	.69	.71	.72	.71

ORE BURDENS OF HARD, SOFT, AND BROWN ORE.

5	570	64.6	26.5	8.9	35.3	14.4	4.8	16.0	29.5	1140	2.31	0.68	1.24	4.23	1.85	.60	.63	.60	.52	.023	.026	.031	.034	.76	.76	.76	.76
6	1099	36.2	53.2	10.6	17.8	26.2	5.2	22.0	28.8	2068	2.25	1.08	1.32	4.65	2.42	.48	.49	.49	.52	.029	.028	.028	.029	.72	.73	.72	.73
7	704	61.5	24.0	14.5	33.1	13.0	7.7	12.4	33.8	1383	2.36	0.54	1.48	4.38	1.97	.48	.51	.43	.43	.031	.029	.031	.033	.68	.69	.68	.68
8	497	61.8	19.1	19.1	35.5	11.0	11.0	15.0	27.5	1106	2.33	0.53	1.11	3 97	2.09	.36	.38	.36	.37	.041	.037	.036	.033	.71	.69	.69	.69
9	1244	60.0	20.0	20.0	35.3	11.7	11.7	11.7	29.6	2934	2.09	0.42	1.05	3.56	2.40	.48	.55	.49	.54	.039	.043	.039	.032	.72	.75	.75	.74
10	1306	59.8	20.1	20.1	35.0	11.7	11.7	11.5	30.1	2733	2.20	0.44	1.15	3.79	2.33	.47	.49	.59	.59	.035	.034	.029	.028	.72	.72	.74	.73
11	1571	59.4	20.3	20.3	32.9	11.2	11.2	10.4	34.3	3269	2.30	0.43	1.41	4.14	1.94	.60	.50	.62	.63	.039	.039	.031	.035	.67	.67	.68	.68
12	386	36 7	42.2	21.1	17.9	20.5	10.2	23.1	28 3	781	2.13	1.00	1.22	4.35	2.54	.43	.41	.44	.55	.031	.037	.021	.021	.70	.68	.71	.71
13	3677	36.1	42.6	21.3	17.8	21.0	10.5	22.0	28.7	7424	2.10	0.92	1.23	4.25	.47	.35	.44	.35	.39	.028	.033	.032	.030	.71	.71	.70	.72

The total number of days represented in the above table is 195; of charges, 14,309; and of pig iron, 30,222 tons. It fairly represents the conditions that

maintained during the production of more than 75,000 tons of basic iron, and that are likely to continue in force in the Birmingham district for some time to come. What may be legitimately inferred from these results? In the first place and principally, that basic pig iron of excellent quality has been and can be made of native materials. Secondly, that the choice of these materials and the proportion in which they are used are of great importance in controlling the nature of the output.

Let us take these points up seriatim and discuss them as briefly as possible. What constitutes good basic iron? So much depends on local requirements, and the purpose for which the iron is to be used, *i.e.*, whether in admixture with other iron, or by itself, that no reply to this question applicable to all cases can be made; as a rule, however, maximum limits can be assigned to silicon, sulphur, and phosphorus, which should not be exceeded under ordinary circumstances. It is not customary to include in the specifications the percentages of manganese, graphitic carbon, or combined carbon, but to limit the demands to silicon, sulphur, and phosphorus. Probably there are but few basic open-hearth steel-makers who would purchase stock carrying more than 1% silicon, or more than 0.05% sulphur, although it might be done in regard to silicon if the iron was intended for mixture with other iron very low in this element. The limit for silicon has to be set somewhere, and at 1% it is certainly low enough, and works no hardship to the producer. If the producer of the basic iron were at the same time the manufacturer of the steel he could not afford to use pig iron of more than 1% silicon, unless he had at the same time abundant opportunity for acquiring good scrap. By stacking the iron carrying an excess of silicon, he could use it in admixture with his very low silicon iron, of which he would probably make a great deal more than of the high silicon. Putting the maximum silicon at 1%, there are a greater or less number of casts, depending largely on the condition of the furnace and the skill of the furnaceman, that will exceed this amount. This iron cannot be shipped under the terms of the contract, but it could very well be used on the spot. Take, for instance, a lot of iron carrying 1.1% silicon, or even 1.2%, it could be used with iron carrying 0.5%, or 0.35%, but cannot be shipped because the maximum silicon is 1% under the contract.

It is greatly to the advantage of the pig-iron producer to keep well within the limits of the specifications, so as to allow for certain unavoidable variations in the working of the furnace. His purpose should be to keep the silicon below 0.75%. This is good practice if he is selling his product on analysis, and especially so if he is making steel himself, as the lower the silicon is kept the better he can use the iron that exceeds the limits he has himself set. For his own purposes he would need metal as good as what he offered for sale, and should insist on having it. Even running the metal direct into ladles that discharge into the steel furnace the silicon can be determined within 20 minutes, and the composition of each cast known almost by the time the ladle is empty.

The silicon is not so apt to cause trouble as the sulphur. Fewer casts show a tendency toward increase of silicon than toward increase of sulphur, and under nearly all circumstances the silicon is the more easily controlled of the two. When the silicon falls the sulphur is apt to increase, but this tendency does not become serious until the silicon is below 0.3%. When this is the case the furnace

is working cold, and steps should immediately be taken to increase the heat and decrease the basicity of the cinder. When the silicon falls below 0.2%, something is radically wrong, for the sulphur rises rapidly and may soon exceed the limit. A careful and experienced furnaceman, by observation of the cinder and the flow of the iron, will generally be able to detect a tendency toward "working cold," but after all the chemist must decide whether or no the limit is reached.

The burden that shows the greatest tendency toward the increase of silicon is No. 4, which carries the largest proportion of hard ore, viz., 81.2% of the ore burden, and no brown ore. The charges under this burden were composed of 9300 lbs. of hard ore, 2100 lbs. of soft, 1000 lbs. of dolomite, and 6500 lbs. of coke. Taking the analyses, as given, we find that in each charge there were in the hard ore 1506.6 lbs. of lime, in the soft ore 23.52 lbs., in the dolomite 303.1 lbs. of lime, and 207.1 lbs. of magnesia, and in the coke 6.50 lbs., or a total of 2046.82 lbs. of total flux, of which 1839.72 lbs., or 89.9%, was lime. Calculating the amount of silica in each charge in the same manner, we find in the hard ore 1249.92 lbs., in the soft ore 361.20 lbs., in the dolomite 15 lbs., and in the coke 293.15 lbs., or a total of 1919.27 lbs. Taking 1 part of magnesia as equal in fluxing power to 1.19 parts of lime, we may say that the 207.1 lbs. of magnesia derived from the dolomite are equivalent to 246.44 lbs. of lime, so that the lime-flux would be 2086.16 lbs. The silica to be fluxed is 1919.27 lbs., so that there is more lime than is required for the ratio SiO_2: $CaO = 1 : 1$. The excess is 166.89 lbs. on each charge. During the period under examination there were 279 charges, with a production of 520 tons of iron, or 1.86 tons per charge. The lime in excess of the above ratio is practically 167 lbs. per charge, so that the 520 tons of iron made were exposed to the desiliconizing action not only of the cinder, as above, but also of 46,593 lbs. of lime not required for the slagging of the silica. Each ton of iron was washed with 89 lbs. of lime. In spite of this, however, the tendency of the iron was decidedly toward an increase of both silicon and sulphur, and the burden was changed. The average silicon on this burden was 0.87%, and the average sulphur 0.047%.

Let us examine the results from a burden of exactly opposite tendencies, the iron being of particularly good quality. We will select No. 12. The charges under this burden were composed of hard ore 3500 lbs., soft ore 4000 lbs., brown ore 2000 lbs., dolomite 4500 lbs., and coke 5500 lbs. The ore burden was composed of 36.7% hard, 42.2% soft, and 21.1% brown, and the total burden of 17.9% hard, 20.5% soft, 10.2% brown, 23.1% dolomite, and 28.3% coke. The number of charges was 386, and the iron made 781 tons, or 2.02 tons per charge. The lime per charge, calculating the magnesia as lime, was 3106.6 lbs., derived as in I. The silica per charge was 1698.35 lbs., derived as in II.:

From—	(I.) Lime. Per Charge. Pounds.	(II.) Silica. Per Charge. Pounds.
Hard ore	567.0	470.40
Soft ore	44.8	688.00
Brown ore	16.8	224.40
Dolomite	1,472.5	67.50
Coke	5.5	248.05
Totals	3,106.6	1,698.35

Subtracting the silica from the lime we have 1408.3 lbs. of lime in excess of that required for slagging the silica under the ratio $SiO_2 : CaO = 1 : 1$ In other words, the 781 tons of iron were subjected to a bath of 543,603 lbs. of lime, *i. e.*, every ton of iron was washed with 694 lbs. of lime, with the result that the highest silicon found was 0.63%, and the lowest 0.27%, the average being 0.46%. The elimination of the sulphur was not so marked, 5% of the casts carrying more than 0.05%, the highest sulphur being 0.063%, the lowest 0.012%, and the average 0.027%. Basic iron carrying a maximum silicon of 0.63% and maximum sulphur 0.063%, with only 5% over 0.05%, the average being 0.027%, is certainly very good iron. There are records covering the production of 11,379 tons, showing a maximum silicon of 0.98%, and an average silicon of 0.48%, not a single cast being up to the limit, and only a very few anywhere near it. Of the 264 casts examined during this particular period, only 15 (5.7%) ran over 0.05% in sulphur, and even then the maximum was 0.065%, the average being 0.031%.

We will take, for the further elucidation of this matter, two more examples, one in which no brown ore was used, and the other in which 10.6% of the ore burden was brown ore.

First, then, No. 1.—Ore burden: Hard, 64%; soft, 36%. Total burden: Hard ore, 33.1%; soft ore, 18.6%; dolomite, 14.5%; coke, 33.8%. Total burden in pounds: Hard ore, 6400 lbs.; soft ore, 3600 lbs.; dolomite, 2800 lbs.; coke, 6500 lbs. Number of charges, 1848. Iron made, 3521 tons. The silica of the burden was 1813.51 lbs., derived as follows: From the hard ore, 859.16 lbs.; from soft ore, 619.20 lbs.; from dolomite, 42 lbs.; from coke, 293.15 lbs.

The lime of the burden, calculating the magnesia of the dolomite as lime, as before, was 2621.97 lbs., derived as follows: From the hard ore, 1036.80 lbs.; from soft ore, 40.32 lbs.; from dolomite, 1538.35 lbs.; from coke, 6.50 lbs.

Using the silica-lime ratio 1 : 1, as before, we find that for every pound of silica in the burden there were 1.44 lbs. of lime. In addition, therefore, to the lime required for slagging the silica, every ton of iron made was washed with 425 lbs. of lime. The iron showed much less tendency toward high silicon and high sulphur than No. 4, in which the hard ore was 81.2% of the ore burden, but a far greater tendency in these directions than No. 12, in which the brown ore formed 21.1% of the ore burden. The iron was a great deal better in respect of silicon than from No. 4, the average silicon being 0.65% as against 0.87%. In sulphur there was not so much difference, the average being 0.04% as against 0.047%; yet in this respect also there was a noticeable improvement.

Finally, let us consider No. 6.—Ore burden: Hard, 36.2%; soft, 53.2%; brown, 10.6%. Total burden: Hard ore, 17.8%; soft, 26.2%; brown, 5.2%; dolomite, 22%; coke, 28.8%. Total burden in pounds: Hard ore, 3400 lbs.; soft, 5000 lbs.; brown, 1000 lbs.; dolomite, 4200 lbs.; coke, 5500 lbs. Number of charges, 1099. Iron made, 2068 tons. The silica of the burden was 1740.21 lbs., derived as follows: From hard ore, 456.96 lbs.; from soft ore, 860 lbs.; from brown ore, 112.20 lbs.; from dolomite, 63 lbs.; from coke, 248.05 lbs.

The lime of the burden, calculated as before, was 2928.60 lbs., derived as follows: From hard ore, 550.80 lbs.; from soft ore, 56 lbs.; from brown ore, 8.40 lbs.; from dolomite, 2307.90 lbs.; from coke, 5.50 lbs.

For every pound of silica in the burden there were 1.68 lbs. of lime. In addi-

tion, therefore, to the lime required for slagging the silica, every ton of iron made was washed with 627 lbs. of lime. The iron was of excellent quality, the average silicon being 0.59%, and the average sulphur 0.028%. Not a single cast exceeded the limit in silicon, the highest silicon being 0.84%, and only 7.1% exceeded the limit in sulphur, the highest sulphur being 0.065%.

To sum up these four cases we may proceed as follows: Total iron made, and from which these conclusions are deduced, 6890 tons from 3612 charges. When the excess of lime was 89 lbs. per ton of iron the silicon and sulphur both showed strong tendencies toward the maximum allowed. When the excess of lime was 425 lbs., no brown ore being used, this tendency was markedly diminished. In Nos. 1 and 4 no brown ore was used, the iron made was 4041 tons, the excess of lime per ton of iron in No. 1 being 425 lbs., and in No. 4, 89 lbs. The quality of the iron made in No. 4, with much the smaller excess of lime, viz., 89 lbs., was much inferior to No. 1, which carried an excess of 425 lbs., but in neither case was it as good as that made in No. 6 and No. 12 with brown ore. Furthermore, in considering other cases, which it is unnecessary to quote in detail, we find that when the excess of lime with a brown ore burden is no greater than 425 lbs. per ton of iron, the quality of the iron is better than when this excess is used with burdens containing no brown ore. In other words, brown ore is of decided advantage irrespective of the excess of lime. The smaller excess of lime in hard-soft burdens doubtless yielded worse iron than the larger excess, because the desiliconizing and desulphurizing action within the furnace was not sufficient. There were only 89 lbs. of lime, above what was required for the cinder, to affect the iron, and experience has shown that much more than this is needed. For instance, when the excess of lime was raised to 425 lbs. per ton of iron the quality of the iron was greatly improved, although no brown ore was used.

In the production of basic iron it is not sufficient that the cinder be merely basic; it must be basic enough to act upon the iron, or else the results will not be satisfactory. Iron has been made of less than 0.2% silicon and less than 0.04% sulphur, the excess of lime per ton of iron being close to 700 lbs. We have had to depend upon the lime and magnesia as desulphurizers, as our ores seldom carry more than 0.3% of manganese If manganese ores, or manganiferous iron ores, could be obtained at a reasonable cost the basicity of the burden could be diminished, for our experience has been that the chief trouble is on account of the sulphur. It is far easier to control the silicon than the sulphur, and we should welcome any agency that promised to reduce the tendency of the iron to carry sulphur. But we set out to make basic iron from materials we were using every day, and that could be obtained at an extremely low cost, and ores carrying manganese did not enter into the calculations. Whether or no it would be advisable to pay the extra price for ores carrying notable amounts of manganese is an open question, and would have to be discussed from the standpoint of current quotations of basic iron. If the market was favorable and good contracts could be made covering a considerable tonnage, the use of such ores might become advisable. This would be particularly the case if the specifications were unusually stringent, or competition more than ordinarily severe, provided, of course, that these ores were low in silica, sulphur, and phosphorus. Some of the manganese

ores available are not distinguished by their freedom from these substances, to say the least of them.

It will be noticed that the flux has been calculated as lime, the magnesia of the dolomite being stated in terms of this base. It is more convenient to adopt this method for purposes of calculation, and when the fluxing ratio between the lime and the magnesia is given no confusion of terms can take place. The ratio adopted is, as already stated, 1 magnesia = 1.19 lime. But irrespective of this ratio it has been found here that the exclusive use of limestone in the production of basic iron is not advisable. The composition of the iron is neither so regular nor so uniform. Limestone was tried, but the results were not satisfactory, and when it was used in conjunction with dolomite the quality of the iron was not so good. For nearly a year dolomite has been employed exclusively, the average silica in 5000 cars being 1.78%. The available supply of this stone is very large, and no fears need be entertained that it will not be sufficient for all ordinary demands for a number of years.

It will also be noticed that no account has been taken of the alumina present in the ores, flux, and fuel. It cannot be stated positively what part alumina plays at such high temperatures and in the presence of large excess of lime and magnesia. At some temperatures it appears to require silica for its removal, and the custom in ordinary practice is to add silicious soft ores to aluminous soft ores in the burden. Under such circumstances alumina is a base, and requires an acid flux, but it by no means follows that it is always a base, and that silica will at all times remove it. The exact rôle of alumina in the blast furnace is a disputed question, and the truth seems to be that it may be base or acid, according to circumstances. Just what those circumstances are, how they are brought about, and how controlled, is beyond the province of this paper. If it be taken as acid the foregoing calculations as to the excess of base in the burdens would have to be modified considerably, the extent of such modification depending upon the saturation point of the alumina with respect to lime, to magnesia, and to lime and magnesia.

If we take the ratio of silica to alumina, under the usual cinder made, as 1 : 1 (as a matter of fact it seems to be 1 : 0.87), the excess of base (lime and magnesia calculated as lime) in No. 4, of 89 lbs. per ton of iron, would entirely disappear, and in place of it we would have 336 lbs. of silica and alumina per ton of iron. The burden would then be acid, and it may be that the unsatisfactory results from this burden are to be attributed to this fact. Taking the other hard-soft burden, No. 1, instead of having an excess of base of 425 lbs. per ton of iron, this amount dwindles to 67 lbs. Taking the two brown ore burdens examined, in the same way, we find that No. 6, instead of having 627 lbs. excess of base per ton of iron, has 336 lbs., and No. 12 instead of 694 lbs. has 425 lbs.

These changes in the preceding calculations do not alter the fact that a very basic burden is required for uniformly good basic iron. In truth they intensify this conclusion, for if the silica and alumina be regarded as the maximum acidity of the burden, it is found, as before, that as we approach more and more a neutral burden the quality of the iron begins to deteriorate, and is at its worst when the line is passed and the burden becomes acid. On the other hand, as we approach the maximum basicity, consistent with fluidity of the cinder and regular

working of the furnace, the quality of the iron improves and is at its best when the excess of base per ton of iron is between 400 lbs. and 600 lbs. This point is the most important that can arise in basic-iron practice, for it determines the efficiency of the desiliconizing and desulphurizing action that is sought to be accomplished. The iron must be subjected to a basic bath, and the blast furnace is even better than a bath, for every particle of iron must trickle through the basic cinder and be exposed on all sides to its powerful action. No exterior desiliconizing or desulphurizing process can be more effectual than the processes within the furnace itself, provided that they are carried on properly. It is not enough that the saturation point of the acid elements of the burden be reached; this indeed is necessary, but unless there is at the same time an excess of material which can remove the silicon and the sulphur from the iron, or prevent their union with the iron, there cannot be a proper desiliconizing and desulphurizing action. The saturation of the acids and the excess of base must go hand in hand, and the calculation of what excess of base should be carried should depend on the output of the furnace. Whether the excess of base prevents the iron from absorbing silicon and sulphur, or whether, after they are once absorbed, the excess of base removes them from the iron is of no special moment. Perhaps both these actions go on within the furnace, a deterrent action and a removing action, the one in the cooler, the other in the hotter zones. If a furnace on basic iron is working cold we might expect to find a decrease of silicon, and an increase of sulphur, and the main point in basic practice is to keep the furnace hot enough to lower the sulphur and cold enough to lower the silicon. This is largely a question of burdening and blowing, and it is here that the coke plays so important a part.

The consumption of coke is always the most interesting question in connection with the raw materials of iron making. It is by far the most costly material, and on this account economies in its use soon become evident. With coke at $1.75 per ton, each 100 lbs. saved represents a saving of 8.75c. If a furnace has been working on 2500 lbs. of coke per ton of iron, and can diminish this by 100 lbs., the saving on 150 tons of iron per day is $13.12, which would pay the wages of the superintendent, the master mechanic, and the chemist, or pay 6% interest on the cost of building 250 coke ovens.

Let us look into the consumption of coke as it relates to the production of basic iron. It has varied from 1.09 tons to 1.53 tons per ton of iron, *i.e.*, from 2441.6 lbs. to 3427 lbs. for 2240 lbs. of iron, the average being 1.33 tons, or 2979 lbs. Taking the cost of the coke as $1.96 per ton of 2240 lbs., the variation in the cost per ton of iron due to variations in coke consumption is from $2.13 to $2.99, or 86c., the average cost being $2.60. The lowest coke consumption was on the following burden:

	Ore Burden. Per Cent.		Total Burden. Per Cent.
Hard ore	60.2	Hard ore	35.1
Soft ore	19.9	Soft ore	11.7
Brown ore	19.9	Brown ore	11.7
		Dolomite	11.9
		Coke	29.6

There were 2033 charges, and 4553 tons of iron were made. There were examined 126 casts, of which 94 showed silicon below 1%, and 89 sulphur below

0.05%. The average composition of the iron below these limits was: Silicon, 0.47%; sulphur, 0.03%; phosphorus, 0.73%.

The analysis of the stock was as follows:

	Hard Ore. Per Cent.	Soft Ore. Per Cent.
Metallic iron	36.71	46.54
Silica	12.59	19.19
Alumina	3.15	3.00
Phosphorus	0.35	0.30
Lime	17.14	1.00
Sulphur	0.07	0.06

	Brown Ore. Per Cent.
Metallic iron	48.81
Silica	11.00
Alumina	3.25
Lime	0.60
Phosphorus	0.40
Sulphur	0.09

	Dolomite. Per Cent.
Silica	1.95
Lime	29.10
Magnesia	19.20
Oxide of iron	0.40
Alumina	0.60

	Coke. Per Cent.
Ash	11.32
Sulphur	0.91

	Ash of Coke. Per Cent.
Silica	45.10
Oxide of iron	12.32
Alumina	31.60
Lime	1.50
Magnesia	Trace.
Sulphur	0.16

The consumption of materials per ton of iron was I.; and the cost of materials per ton of iron was II.:

	(I.) Tons.	(II.) Cost.
Ore	2.19	$1.765
Dolomite	0.45	0.336
Coke	1.09	2.429
Totals	3.73	$4.530

The burden was not the same throughout this period, but the average charge was as follows, in pounds: Hard ore, 6532; soft ore, 2276; brown ore, 2276; dolomite, 2234; coke, 5500—total, 18,818 lbs.

Taking the analyses as quoted, and considering the silica and alumina to equal the acid elements, we would have in this burden 2340.14 lbs. acid elements, derived as follows: From the hard ore, 1028.13 lbs.; from soft ore, 505.04 lbs.; from brown ore, 324.33 lbs.; from dolomite, 56.96 lbs.; from coke, 425.68 lbs.

Similarly we would have 2325.67 lbs. of basic elements, derived as follows: From hard ore, 1119.58 lbs.; from soft ore, 22.76 lbs.; from brown ore, 13.65 lbs.; from dolomite, 1160.34 lbs.; from coke, 9.34 lbs.—total, 2325.67 lbs.

By the method of calculation adopted this would be practically a neutral burden, although it must be said that there is an excess of 14.47 lbs. of acid elements per charge. But as the average charge was 18,818 lbs., this excess may be disregarded. We cannot but think that the good quality of the iron made was due to the brown ore used, for when this was omitted and the ore burden was composed entirely of hard and soft red ore, the basic elements being slightly in excess, the quality of the iron was by no means so good.

But what we desire to emphasize is that with this burden the consumption of coke was 1.09 lbs. per pound of iron, or 2441.6 lbs. per ton of iron of 2240 lbs.

Now let us examine a case in which the consumption of coke was 1.53 lbs. per pound of iron, or 3427 lbs. per ton. The burdens were as follows:

	Ore Burden. Per Cent.
Hard ore	61.2
Soft ore	34.3
Brown ore	4.5

	Total Burden. Per Cent.
Hard ore	31.5
Soft ore	17.6
Brown ore	2.4
Dolomite	15.1
Coke	33.4

The burden was not the same during this period, but the average can be stated as follows, in pounds· Hard ore, 6245; soft ore, 3314; brown ore, 512; dolomite, 2900; coke, 6500—total, 19,471 lbs.

The number of charges was 2381, and 4500 tons of iron were made. The consumption of materials per ton of iron was: Ore, 2.37 tons; dolomite, 0.70 coke, 1.53—total, 4.60 tons per ton of iron.

The cost of materials per ton of iron was: Ore, $1.643; dolomite, $0.219; coke, $2.864—making a total of $4.726.

There was very little difference in the analysis of the materials during this period and the one just considered—so little, in fact, that they may be taken as the same in quality.

Pursuing the same plan as before in calculating the acid and basic elements of the burden, we have 2333.79 lbs. acid elements, derived as in I.; of basic elements we have 2710.36 lbs., derived as in II.:

From—	Acid (I). Pounds.	Basic (II). Pounds.
Hard ore	982.96	1,070.39
Soft ore	735.37	33.14
Brown ore	72.96	3.07
Dolomite	73.95	1,506.26
Coke	498.55	97.50

There was an excess of lime of 376.57 lbs. per charge, which is equivalent to saying that each ton of iron made was washed with 197.12 lbs. of lime. The quality of the iron made was good. There were examined during this period 120 casts, of which 90, or 75%, showed silicon below 1%, and sulphur below 0.05%, the average for these two elements and for phosphorus being as follows: Silicon, 0.64%; sulphur, 0.031%; phosphorus, 0.71%.

So far as concerns the quality of the iron made there is not much to choose between these two results, the one with 1.09 tons of coke per ton of iron, and the other with 1.53 tons. But the lower yield of the best iron obtained from this latter burden, viz., 9.1%, makes it less desirable, even if the difference of cost of 19c. per ton of iron were not considered. As it is, however, the burden with much less brown ore and requiring much more coke is handicapped with a difference of 19c. per ton of iron and a 9.1% less yield of the best iron.

In considering the matter further we are struck by the fact that even with an excess of lime in the burden the yield of the best iron was less by 9.1% than when the burden was slightly acid but carried over 19% of brown ore.

If now we examine the table of Ore Burdens of Hard and Soft Ore it will be seen that with hard-soft burdens only 17.5% of the iron was made with a coke consumption as low as 1.48 tons per ton of iron, the average being 1.52. With

hard-soft-brown burdens only 6% of the iron was made with a coke consumption as high as 1.48 tons per ton of iron, the average being 1.24. As a general rule brown ore burdens will require 0.28 ton (627.2 lbs.) less of coke per ton of iron made than burdens containing no brown ore. There is possible in this item alone a saving of 54.8c. per ton of iron, which, however, is to some extent offset by other considerations.

In the production of ordinary grades of iron the best burden is the burden that will yield at the least cost the highest percentage of the most salable iron. In the production of basic iron the same principle maintains, but in a much more marked degree, for every carload is sold on analysis and every cast that does not come up to the specifications must find some other market. It is a source of great satisfaction to the pioneers in this movement to realize that in spite of the numerous obstacles, such as irregularity of stock with its consequent frequent changes of burden, unavoidable experimentation, etc., fully 80% of the iron made was of excellent quality. The average composition of the casts on which this assertion is based was as follows: Silicon, 0.51%; sulphur, 0.032%; phosphorus, 0.73%; of the other 20% fully 15% could be used in admixture with the above in judicious amounts, leaving but 5% unsuitable for the basic open hearth. In other words, 95% of the iron made is suitable for use as basic iron, and could be so employed without serious impairment of the excellence of the stock.

It would seem that these results abundantly warrant the assertion that the production of pig iron suitable for conversion into steel by the basic open-hearth process has passed far beyond the experimental stage, and may now be regarded as established on a commercial basis of unquestionable stability. Whether or no advantage will be taken of this fact, and steel works built here with the purpose of using this iron, remains to be seen.

The following propositions appear to have been established by experience:

1. Basic open-hearth steel has been made here of pig iron produced from native materials. It was of excellent quality. It was made in 1888, in 1892–93, and in 1896. The total amount is about 4000 tons.

2. Pig iron in every way suitable for conversion into steel by the basic open-hearth process has been made here in large amounts. During the 12 months ending September 30, 1896, the amount of this iron exceeded 75,000 tons, and large shipments were made to many of the heaviest producers of this class of steel in the country.

3. The extent of the deposits of ore, coal, and dolomite from which this iron was made is very great, and there need be no fear of exhaustion within the next generation.

4. The cost of materials per ton of iron is less than at any other point in the United States, or elsewhere, and will probably continue so for many years.

5. The climate is mild, labor is abundant and efficient, and transportation facilities are all that could be desired.

Iron Ore in Spain.

By Reginald W. Barrington.

The improvement in the iron-ore business which became manifest toward the close of 1895 continued steadily during 1896. Looking back I think it has been a good time for almost everybody engaged in the industry, and after so many years of depression the change has been a most welcome one. The period has been one of steady increase in demand and prices, and at the present time the call for good iron ore all over Spain is greater than the production. So many mines were shut down during the long-continued depression that it was only toward the close of 1896 that work could be resumed in many of the old properties.

I estimate the total export of iron ore from Spain in 1896 at 6,619,379 tons, against 5,179,760 tons during 1895; and this great increase notwithstanding that, owing to great scarcity of tonnage at the close of the year, large quantities of ore remained in stock awaiting shipment; and further, practically no ore from the numerous new deposits which were developed during the year in the south of Spain, especially in the Province of Almeria, entered into the above figures. These new mines will doubtless contribute very largely to the shipments from the Peninsula during the present year.

Almeria.—Until quite recently the only ore produced in any quantity in this province came from the district of Bedar. It was shipped from Garrucha, being commonly known in the market as Garrucha ore. The export of this ore in 1895 amounted to 98,688 tons. The Bilbao firm, Messrs. Chavarri, Lecoq & Co., has now acquired the important group of mines situated at Cañadicas and Serrena, from which sierra to Garrucha it has constructed a railway, opened during the latter part of 1896, a distance of about 20 kilometers, and the mines are expected to give an output of about 1000 tons per day of hematite ore of excellent quality.

A little farther to the westward is a fine group of mines in the Sierra La Redonda in the Atalaya range in the district of Lubrin. The ore is of exceptionally pure quality, containing over 50% iron and 4 to 5% manganese. The deposits are 820 meters above the sea-level, and it is proposed to transport the ore by means of a wire cable to Garrucha for shipment, a distance of 26 kilometers.

Another important group of mines are those in the Sierra de Almagro, which it is proposed to connect by an aërial cable with Villaricos.

Farther down the coast we come to the Lucianena mines, which are said to be the finest group in the new district. These mines have just been splendidly equipped, and are being exploited by Messrs. La Sota of Bilbao, who last year completed their railway from the mines to Aguas Almalgas, where they have also erected a costly pier for the economical and easy shipment of the ore.

At Carboneras, just to the east of Cape de Gata, several new iron mines are also being worked.

For shipment at the fine new port of Almeria many groups of iron mines, situated within easy distance of the new railway from Linares to Almeria, have been taken on lease and purchased by both Spanish and English companies.

Many of them are without doubt very valuable properties, but among them there are some wildcat concerns which the foreign investor will do well to be shy of. The only ore shipped in any quantity in 1896 was from the mines worked by the Compañia de Portman (which for many years exploited the Humo and Crisoleja mines in the vicinity of Cartagena and Portman). The important deposit of iron ore it is now working is situated at Pilatos in the district of Hueneja. From the mines to the railway station of Hueneja is a distance of about 10 kilometers, over which the ore is economically conveyed by a wire-rope railway. It is then taken by the Linares-Almeria railway to Almeria, a distance of about 70 kilom.

The ore at this mine may be said to exist in the form of an enormous pocket of ore of so clean a quality that practically no picking is required, and the cost of mining is therefore covered by a few pence per ton. The mechanical condition of the mineral is unfortunately small and its nature somewhat specular. On examination it appears to be a hard ore shattered into small particles. The poorest ore that the mine produces is said to yield over 60% of metallic iron and to be exceptionally free from deleterious matter.

These mines are the most distant from Almeria of any now being worked. The next in the territory tributary to the Linares-Almeria railway are the deposits in the district of Fiñana, one to the west of Fiñana in the Sierra Nevada's most easterly extremity, and the other to the northeast in the Sierra de Baza, the nature of the ore being of entirely different character, as shown in the analysis I give further on. A cañon, or "rambla," runs up from Fiñana into the Baza sierra; and at a distance of about five kilometers, and on the slopes of the sierra, the iron-ore outcrops begin to make their appearance, and for a distance of 10 kilometers I traced the iron lode, fine bluffs cropping up at short distances.

So far no iron-mining has been done in this locality, and it is therefore impossible to estimate to what depth the ore may penetrate. One thing is evident to the naked eye, however, viz., any one handling these ores will have to contend with the presence of a certain quantity of copper, which may be a serious drawback to their value for smelting purposes. The following is an analysis of a general sample taken by me from the surface: Peroxide of iron, 73.243% (metallic iron, 57.00%); peroxide of manganese, 1.030%; silica, 1.510%; alumina, 0.510%; lime, 2.150%; magnesia, 1.900%; sulphuric acid, 1.213% (sulphur, 0.485%); phosphoric acid, 0.005% (phosphorus, 0.002%); copper, 0.530%; loss on calcination, 7.630%; moisture, 10.050%—total, 99 771%.

Owing to the topographical situation of the mines in this district, the only feasible means of transporting the ore will be by wire cables to Fiñana station.

The iron-ore deposits in the westerly extremities of the Sierra Nevada are situated within 6 kilometers of Fiñana, and here the geological formation is of a much more settled character than in the Sierra de Baza, the formation being limestone; and already sufficient preliminary work has been done to be able to estimate that the ore body runs to a considerable depth, while its length and width cover many kilometers. A general sample taken by me from the surface outcrops gave the following result: Peroxide of iron, 81.286% (metallic iron, 58.030%); silica, 3.400%; alumina, 1.850%; lime, 0.600%; magnesia, trace; sulphuric acid, 0.130% (sulphur, 0.052%); phosphoric acid, 0.126% (phosphorus,

0.055%); loss on calcination, etc., 10.300%; copper, 0.150%; arsenic, 0.075%; moisture, 1.950%—total, 99.867%. Nearly all these deposits have a good road to Fiñana, and operations have already been started at several of them.

Gergal.—This is the next important iron-mining district along the Linares-Almeria railway, and is situated nearer to Almeria than either of the two already mentioned. As in Fiñana, here also the ore deposits occur in two distinct ranges, in each of which the composition of the ore and the geological formation are entirely distinct. Sierra de Aulargo is slightly to the south of the Sierra de Filabres in the Penebetica range, and the iron outcrops begin to make their appearance within 2 kilometers of the station of Gergal and Alcubillas, and continue in a northwesterly direction for many miles, almost without a break. In the Cerro del Layon, a small and almost isolated mountain which appears to be practically one mass of iron ore, and is situated also very close to the railroad, several mines are now working, and their ore is being mixed with that from the Sierra de Aulargo. Gergal is distant only a little over 40 kilometers from Almeria, and the ore, which is of rich quality, can be put on board ship at a moderate cost. The mines have the advantage of being within a short distance of Gergal station, with which they are connected by a good cart-road. Already many mines are being actively worked.

The following analyses are made from general samples taken from the lode at the principal mines in the Sierra Aulargo and Cerro del Layon:

Mines.	FeO_3	MnO_2	SiO_2	Al_2O_3	CaO	MgO	SO_3	P_2O_5	Cu
Sierra Aulargo	78.336	3.586	1.714	0.350	0.900	0.240	0.180	0.016	0.180
Cerro del Layon.....	88.000	Trace.	1.150	0.680	0.500	0.300	0.375	0.070	Trace.

Mines.	As	Loss in Ignition.	H_2O	Total.	Fe	Mn	S	P
Sierra Aulargo	.. ..	4.186	10 400	100.088	61.200	2.270	0.077	0.007
Cerro del Layon.....	0.077	7.000	1.650	99.802	62.630	.....	0.150	0.030

Cerro de Enmedio; Cerro de Abar; Barraco del Manco.—The mines in these ranges are situated from 3 to 8 kilometers northeast of the town of Gergal, and from 5 to 12 kilometers from Gergal Alcubillas station. Hardly any of them have wagon communication with the "pueblo" or station of Gergal, although there is talk of connecting some of the mines with the main line by a narrow-gauge railway. Many of the mining concessions here have been secured from the original owners by foreign syndicates, and those remaining in the hands of local people are being offered in the market at exorbitant royalties, which, added to the expense of costly plant for transport, makes the idea of working them at a profit under existing conditions very difficult. There is no doubt that in some places the deposits of ore are enormous, and considerable preliminary work is being done to investigate them. The owner of the mine "Mi Emliano," which may be said to be the pioneer of the district, has already expended a considerable sum in thoroughly exploring the ore body, with the most satisfactory results. The following analysis is of a general sample taken from some thousand tons of ore stacked at the different workings of the mine; the other analyses

are from samples taken from other mines in the district where, however, no exploratory work had yet been done:

Mines.	FeO_3	MnO_2	SiO_2	Al_2O_3	CaO	MgO	SO_3	P_2O_5	Cu	Loss in Ignition.	H_2O	Total	Fe	Mn
Mi Emiliano	81.021	1.860	1.950	0.620	0.650	0.440	0.175	0.035	0.093	5.400	7.480	99.724	61.30	1.18
Others..... ...	91.63	2.00	0.68	0.28	0.31	0.18	0.03	0.05	Trace	4.20	0.80	100.16	64.14	1.30
	86.31	1.42	1.74	0.64	0.94	0.01	0.14	0.01	0.01	2.38	6.45	100.05	62.50	0.90

Magnetic-iron Mines.—I will now briefly describe the two principal deposits now working in the south of Spain which produce this class of ore.

Marbella Mines.—These are situated in the Province of Malaga, and have for many years been producing ore of a quality now well known in the American and European markets. These mines were first started in the early seventies and their annual output ranged from 50,000 to 100,000 tons to within the last few years, during which there has been a considerable falling off. The ore from these mines contains over 60% of metallic iron, but the absence of combined water makes it rather hard for smelting, and unless mixed with other ore it cannot be employed in furnaces where coal and not coke is used. Owing, however, to the ore being exceptionally pure as regards phosphorus, a very good class of low-phosphorus pig iron can be produced from it. For many years the greater part of the output from these mines has been shipped to the United States.

Adjoining the mines belonging to the Marbella Iron Ore Company, Limited, there is a small river, in the bed of which is found a large quantity of magnetic-iron sand, which when washed and selected yields over 60% metallic iron. The mechanical condition of this sand is of course a drawback to its use on a large scale unless mixed with lump ore, the percentage of lump and rubble in it being under 5. There is, however, one special feature in favor of this sand in that, besides being low in phosphorus and sulphur, it is free from titanium. Magnetic-iron sand found in other districts generally contains such a high percentage of titanium as to make it objectionable in the furnace.

Cehegin Mines.—Less than a year ago I referred to these mines in a paper upon the "Iron-ore Resources of Spain" as one of the many good deposits situated in the interior which, owing to their isolated position and the difficulties in transporting the ore to the seaboard, were almost without present value. It is therefore with the greatest pleasure that I am now able to relate that, notwithstanding the many drawbacks, an Englishman has been sufficiently enterprising to overcome them, and has within a few months organized active mining operations at the Carmela mine with satisfactory results. The output of this mine is now over 4000 tons per month, averaging 61% Fe. The mines are situated at Cehegin, in the district of Calasparra in the Province of Murcia. The ore is magnetic, mixed here and there with red hematite. The mines cover an area of over 4,000,000 sq. meters, and are distant from Cartagena, which is the nearest shipping port, a little more than a hundred miles, ninety of which is covered by railroad and the remainder by cart-road. The mass of mineral is undoubtedly enormous, the quantity of available ore having been estimated by different mining engineers as between 2,500,000 and 4,000,000 tons. The following is a general analysis of ore from the surface, in connection with which it should be said that

as developments progress the quality appears to be improving: Carbonic acid, 2.80%; silica, 2.48%; ferric oxide, 90.40%; metallic iron, 63.28%; manganese oxide, 0.42%; alumina, 1.69%; lime, 0.78%; magnesia, 0.32%; sulphuric acid, 0.22%; phosphoric acid, 0.07%; loss on calcination, 0.82%—total, 100.00.

A ready market has already been found for this mineral, and when the mines are all equipped for working the production will be over 10,000 tons monthly. There is no doubt that, owing to the first-class results obtained from laboratory investigations, the demand for the ore from the above district will probably exceed the supply.

Cartagena.—In Vol. II. of THE MINERAL INDUSTRY will be found a description of the various ores mined in the sierra of Cartagena and Portman, since which there has been no important change in the character of the ore produced, the chief feature being the scarcity of No. 1 or 20% manganiferous ore, which a year ago was sold at 12s. and is to-day almost unobtainable at 15s. 6d. The sierra is studded with innumerable small mining concessions, running into one another, and in many cases these mines are again divided and leased in smaller portions. The number of mines officially registered as existing and paying government royalty in the Province of Murcia in 1894 was no less than 231, fully 75% of which belonged to the Cartagena district. These mines, however, only covered an area of 33,901 hectares. Thus, while the Province of Murcia represented 16.67% of the entire number of mining concessions throughout Spain, it only represented 6.23% of the mining area.

At the present time there are no large companies working iron mines in the Cartagena district, the majority of them being operated in a penurious and hand-to-mouth manner. The miners are hard-working and good, practical men, but, owing to their primitive appliances, and no means to provide themselves with better, many mines, which in the hands of their original owners would be developed into splendid properties, drag along a miserable existence under the control of the lessee, who potters about and cuts ten tons where a hundred might be produced under different management, and utilizes only the best and most easily obtained ore, throwing everything else on the tailings-heap. The real cause for this unsatisfactory mode of working is that the mines are seldom, if ever, worked by their original owners, who lease them to private individuals at a royalty of from 12 to 25% of the value of the ore cut and cleaned at the mouth of the mine; and in most cases these people again subdivide the mine into smaller lots, which they let out to the working miner at a still higher royalty.

Nearly all the ore is brought down from the mines to the railway depositories at the foot of the sierra on donkeys, for which work many thousands of animals are employed. A donkey-load is about 125 kgms.; and as the donkeys make from four to twelve journeys from the mines to the depositories, a donkey may be estimated to carry about one ton per day. From the depositories the ore is carried to Cartagena for shipment, a distance of 12 kilometers, by a steam tramway capable of carrying about 3000 tons per day. An extension of this line has just been made into the southeastern part of the sierra, which will be the means of opening up many mines which up to now have been lying dormant, and thus give fresh impulse to the export of iron and manganiferous ore from the port of Cartagena.

Other iron ore mines in South-Spanish districts from which ore was shipped in 1896 were: Pedroso, shipping from Sevilla, which were started in 1894 and produced in 1895 about 125,000 tons of ore and a considerably larger quantity during 1896; Morata, shipping from Parazuelos; Sierra de Enmedio, from Aguilas; and Sierra de Alhamilla, making shipments just outside of the port of Almeria. Iron ore is also being prepared for shipment from Mazarron and from Palomares.

The Bertrand-Thiel Open-Hearth Steel Process.*

By E. Bertrand.

In the autumn of 1896 some work was undertaken at Kladno, Bohemia, to answer these questions: 1. May the process be worked to advantage with nearly 100% of silicious and phosphoric pig iron? 2. What proportion of highly phosphoric (basic) pig iron may be worked with good results?

A series of experimental heats was made by charging gray forge pig containing about 3.8% Ca, 1% Mn, 1.6% P, and 1% SiO_2 into the primary furnace, adding, besides the lime, large quantities of magnetic Gellivara ore, with about 65% of iron. The finishing furnace was here actually used only for finishing the heat. Twelve tons of forge pig iron were charged into the primary furnace. Since the heats boiled up excessively, repeatedly causing large quantities of slag to flow out of the furnace doors, a small quantity of pig iron was replaced by scrap in the subsequent heats, which helped to quiet down the metal. The sample ingot from the primary furnace was taken when the metal was tapped and flowed down into the finishing furnace. The work done by the primary furnace was follows:

	C.	P.	Si.	Mn.	Total.
Pig iron	3.8%	1.6%	1.00%	1.00%	7.4%
Ingot from primary furnace	2.2	0.4	0.05	0.05	2.7

The oxidation of the phosphorus in the primary furnace before the carbon is highly important, since it greatly facilities the work in the finishing furnace, inasmuch as only so much lime need be added in the finishing furnace as is necessary to perfectly neutralize the phosphoric anhydride resulting from the rest of the phosphorus, and it is therefore very easy to make the finishing slag highly basic, and thus insure a low percentage of phosphorus in the finished metal. In fact, the whole work in the primary furnace may be regulated by the amount of phosphorus to be left in the metal when tapping it into* the finishing furnace.

As regards the temperature of the heat, this to a great extent depends upon the silicon in the pig iron: whose oxidation helps to give the heat the proper temperature. A certain percentage of silicon is therefore favorable for the process, since its oxidation helps to heat the bath to the necessary temperature, and its further acting as a reducing agent upon the ore added helps to increase the yield of metallic iron. At the commencement a heat lasted about five hours in the primary furnace, which was decreased to 4½ hours or 4¾ hours as the men got accustomed to the work; in the same way the time in the finishing furnace was gradually decreased to about 2 hours or 2¼ hours.

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*From a paper read before the Iron and Steel Institute.

Recent Economies in Iron and Steel Manufacture.

By H. G. Graves.

In recent years there have not been such great economies in the manufacture of iron and steel as were made when the hot blast was applied or when the Bessemer process was invented. In the first case, for instance, the cost of producing a ton of pig iron was £4 12s. in 1828, but when the hot blast was generally adopted in 1832, the cost dropped to £2 12s., while the coal consumption fell from 8.1 tons to 2.6 tons per ton of pig. An even more marvelous economy was made when the invention of Sir Henry Bessemer reduced the price of steel from £60 to £20 per ton and the consumption of coal to zero. It does not seem possible, although it is never safe to prophesy, that such marked changes will ever again be made in the iron and steel industry, and yet recent years have produced improvements and economies which, compared with the instances quoted above, are only minor changes but nevertheless are of great importance.

Thus there are two new processes, one for the manufacture of wrought iron and the other for open-hearth steel, which have more than passed the experimental stage and have shown a very material economy.

The Bonehill Direct Puddling Process.—The direct puddling process as carried on by M. E. Bonehill, at Hourpes, in Belgium, is not entirely new except in that it has now been successfully worked for some three years with satisfaction to all concerned. The wrought-iron industry is indeed in a decadent state, as the aspect of so many works proves, but the steel makers in spite of their sanguine expectations are not likely to entirely oust the wrought metal for many years to come, so that a process which reduces the waste of iron from 16 to 6% and the consumption of fuel from 100 to 22% is very worthy of attention. At Hourpes the metal from the blast furnace is tapped into a reservoir or mixer capable of holding 35 tons, or about one-third of the day's output of one furnace. Here it is kept liquid by burning gas from a small producer, using a ton of coal in 12 hours, until it is drawn off into ladles on trucks, which convey it to the puddling furnaces. Of these there are four, all gas-fired and provided with regenerators which are used to preheat the air, while the gas is introduced direct from producers. Each puddling furnace has two working-doors on each side. The molten metal is poured in through one of these on a layer of scrap, which is thrown in to protect the bottom. The charge is worked through in less than an hour, the atmosphere being oxidizing for about half that time, until the iron comes to nature, when it is balled and taken away to the shingling hammers and rolls. Eight puddlers at one furnace can make about 12 tons of puddled iron in 12 hours, or 3300 lbs. per man as compared with 2050 lbs. per man working in the ordinary way on 600-lb. charges of pig iron charged cold and melted down in a single coal-fired furnace. The consumption of coal for an outturn of 96 tons daily amounts to 2 tons for heating the reservoir and 19.2 tons for firing the furnaces, or a total of 21.2 tons. The ordinary consumption of coal is at least equal to the weight of puddled iron produced. In South Staffordshire, for instance, it reaches 26 cwt. per ton of iron. The loss of metal in the Bonehill process is reduced from 16 to 6%

as compared with that in other Belgian works, where the practice is followed of using iron comparatively low in metalloids and of making small additions only. Two analyses of the iron for puddling are as follows: No. I.—C, 2.95%; Si, 0.43%; P, 1.40%; S, 0.50%. No. II.—C, 3.30%; Si, 0.65%; P, 1.15%; S, 0.32%.

In the South Staffordshire district of England it is the practice to use a much higher content of silicon, generally over 1% and sometimes up to 2%, and the melting-down period aids in its removal. Absolute waste of the iron charged is to a large extent prevented by the reduction of metal from the fettling, but in the Bonehill direct process practically none of these expensive additions are used. Only a little purple ore is thrown in before the charge to renew the bottom, and the cost of this may be approximately given as one cent for 100 lbs. of iron, which is more than covered by the selling price of the cinder. A total saving of $2.87 per ton of iron is stated to be obtained, divided as follows: Labor, $0.47; fuel, $1.52, and iron, $0.88. The Belgian rate of wages for puddlers is a trifle under a dollar per day and the value of the coal and iron is taken as $2 and $9 respectively.

The Thiel Process.—The second of the processes referred to is that devised by Messrs. Bertrand and Otto Thiel and now at work at Kladno in Bohemia. Practically it consists in doing in two furnaces what had previously been done in one, when converting pig iron into steel. A warm advocate of this process is to be found in the United States in Mr. J. Hartshorne, while in England Mr. Percy C. Gilchrist acts a similar part. The inventors claim that they can greatly shorten the time of the operation, so as to get a larger output from a given number of furnaces; that they can get a larger output from the same amount of raw material; and that they can obtain a product of desired composition with greater certainty and regularity, even when using an inferior quality of raw material. The results already obtained appear to bear out these contentions. Two furnaces are used, one rated at 12 tons and the other, standing in front of and slightly lower than the former, at 20 tons capacity. Both are lined with basic material but there is no reason against the use of an acid lining. The process consists essentially in dividing the charge between the two furnaces, tapping the metal from the upper into the lower one, and removing the slag from the metal during its progress from one furnace to another. The pig with some scrap, ore, and limestone is charged into the upper furnace, melted and raised to a high temperature. The remainder of the scrap is charged into the lower furnace two hours afterward. An hour later, the metal in the small furnace is hot and fluid; all the silicon and most of the phosphorus and carbon have been eliminated and the bath is ready for tapping. As the metal flows into the lower furnace the voluminous slag is carefully removed, and when the two metals mix a lively reaction ensues and lasts for about ten minutes. The phosphorus in the bath after this period has become reduced to about 0.03% or it may be brought down to 0.02% by leaving it for another quarter of an hour. Ferromanganese or spiegel is added and the furnace is then tapped. An example of the charges for the furnaces is as follows:

Material.	Upper Furnace.	Lower Furnace.
	Tons.	Tons.
Basic Bessemer pig iron	7.50	1.0
Steel scrap	5.50	8.0
Limestone	0.15	0.8
Magnetic iron ore	0.20	*Nil.*

Five or six heats of 22 tons of metal charged are now worked through in 24 hours, but this could be doubled by using another primary furnace. Mr. Gilchrist goes even further and proposes a series of three furnaces of which the last would receive the nearly finished metal so as to reduce its phosphorus to 0.01%. The total cost of conversion is reduced by 27½% and the output, partly due to the use of the second furnace, is increased 72.3%. Steel containing as much as 0.8 to 0.9% of carbon down to 0.2% is readily made, but milder and harder steels can also be manufactured. Another advantage is that 60 to 70% or more of the sulphur is eliminated.

The Walrand Process.—Another process of the greatest importance to small users and to the makers of small steel castings is the Walrand process. As it dates back to a period antecedent to those mentioned above it is perhaps better known. It consists in the addition of silicon iron before an after-blow so that the metal may be brought to a high temperature and thereby obviate difficulties due to radiation and consequent cooling. There are now about a dozen of these small converters at work and the results are so successful that the process of adding silicon iron in large converters has in one or two instances been adopted.

These three processes are somewhat briefly sketched as an example of what has been done recently in a large way, but though they are the most important innovations on a grand scale, it must not be considered that they are the only ones that have come into prominence during the last year or two.

By-products in Puddling.—Turning now to somewhat smaller matters, mention may first be made of a method for saving a by-product of interest to the puddling process. As worked in Great Britain, the latter demands large additions of fettling, and one source of supply is found in the waste from pickling works. Turner's process of treating this by-product has been in successful use for five years at Walsall. The spent acid liquor is concentrated in a shallow pan over a coke fire. The ferric chloride thus obtained is raked out to the space in front of the fire, where it is decomposed into chlorine and ferric oxide. The chlorine passing over the bridge combines with steam and is condensed as hydrochloric acid in a tower, while the oxide is available for fettling. Two furnaces at Messrs. Walker Bros. work up 1000 gals. of waste daily, yielding 300 to 350 gals. of hydrochloric acid and about one cwt. of ferric oxide with a consumption of four tons of coke per week.

Utilization of Blast-furnace Slag.—The recovery and utilization of the waste products of the iron and steel industry are subjects in which there are still large advances to be made although very much has already been done. The problem of the disposal of blast-furnace slag has had more attention than any other, but the recovery of by-products from the gases of coal-fired furnaces

has also attained much success. Then, too, there are products of lesser importance, such as cyanides obtained by tapping the furnaces low down, and recently iodides are stated to have been recovered from flue dust. The use of flue dust as a source of zinc is well known, but no one prefers to use a zinciferous iron ore, and in consequence at the zinc mines of the Real Asturiana Company, in the north of Spain, there are many thousand tons of magnetic concentrates, somewhat finely divided it is true, awaiting a purchaser. Another product of the blast furnace that may be of value is the water used in granulating slag, since, if a recent note in a German paper is to be trusted, it has great medicinal value for the treatment of rheumatism.

As a road metal, slag finds an extensive use on country highways, but for heavy wear a harder and more durable stone, such as that from Leicestershire or the Channel Islands is to be preferred. Paving blocks and slabs have been cast in millions for some years in the Cleveland district and have been used with good results. Unless they are carefully annealed, however, they are liable to disintegration, and their surface is such that they will not hold the mortar for building purposes. The plan adopted is to run the molten slag from the furnaces in a bogie truck to a rotary casting-table which automatically drops the block as soon as it has a sufficiently solid exterior. They are then transferred to an oven where they are annealed by allowing them to cool slowly for about eight hours. Artificial stone made from broken slag and slag cement is coming into favor, as is also concrete made from the same materials. Thus one firm specially prepares slag for all classes of concrete at prices ranging from $1.50 to $1.75 per ton to pass ⅝ to 1-in. screens and machine-broken slag between 2¼ and 2½-in. mesh is offered at 60c. per ton. The same firm also quotes prices for lintels, sills, copings, sinks, garden edging, and many other articles made of slag concrete. A French works makes brick from a mixture of lime and slag dust, 1800 being made from a cubic yard of the mixture and costing 24c. per hundred to manufacture. A brick press of ordinary type is used and the brick are stocked for six or seven months before they are used so that they may harden properly.

Slag cement for some years was under a cloud, partly because it was not ground sufficiently fine and partly because slag was used to adulterate Portland cement. Much care is requisite in the selection of the slag, which should be of uniform composition and fairly free from the exceedingly detrimental element sulphur. It should be ground so fine that all of it passes a sieve with 75 meshes to the linear inch, and not more than 15 to 20% should remain on a screen of 180 mesh. This operation is necessarily costly, although it is reduced considerably by granulating the slag. The cement weighs 85 to 95 lbs. per bu., or over 15% less than Portland cement, for which it may be substituted under practically all circumstances. In lots of 50 tons this cement sells at about $1.30 per bbl. of 330 lbs. Much of it is utilized in the manufacture of paving tiles, bricks, and blocks, forming a mixture of broken slag or gravel and sand with the cement in variable proportions. The mass is molded wet under hydraulic pressure as is done in making artificial paving stone. The cost amounts to about 75c. per sq. yard.

Slag wool is another product which finds a ready sale at about $50 per long

ton. It is extensively used for pugging in floors and roofs, casing iron work and steam pipes, and for insulating, fireproofing, and packing purposes generally. For its production a strong jet of air or steam is caused to impinge on a stream of slag melted down with a suitable admixture of silica in a reverberatory furnace. Fine shot of slag with long, tail-like appendages are thus produced and by bending at right angles the passage for the laden air current these shot are separated while the hair or wool is carried forward to a settling chamber. The coarse material separated is in an excellent state for further grinding for the manufacture of cement.

The utilization of the waste heat in slag is a problem still awaiting solution, although several plans have been proposed and carried into effect on a more or less considerable scale in the way of evaporating brine, raising steam, transferring the molten slag directly to glass furnaces, etc.

By-products from Blast-furnace Gases.—The recovery of by-products from blast-furnace gases stands in much the same position that it did a year or more ago. Those companies in Great Britain which laid down plant for separation of ammonia and tar have continued on the whole to do well, and on the European continent similar results have been attained, with this addition, that in one instance iodine is found to occur in the water after it has been used for washing the gases. At the Esch blast furnace, making 100 tons of iron daily, there is obtained 1600 lbs. of potassium iodide, 6 tons of ammonia, and 75 tons of potassium chloride in the year, these products being valued at $5750.

Basic slag, concerning which it was prophesied some years ago that it would become the main product with steel as the by-product, is now so thoroughly well established in favor that the only question is as to its method of valuation. German papers especially have been full of discussion on the trustworthiness of the Wagner and other citrate methods, and the result appears to be that as the method is purely empirical it must, if it is to be of any value, be carried out on very definitely prescribed lines. In the early period of the basic-steel industry, Germany absorbed all of the basic slag that was produced, but now it is extensively employed in England and France as well, and the price fluctuates around $7 to $8 per long ton in bags. The quantity of phosphoric acid fluctuates between 8 and 21% and, though most if not all of this is finally of manurial value, it is important that as large a proportion as possible should be soluble in the citrate solution so as to be taken up quickly by vegetation. On this account slags richer in silica are generally to be preferred, but it is even more important that the slag should be very finely divided, and a good standard is that 70 to 90% should pass through a 100-mesh sieve. As showing the importance of the basic-slag industry in England, it might be mentioned that it is one of those set down for investigation by a committee appointed to inquire into certain miscellaneous dangerous trades, of which one is the manufacture of silicate or slag wool.

THE DUQUESNE WORKS OF THE CARNEGIE STEEL COMPANY.

The new Duquesne blast furnaces of the Carnegie Steel Company constitute an unique plant. There are four furnaces, 560 ft. apart, 100 ft. high, 20 ft. diameter of bosh, 12 ft. wells, and 10 tuyères. Each furnace has four Cowper stoves, 90 ft. high and 20 ft. diameter, with a flame flue in the center, and a separate stack to each. For each furnace a separately covered casting shed is provided, and a tramway for carrying the charges, usually consisting of 5 tons of ore in a car, to the top of the furnace, tipping automatically on to a bell, and dispensing entirely with fillers and chargers. Each of these mechanical fillers has a separate steam-haulage engine. Each pair of furnaces has a battery of Babcock & Wilcox tubular boilers and blowing engines, which are of the usual vertical self-contained compound condensing type, with steam cylinders 28-in. and 42-in. diameter, by 5 ft. 6 in. stroke. Each group of two furnaces has five engines, so there is one always in reserve. The engines work under 120 lbs. steam pressure and deliver blast at 15 lbs. pressure at the tuyères, but they can be forced up to 25 lbs. Besides the blowing plant, there are four pairs of vertical compound condensing pumping engines, pumping to the water-tower for the supply of the tuyères. The water-tower is of wrought iron, and 120 ft. high. A pig-iron breaker is provided for each furnace, and the pigs are cast in chills 26 ft. long. All the overhead cranes and the pig-iron breaking machines are driven by electric motors. Separate motors are provided for incandescent and arc lighting. At furnaces Nos. 1 and 2 the air supply has been at an average rate of 38,000 cu. ft. per minute; at Nos. 3 and 4 an average of 50,000 cu. ft. will probably be attained. It is the practice at the Duquesne Works, as at the Edgar Thompson to maintain the temperature of the blast at 1000° F. as compared with 1200° to 1300° at other works at Pittsburg.

The new furnaces at Duquesne, of which only two are yet in operation, have more than realized the anticipations as to their output. It was expected that they would produce an average of 500 tons a day per furnace, but since they were put in blast they have done more than that, and one of them has produced as much as 690 tons in a day. The best month's run at the Duquesne Works was 17,182 tons, the best week's 4294, or a daily average of 613 tons; the best record for a day has been 690 tons. The plant of four furnaces may be regarded as equal to an output of about 800,000 tons of pig iron per annum. The cost is understood to have been from $700,000 to $750,000 per furnace. An additional furnace is now under construction at Duquesne of a still larger size, in which there are some marked advances. The hearth is to be 15 ft. diameter and 7 ft. deep, and there will be 21 tuyères in two rows. This new furnace is expected to make 1000 tons of pig iron a day.

The steel works at Duquesne, the most recently built of the series owned by the Carnegie Steel Company, are located close to the new blast furnaces on the opposite side of the Monongahela River from McKeesport and about two miles below. The entire product is made into billets and bars, and for this purpose the mill is probably the finest in the world. With only two nominally 10-ton converters, but really 8½ tons, as much as 38,000 tons of ingots have been made in a month.

LEAD.

By Walter Renton Ingalls.

Notwithstanding the low prices, and the strike at Leadville, which caused a suspension of work there for six months of the year, the production from domestic ores in 1896 was large, amounting to 174,692 tons, of which 135,332 tons were desilverized lead, 33,428 tons were soft lead from the Missouri and Kansas districts, and 5932 tons were hard or antimonial lead. In addition to the domestic production there were 80,159 tons imported in all forms, chiefly as base bullion from Mexico and Canada.

The domestic consumption showed a decrease, amounting to 218,319 tons, against 232,854 tons in 1895. There was, however, a very large increase in exports, the total reported being 52,483 tons, as against 18,130 tons in 1895. This increase absorbed the surplus portion of the supply, so that the stocks at the close of the year showed a slight reduction from those at the opening.

The lead industry of the United States has now become so complicated that it is impossible to distribute the production among the States in which it originated. The statistics are compiled from returns of the refiners who finally turn it out in marketable form. They can, of course, trace the base bullion back to the smelters, but the latter receive ores from numerous districts, in some cases from sources that they do not know, and in any event can hardly specify how much lead from the mines of any particular State entered the base bullion which they shipped. Indeed it is no longer even possible to assume that the production of desilverized lead represents the production of the Mississippi Valley, since Joplin ore is carried to Colorado, where it is smelted with silver ores, and finally enters the market as desilverized.

PRODUCTION AND CONSUMPTION OF LEAD IN THE UNITED STATES. (IN SHORT TONS.)

Year.	Stock of Refined, Jan. 1.	Produced from United States Ores.				Imported in Ores and Bullion.	Total Supply.	Consumed.	Exported in all Forms.	Stock of Refined, Dec. 31.
		Desilverized.	Soft.	Antimonial.	Totals.					
1892	9,000	145,217	31,655	5,805	182,677	40,883	232,560	214,186	12,874	5,500
1893	5,500	130,026	31,369	5,283	166,678	64,270	236,443	195,457	35,000	5,991
1894	5,991	119,486	37,044	4,337	160,867	69,741	236,599	191,146	40,000	5,453
1895	a12,634	119,057	32,797	5,000	156,854	100,471	269,959	232,854	18,130	a18,975
1896	a18,975	135,332	33,428	5,932	174,692	80,159	273,926	218,319	52,483	a13,024

(a) Includes lead in bond, which amounted to 4124 tons at the end of 1896, and 9865 tons at the end of 1895.

PRODUCTION OF WHITE LEAD IN THE UNITED STATES.

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1893.....	88,500	80,286	$9,469,500	$107.00	1895.....	95,389	86,537	$9,061,965	$95.00
1894.....	87,242	78,155	8,445,174	108.00	1896.....	94,818	86,018	7,832.915	82.61

Antimonial lead is produced as a by-product in refining base bullion. The skimmings from the softening furnaces are put aside and when accumulated in sufficient quantity are liquated in a reverberatory furnace to eliminate metallic lead as far as possible and also the silver. In this way the skimmings are reduced to 3 ozs. silver or less. The desilverized skimmings are then smelted in a small blast furnace with an equal weight of slag, from 13 to 28% of non-argentiferous galena and 10 or 11% of coke. The amount of galena is regulated so that no matte will be formed, its sulphur acting as a reducing agent. A good deal of antimony goes into the slag, but the more part is reduced with the lead, forming an alloy known as "hard lead." The percentage of antimony in the latter varies according to the conditions under which it is made. Formerly, when it commanded a premium over soft lead, it was designed to contain only 10 or 12%, but now it is at a discount of about ½c. per lb. from the price of ordinary lead, and it is therefore an object to produce as little lead as possible in this form. Consequently the greater part of the "hard lead" made at present is run up to from 18 to 20% antimony, and certain smelters make it as high as 35 or 40%. This antimonial lead contains generally about 6 ozs. silver per ton, and sometimes as much as 10 ozs., which it does not pay to separate.

The make of antimonial lead is consumed partly in the preparation of alloys of which both lead and antimony are constituents, type metal for example, although the type founders generally prefer to use refined antimony in their melting pots. A large amount of antimonial lead is consumed in the manufacture of cases for funeral caskets. This is, of course, a final consumption, since metal used therefor is not likely to appear on the market again as old material.

Colorado.—The production of Leadville suffered from the strike, which practically put a stop to all production during the last six months of the year. According to the local correspondent of the *Engineering and Mining Journal* the total production was 40,500 tons of carbonate ore, 194,059 tons of argentiferous and manganiferous limonite, 106,308 tons of sulphide, and 54,452 tons of silicious. The production of lead by the Leadville smelters and by Valley smelters from Leadville ores is reported as having been 18,170 short tons. This figure is undoubtedly too high, however, since the Leadville smelters buy ores in other districts, the metal yield of which is counted as Leadville production.

Idaho.—The labor difficulties in the Cœur d'Alêne district having finally been settled it was possible to do more mining, and the production increased largely. Several new dressing works were erected, and the Helena & Frisco property was purchased by the Exploration Company of London, which it was

said proposed to secure other mines in the Cœur d'Alêne and develop them on a larger scale. The silver-lead veins of the Cœur d'Alêne are very extensive, and with freedom from interference by labor organizations the companies can work profitably. It is to this district that American smelters now have to look for the largest supply of lead ore.

Southeastern Missouri.—The production of lead in this district is steadily increasing. The ore deposits are large and their exploitation is carried on by strong companies using modern methods and economical plants. There is good reason to believe that lead could be produced by these companies without loss if the price were as low as 2¼c. per lb. at St. Louis.

Joplin District.—Lead mining in this district, covering the southwestern corner of Missouri and the southeastern part of Kansas, is governed chiefly by the conditions that rule in the zinc market, since the lead and zinc ores generally occur together, and the latter are by far the more important. The Joplin correspondent of the *Engineering and Mining Journal* reports that in 1896 the prices for lead ore were very unsatisfactory. Sales were consequently light, and at the end of the year there was a stock of about 1000 tons of ore on hand. The total production of the district was 26,927 tons of 2000 lbs., against 31,294 in 1895.

The ore of the Joplin district is sold chiefly to the Picher Lead Company (producer of pig and white lead) of Joplin, and to Case & Serge of Grand Falls. Recently, however, a good deal has been shipped to the silver-lead smelters of Colorado, to be used in the reduction of dry silver ores, whence this lead finally appears in the market as desilverized.

The average monthly price of lead ore in Joplin per 1000 lbs. in 1896 was as follows: January, $17.75; February, $17.50; March, $17.25; April, $16.50; May, $16.50; June, $16.50; July, $16.00; August, $14.75; September, $14.00; October, $14.25; November, $15.00; December, $16.50. The average for the year was $16.04.

Utah.—The production of lead in Utah increased slightly, notwithstanding the ore output of the Bingham mines fell from 75,000 to 54,000 tons. In Tintic and Park City, however, there was much activity.

British Columbia.—The production of lead in this province in 1896 was 10,977 metric tons. Practically the whole of this came into the United States, partly in ore and partly as base bullion. A good deal of the former was smelted and refined in bond in Colorado, the soft lead being shipped to Liverpool via Galveston. The base bullion produced in British Columbia goes entirely to San Francisco, where of course soft lead commands a comparatively high price so long as the supply is not excessive.

The principal lead-producing district of British Columbia is the Slocan, situated in the division of West Kootenay, a short distance north of the international boundary. In an area of 15 by 25 miles have been developed many high-grade veins of silver-lead ore and about 50 are now making shipments. According to Professor William Carlyle, the government mineralogist, the production in 1896 was 18,215 short tons of ore, yielding 2,141,088 ozs. of silver and 9605 tons of lead, an average of 117.4 ozs. silver and 52.7% lead per ton.

The majority of these mines are located near the summits of high, precipitous mountains, at an elevation of between 5500 and 6500 ft. above sea level. Although many of the veins are capable of being worked by tunnel, and the cost of mining is consequently low, it is necessary in order to work them economically to provide tramways, and dressing works are also desirable. So far but little foreign capital has gone into the Slocan, and development has been slow. It has been carried out almost entirely by Americans. Mining conditions, however, are favorable. There is an abundance of water and timber, and the climate is good. The Columbia River and short railways place the various camps in close connection with the Canadian Pacific upon the north and the Northern Pacific and Great Northern railways to the south, and the construction of the Crow's Nest Railway will provide cheap fuel from the great coal deposits in the Rockies.

The largest body of ore in the district is in the Slocan Star mine, which in some places is 25 ft. wide. Many of the veins in the district, however, are small. The ore is argentiferous galena with zinc blende and a little tetrahedrite in a matrix of quartz and spathic iron. The veins cut across stratified and eruptive rocks, often granite. Besides the sulphide ore a certain amount of carbonate is produced from the upper levels.

England.—The lead mines on the borders of Cumberland and Northumberland, at Nenthead, which have been worked for centuries in a very primitive manner, have been taken over by the Vieille Montagne Company, of Belgium. An increased force of men has been put to work and much money is being spent on development and metallurgical machinery. It is expected that the result will be an important revival in English lead mining.

Germany.—It is reported that the Prussian railway authorities intend to reduce the rates for the conveyance of ores to the German lead works. The native supply of lead ore is nearly exhausted, and the works are now largely dependent on the poorer Australian ores. Owing to English competition it is thought advisable to reduce the railway rate to 0.5c. per ton kilometer.

Mexico.—There are no official statistics of the production of lead in Mexico. This can be arrived at, however, with sufficient accuracy by an analysis of the statistics of the United States, since the entire output of Mexico reaches the United States either in the form of ore or base bullion. The United States also receives lead in ore and bullion from British Columbia, and lead in ore from Peru and Chile. The South American supply is so far very small (though it is likely to become more important), and at present it may be assumed that the imports of lead into the United States, less the production of British Columbia, is the production of Mexico. Proceeding in this manner it appears that the Mexican production in 1896 was 61,743 tons, or allowing a little for South American ore, say 61,000 tons, against 80,000 in 1895.

This great decrease is attributable chiefly to the Sierra Mojada mines, where the most easily accessible ore bodies are approaching exhaustion. This is probably true also to a certain extent in other mining districts of Mexico. The establishment of the smelting industry in that country a few years ago opened a market for many ores which could not previously be mined profitably. With great reserves to draw from their production at first increased enormously, but

as they have been extracted new bodies of equal magnitude and quality have not been opened. The decrease in the production of lead in Mexico may be regarded therefore as permanent.

Besides the falling off on the part of the Sierra Mojada mines the output of Mexico was also affected by the cave-in of one of the most important producers of the Chihuahua Mining Company in the Santa Eulalia district.

The smelters of Monterey, San Luis Potosi, and Aguas Calientes, all but one of whom are American companies, operated successfully, and at the end of the year plans were being laid for a new four-furnace works at Chihuahua. The Compañia de Peñoles, operating the Mapimi mines, constructed a narrow-gauge railway from its property to the main line of the Mexican Central, and is pushing operations on a larger scale. The works at Culiacan, Sinaloa, were obliged to suspend, however, owing to insufficient ore supply, and the strong competition of the Mazatlan buyers, who export to San Francisco, and elsewhere.

By far the more part of the Mexican lead is shipped in the form of base bullion by rail to Tampico, and thence by the Ward line of steamers to New York, for refining at Newark and Perth Amboy, or, in the form of ore and base bullion, by rail to El Paso, Tex., and Kansas City, Mo. A small amount of ore goes to San Francisco and Guaymas through Mazatlan and the Pacific ports, and some comes from Yucatan directly to New York.

Spain.—According to Señor Don Roman Oriol the total production of lead in Spain in 1896 was 170,790 tons against 160,786 tons in 1895. The condition of the lead mining industry in this country is particularly treated by Mr. Barrington in the subjoined article.

LEAD PRODUCTION OF THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Austria.	Belgium.	Canada.	France.	Germany	Greece.	Hungary	Italy.	Japan.
1891	7,583	12,698	267	6,680	95,615	14,528	2,173	18,500	803
1892	7,252	10,146	802	8,776	97,742	14,426	2,335	22,000	820
1893	7,212	12,006	968	8,119	94,659	14,534	2,514	19,898	941
1894	7,570	14,120	2,587	8,758	100,753	14,434	2,113	19,605	.....
1895	8,085	15,573	10,467	.....	111,058	19,845	2,277	20,353	.....
1896	.....	.....	10,977	.....	113,792	15,180	.....	.....	.....

Year.	Mexico.	New South Wales.	Russia.	Spain.	Sweden.	United Kingdom. (*b*)		United States.	Totals.
						Foreign Ores.	Native Ores.		
1891	30,186	56,476	558	159,085	299	16,697	32,733	160,347	615,228
1892	47,532	46,656	883	219,260	799	14,922	30,013	186,548	710,912
1893	60,525	59,775	844	169,318	461	8,013	30,173	152,080	642,040
1894	57,000	43,225	743	152,620	330	12,678	30,162	145,906	612,604
1895	80,000	30,182	.....	160,786	1,256	25,836	29,464	142,298	.....
1896	61,000	.....	.....	170,790	.....	.....	.....	.....	.....

(*a*) The statistics for Austria, Belgium, Canada, France, Germany, Hungary, Italy, Japan, New South Wales, Russia, and Sweden are from the official reports of the respective governments. Those for Greece are based on the authorities given under the general table of mineral production of Greece in a subsequent part of this volume. Those for Mexico and the United States are from data collected by THE MINERAL INDUSTRY. Those for the United Kingdom as specified in note *b*. Those for Spain are the official figures, except for 1896, which are given as reported by Señor Don Roman Oriol.

(*b*) The production of lead in the United Kingdom is given in two columns. One gives the amount of lead derived from domestic ores, the yield of which is calculated at 95%, as reported in the official British blue books. The other column gives the production of lead in the United Kingdom from foreign ores smelted there. These figures, which are not reported in the official blue books, are obtained by deducting the production of British lead from the total output of the lead smelters of the United Kingdom as stated in the *Statistische Zusammenstellungen über Blei, Kupfer, Zink and Zinn*, of the Metallgesellschaft, Frankfurt-am-Main, 1894 and 1895.

LEAD PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS.)

Year.	Murcia.	Jaen (Linares).	Cordoba.	Almeria.	Guipuzcoa.	Totals.
1894.....	85,465	31,529	17,795	13,295	4,536	152,620
1895.....	79,636	38,847	25,260	13,707	3,336	160,786
1896.....	85,000	40,500	26,000	15,000	4,290	170,790

(*a*) Figures for 1894 and 1895 from the *Annario de la Mineria, Metalurgia, y Electricidad de España* Madrid, by Señor Don Roman Oriol; figures for 1896 from the *Revista Minera*, Feb. 8, 1897.

PRODUCTION, IMPORTS, EXPORTS, AND CONSUMPTION OF LEAD IN THE CHIEF COUNTRIES OF THE WORLD. (IN METRIC TONS.)

		Aust'a-Hungary. (*a*)	Belgium. (*b*)	France. (*c*)	Germany. (*d*)	Great Britain. (*e*)	Italy. (*f*)	Russia. (*g*)	Spain. (*h*)	Switzerland (*i*)	United States (*j*)
1891	Production	9,756	12,698	6,680	95,615	49,430	18,500	558	159,085	*Nil.*	176,851
	Imports	4,561	34,093	74,502	17,625	172,439	4,803	20,336	*Nil.*	2,800	25,777
	Totals	13,317	46,791	81,182	113,240	221,869	23,303	20,894	159,085	2,800	202,628
	Exports	255	24,073	10,657	24,971	65,289	2,572	*Nil.*	146,029	129	2,700
	Consumption	13,062	22,718	70,525	88,269	156,580	20,731	20,894	12,956	2,671	199,928
1892	Production	9,587	10,146	8,876	97,742	44,935	22,000	883	219,260	*Nil.*	182,677
	Imports	7,348	31,054	72,721	17,501	185,707	2,396	26,075	*Nil.*	3,020	40,883
	Totals	16,935	42,200	81,597	115,243	230,642	24,396	26,958	219,260	3,020	223,560
	Exports	202	24,709	10,057	25,647	74,956	3,017	*Nil.*	154,969	114	112,874
	Consumption	16,733	17,491	71,540	89,596	155,686	21,379	26,958	64,291	2,906	210,686
1893	Production	9,726	12,006	8,119	94,659	38,186	19,898	844	169,318	*Nil.*	166,678
	Imports	6,286	38,028	77,679	23,857	191,260	1,870	28,712	*Nil.*	3,159	64,270
	Totals	16,012	50,034	85,798	118,516	229,446	21,768	29,556	169,318	3,159	230,948
	Exports	270	29,947	9,243	23,945	68,790	3,752	*Nil.*	158,737	89	35,000
	Consumption	15,742	20,087	76,555	94,571	160,656	18,016	29,556	10,581	3,070	195,948
1894	Production	9,683	14,120	8,758	100,753	42,840	19,605	743	152,620	*Nil.*	160,867
	Imports	8,842	43,491	84,674	24,208	164,450	1,499	30,755	*Nil.*	1,412	69,741
	Totals	18,525	57,611	93,432	124,961	207,290	21,104	31,498	152,620	1,412	230,608
	Exports	242	34,690	8,133	24,354	61,869	2,157	*Nil.*	158,735	.....	40,000
	Consumption	18,283	22,921	85,299	100,607	145,421	18,947	31,498	.....	1,412	190,608
1895	Production	10,362	15,573	.....	111,058	55,300	20,353	.....	160,786	.....	156,854
	Imports	8,974	45,594	.....	28,449	179,882	1,780	.....	*Nil.*	.....	100,471
	Totals	19,336	61,167	.....	139,507	235,182	22,133	.....	160,786	.....	257,325
	Exports	208	39,996	.....	27,855	70,148	3,646	.....	151,129	.....	18,130
	Consumption	19,128	21,171	.....	111,652	165,034	18,487	.....	9,657	.....	239,195

(*a*) From *Statistisches Jahrbuch des K. K. Ackerbau Ministerium* and *Magyar Statisztikai Evkönyv.*
(*b*) Production from *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques.* Imports and exports from *Annuaire Statistique de la Belgique.*
(*c*) From *Statistique de l'Industrie Minérale.*
(*d*) Production, imports, and exports from *Statistisches Jahrbuch für das Deutsche Reich.*
(*e*) Production from statistics of the *Metallgesellschaft*, Frankfurt-am-Main. Imports and exports from Board of Trade returns.
(*f*) From *Rivista del Servizio Minerario.*
(*g*) From *Sbornik Statisticheskikh Svedenie o Gornozavodskoi Promyshlennostie Rossie v zavodskom Godu*, St. Pet rsburg.
(*h*) From the *Reports of the Comision Ejecutiva de Estadistica Minera.* Imports and exports from the *Revista Minera de España.*
(*i*) From *British Statistical Abstracts* and from figures furnished by the *Direction Générale des Douanes Fédérales*, Berne.
(*j*) The statistics of production are those collected by THE MINERAL INDUSTRY; those of imports and exports are from the *Reports of the Bureau of Statistics*, Washington.

The New York Lead Market in 1896.

During 1896 there was a great depression in the value of the metal, due mainly to the poor consumption, especially during the summer months, and also to a very heavy production. While it was feared a few years ago, when the duty on foreign importations, either in the form of pig lead or ores, was reduced from 2c. to 1c. per lb. of pig lead, that home production would materially suffer, just the reverse has been the case, and from year to year a rather large increase in the production has been noticeable. Nor is this unnatural. Low prices of an article always means the greatest economy in producing and refining and reducing general expenses, and this is usually best attained when the output of a single producer is enlarged as much as possible. Neither had the decline in silver any influence on the production of lead. More than 80% of the production of American lead comes from ores carrying gold and silver, and very often it is then considered as a by-product only, the precious metals being the main components of the ores. But the output of soft lead, which comes mainly from Missouri, has also increased, though during the time of the greatest depression in the summer hardly any producers were able to make both ends meet, and that exercised a check on the rapidly increasing production.

Of the lead refined in bond in the United States practically every pound was again exported, and thus the industry has been carried on without any detriment to home producers, while our refiners have been able to successfully compete with Europe, much to the latter's chagrin. The supply of lead bullion from Mexico continued to increase.

The stocks at the beginning of 1896 were hardly heavier than 12 months previous thereto, and the opening prices for the year were moderately low at about 3@3⅛c., which was then considerably lower than the average of the preceding year. Fair business was doing at these figures during the month of January, and in February a decidedly better demand set in, prices advancing slowly to 3¼c., which latter price unfortunately proved to be the highest quotation of the year.

In March quotations fell off, and during all that month, April and May values remained fairly steady at 3.05@3.10c. Toward the end of the latter month renewed flatness set in and a fairly large business took place, when at the end of June the market was disturbed by reports from Leadville that a rather serious strike had broken out, which threatened to have a decided influence on the production and value of lead. The Miners' Union had decided not to allow their members to enter the mines except at a minimum pay of $3 per day, while mine owners, under the depressed values of both lead and silver, did not see their way to submit to this dictation. A long and bitter fight followed, during which production suffered, but the poor consumption then existing was more than an offset for this shortage, and the main sufferers were the Leadville camp and its mining industries.

The political situation at this time of the year had a marked influence on the article, and after the Chicago convention deliveries of refined lead continued to fall off more and more; in the main centers of the United States

money became decidedly dearer, and some of the banks in the West were unable to give any support even to the best of their clients. This led to rather large realizations on the stocks then accumulated, and consequently a very large decline set in, which by the end of July brought prices down to 2⅞c., New York, followed by a still heavier drop in August, when large quantities were marked down to 2.70c., and by the end of the month as low as 2.60c., New York, was accepted for several lots. This was the lowest price ever reported in the history of lead in the United States. Even some 20 years ago on the occasion of the great discoveries in the Leadville camp, and when the fear of a deluge of lead made the trade rather nervous, values never came down as low as during this time. It must be stated that this figure in New York was rather below the figures accepted in the West, as the lowest prices on record are 2.45c., St. Louis, and 2.50c., Chicago, which are fully $2 per ton more than the then New York price; but this difference can mostly be explained by the fact that large quantities of lead could only be marketed in the East, and that therefore the buyers had to be given larger concessions.

After the first pressure was over values slowly advanced to 2.70c., and by the end of September 2.80c., New York, was freely paid. Then with approaching elections and the uncertainty at that time prevailing in the trade, a decline again set in, and by the middle of October 2.72½c., New York, and 2.50c., St. Louis, were the highest prices obtainable. After that a marked improvement was noticeable, and before the end of the month nothing could be had below 2⅞c.

The month of November showed a very active trade; there was a good demand all around, and stocks, which had accumulated to a considerable extent at the different smelting works, could easily be marketed at continually rising prices. Thus at the end of November the 3c. mark in New York and 2.75@2.80c. in St. Louis were again established. Even during the month of December, when outdoor work begins to be interrupted and when prices usually show a decline, the firmness continued, as the quantities offered continued light in spite of the fact that some second-hand lots, purchased at low prices during the depression in the summer, were pressed for sale, the market closed fairly strong at 3.02½@3.05c., New York, and 2.77½@2.80c., St. Louis.

The following table shows the average price of lead in New York for each month of the last five years, complied from the weekly reports given in the *Engineering and Mining Journal.* For the years prior to 1892 statistics will be found in THE MINERAL INDUSTRY, Vols. I.–IV. The quotations are for spot lead in carload lots or over:

AVERAGE MONTHLY PRICES OF LEAD IN NEW YORK, IN CENTS PER POUND.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1892	4.20	4.12	4.21	4.15	4.22	4.16	4.13	4.11	4.11	4.02	3.84	3.80	4.09
1893	3.87	4.22	3.96	4.08	3.89	3.77	3.58	3.41	3.80	3.51	3.41	3.27	3.73
1894	3.19	3.31	3.37	3.43	3.39	3.31	3.50	3.41	3.17	3.12	3.14	3.10	3.29
1895	3.10	3.12	3.12	3.08	3.16	3.25	3.25	3.50	3.35	3.33	3.25	3.22	3.23
1896	3.08	3.19	3.14	3.07	3.03	3.03	2.96	2.73	2.77	2.80	2.96	3.04	2.98

The average price for the year was the lowest on record. The lowest average was 2.73c. in August, and the highest 3.19c. in February.

The St. Louis Lead Market in 1896.

The position of St. Louis in proximity to a large lead-producing region, to which it is also the center for supplies, gives it an importance as a primary market which no other city in the United States can claim. Notwithstanding the depression in business and the low prices obtained for the products, the receipts for 1896 were the largest ever reported, and the same may be said of the shipments. Much complaint was made by the producers of high freight rates, but all efforts to obtain concessions from the railroads failed. A considerable quantity of metal continued to come to the city by boat. The arrivals by river in 1896 were 26.2% of the total, against 22% in 1895 and 24% in 1894.

The following table shows the receipts and shipments at St. Louis for five years ending with 1896, in tons of 2240 lbs.

RECEIPTS AND SHIPMENTS OF PIG LEAD AT ST. LOUIS.

Source of Supply.	1892.	1893.	1894.	1895.	1896.
	Long Tons.	Long Tons.	Long Tons.	Long Tons.	Long Tons.
By railroad	54,401	42,106	38,889	41,673	51,145
By boat	116	6,056	13,369	11,931	18,300
Total receipts	54,517	48,162	52,258	53,604	69,505
Shipments	38,235	34,586	38,724	34,162	50,167

The total increase in receipts was 15,901 tons, or 29.7%, while the gain in shipments was 16,005 tons, or 46.8%, as compared with 1895. The railroad receipts were exceeded in 1892, but the total in that year was much below that of 1896, the receipts by river having grown largely during the past five years.

In prices the general course of the St. Louis market followed that of New York with some closeness, but several times during the year the St. Louis quotation was above that of New York, allowance being made for freights. Production continued steadily most of the year, with prospects of an increase to be expected in 1897.

The following table shows the monthly range of prices at St. Louis, in cents per pound, during 1896:

RANGE OF PRICES OF LEAD PER POUND IN ST. LOUIS.

Month.	Highest	Lowest	Month.	Highest	Lowest	Month.	Highest	Lowest	Month.	Highest	Lowest
	Cents.	Cents.		Cents.	Cents.		Cents.	Cents.		Cents.	Cents.
January	2.95	2.75	April	2.88	2.75	July	2.83	2.60	October	2.70	2.45
February	3.03	2.77	May	2.85	2.77	August	2.63	2.43	November	2.85	2.60
March	3.00	2.75	June	2.83	2.75	September	2.60	2.50	December	2.85	2.70

August showed the lowest prices and February the highest. The lowest price quoted was 2.43c. and the highest was 3.03c. The range of the year was therefore 0.60c. At the close of the year 2.80c. was the quotation.

The Joplin Lead Ore Market in 1896.

Lead mining in this district is governed by the same conditions that rule in regard to zinc mining, as in most cases the same ground produces both ores. In 1896, however, the pig lead market was in so unsettled a condition that prices were extremely unsatisfactory for ores and sales light; while there was at the close of the year a surplus on hand of about 2,000,000 lbs. remaining to be sold.

The output of lead in 1896 was 53,853,920 lbs. as against 62,587,470 lbs. for 1895. The falling off in the production to the extent of 8,733,550 lbs. was due to the low price and the lack of prospect work which would have been carried on in a fairly prosperous year. Towards the close of the year there were signs of increased activity in mining.

Joplin has an excellent local market for lead. The Picher Lead Company, whose works are very extensive, and which makes white lead as well as pig lead, and the Case & Serge Smelting Works at Grand Falls, about five miles from the city, consume large quantities of ore. A good deal is also shipped to Colorado for fluxing silver ores.

The average price for the year was $16.04 per 1000 lbs. and the average monthly prices were as follows:

LEAD ORE PRICES AT JOPLIN IN 1896.

Month.	Per 1,000 Pounds.	Month.	Per 1,000 Pounds.	Month.	Per 1,000 Pounds.	Month.	Per 1,000 Pounds.
January.....	$17.75	April.....	$16.50	July.....	$16.00	October.....	$14.25
February....	17.50	May.....	16.50	August.....	14.75	November.....	15.00
March.....	17.25	June.....	16.50	September....	14.00	December.....	16.50

The lowest price was reached in September; in October there was a slight recovery, which became marked in November and December. The closing price was $1.25 per 1000 lbs. below that of January, but showed an advance of $2.50 over the lowest, in September. The extreme range of prices was $3.75.

The London Lead Market in 1896.

The prevailing note of this market during 1896 was the comparative steadiness of values, the range of which was considerably less wide than that of the preceding year. In 1895 the main tendency was upward, a rise of about £2 per ton taking place from the low level which had been touched in 1894. During 1896 the value varied only to the extent of about £1 from highest to lowest.

January opened with a shade more inquiry from consumers, and sellers of soft foreign raised their price very slightly, £11 6s. 3d. being asked, but the demand was only momentary, and the month was mainly marked by apathy and a downward tendency, provincial consumers holding aloof and most rollers complaining of the lack of orders; £11 1s. 3d. was the lowest point touched and the value eventually stiffened again toward the close of the month, the quantities of Australian lead which were arriving being readily taken up, while America seemed disinclined to sell. We may here note that these arrivals of large quantities of Australian desilverized lead remained throughout

the year a regular feature of the market, and were, doubtless, the chief reason why lead did not attain during the year a higher level of values. Early in February £11 6s. 3d. was paid for soft foreign lead, and, although there was no activity in consumptive demand, there was, on the other hand, very little lead offered, and £11 8s. 9d. was soon reached. Toward the end of the month the first really brisk demand of the year made itself felt and a large business with the provinces took place at up to £11 10s.

America reported a firm market with price at 3.17½c., and the subsidence of provincial demand was counteracted by the giving out of government orders, and by renewed buying in London. The government again bought early in March, but the slackness of orders for sheet and pipes acted as a depressant, and many rollers sought to have deliveries postponed. This, with the sale of some large parcels of newly arrived Australian lead, and the fact that further 1500 to 2000 tons were in sight, flattened the market down to £10 18s. 9d. At about the lowest limit, the Americans also made sales, but more buyers coming out at the cheap prices, we witnessed a recovery to £11 5s., followed, however, by a relapse to £11 1s. 3d. April brought no improvement; on the contrary, with the constant arrivals of Broken Hill lead (the shipments for March alone amounted to 4500 tons), and with renewed sales of American lead, the general trend of values was downward, and there was no demand worth naming to counteract the depression. Ultimately, £10 17s. 6d. was reached, but just at the close a spurt in demand induced a rally to £11 2s. 6d. for soft foreign. May, too, was a dull month, with dealers depressing the market by sales; and with the exception of a brief spell of consumptive inquiry during the third week, nothing occurred to relieve the monotony of the market, while no noteworthy alteration in values took place. Throughout June the same unsatisfactory conditions prevailed, for the most part, though toward the close of the month the firmer reports from America, where the big strike at Leadville seemed likely to be prolonged, caused a momentary firmness here, and were accompanied by some signs of life on the part of consumers. All through the month values remained in the near neighborhood of £11, and this statement holds good also of July, a transitory scarcity of London lead being without any very appreciable effect on values, £11 2s. 6d. being the highest price paid for prompt lead. America continued to be a seller at cheap prices, and as to the provinces they seemed to be fairly well bought.

August opened in steady tone without pressure of sales, but the scarcity of spot lead having been relieved, the value eased off to £10 18s. 9d. and touched, about the end of the second week, under the influence of sales of second-hand lots, £10 16s. 3d. The lower values ruling in London induced provincial consumers to abstain from purchases, but a little speculative buying here hardened the price just a little, and before the month closed there was again a certain amount of difficulty experienced in getting early delivery, and this factor was aggravated in the following month (September), when American supplies were practically cut off, while Spain also remained out of the market, especially toward the close. The rise, which carried the value up to £11 6s. 3d., proved too rapid for most provincial consumers, and these, having missed their market, held off in the expectation of a relapse, while lead at current rates.

could be had only for forward delivery, for which there was but small demand. Consequently business was of a limited description.

During the greater part of October demand was of a limited description, and the scarcity of prompt lead having been relieved by the arrival of several parcels in the Thames early in the second week, while America began about the same time to press sales, the tone grew very flat and the article became almost unsalable. In contrast with this state of things, the last week of the month developed considerable firmness, chiefly due to the stiffening of rates of freight, especially those from Spain, which became almost prohibitive, from £11 per ton, buyers, prices improved before the close to £11 5s.

November witnessed a continuance of the upward movement, the freight difficulty, not only as regards Spain, but also America, having become greatly accentuated. A fair amount of business with the provinces was done at advancing prices and spot lead was very difficult to obtain. Ultimately £11 15s. per ton was paid for soft foreign and even £11 16s. 3d. was reported. On the other hand, London manufacturers gave little or no support to the market, professing early in the month to be covered, in some cases, to the end of the year; but the scantiness of early lead was very marked, and the falling off in the quantities of Broken Hill lead arriving (which had dwindled down to about 150 tons per week), contributed to this feature. At the close of November, however, the tendency was rather toward a relaxation of the tension, and demand being restricted, values eased off somewhat, the final figures early in December being £11 10s. for soft foreign, while English ruled as usual at 2s. 6d. to 3s. 9d. per ton higher.

Toward the end of the month, however, the market again showed a firmer tendency, and we have to record the following closing prices: £11 11s. 3d. to £11 12s. 6d. for Spanish and American lead, and £11 15s. to £11 16s. 3d. for English brands. The average price for the month of December, 1896, was about £11 8s. 9d. to £11 11s. 3d., which is higher than any quotation reported for many years past.

The following table shows the average price of Spanish lead in London monthly for each month of the five years ending with 1896, in sterling per short ton. Quotations for the years prior to 1892 will be found in Vols. I.–IV. of THE MINERAL INDUSTRY.

AVERAGE MONTHLY PRICES OF SPANISH LEAD IN LONDON, PER TON OF 2000 LBS.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
1892..	10 18 1	10 13 11	10 17 4	10 12 4	10 11 9	10 13 5	10 19 3	10 5 11	10 3 4	10 7 0	10 1 10	9 19 0	10 9 5
1893..	9 15 11	9 10 9	9 16 3	9 14 0	9 11 3	9 7 3	10 0 9	9 19 6	9 15 7	9 12 0	9 14 8	9 10 0	9 14 0
1894..	9 8 1½	9 5 7½	9 5 0	9 3 9	9 1 10½	9 2 6	9 9 6½	9 7 ½	9 17 6	9 17 6	9 14 2	9 11 3	9 8 7
1895..	9 12 6	9 11 3	9 15 0	10 0 0	10 2 6	10 6 3	10 12 6	10 18 9	10 18 9	11 3 9	11 12 6	11 8 10	10 15 3
1896..	11 3 6	11 7 1	11 3 3	10 19 3	11 1 2	11 0 3	11 0 7	10 17 3	11 3 4	11 4 2	11 10 0	11 10 11	11 3 5

The price of English lead is usually from 2s. 6d. to 4s. per ton higher than that of Spanish lead. The lowest price named was £10 17s. 3d. in August; the highest was £11 10s. 11d. in December.

Lead in Spain.

By Reginald W. Barrington.

The result of all kinds of mining in Spain during 1896 must be classed as highly satisfactory. Almost every branch has been affected by the general improvement in demand and price, and has still further been benefited by the high foreign exchange; English and French currency having been at a premium, fluctuating from 20 to 26% during the year.

The statistics of lead, collected from private sources, distinctly show that last year was one of revival both as regards prices and production. The amount of pig lead shipped from Spain during 1896 was 167,500 tons, as against 151,129 in 1895, an increase of 16,371 tons. The average price per quintal of pig lead delivered on wharf at Cartagena in 1896 was 57.77 reals; in 1895, 49.50 reals. While silver was worth 14.95 reals per oz. in 1896, against 13.94 reals in 1895.

The following table shows the export of pig lead from the district of Cartagena in 1895 and 1896, which was somewhat smaller than that of the two previous years, although the province of Murcia is still at the head of the lead-producing districts of Spain. This slight decrease in the export may be attributed to the great falling off in the amount of lead ore sent from the Linares district and formerly smelted and shipped from Cartagena.

Destination.	1895.		1896.	
	Bars.	Kilograms.	Bars.	Kilograms.
Marseilles	315,314	19,415,080	393,538	24,810,387
Newcastle	201,009	14,364,093	241,141	14,378,334
London	191,405	12,078,800	192,269	13,122,412
Antwerp	106,209	6,433,073	65,334	4,008,842
Liverpool	47,497	2,864,284	.....	.....
Hamburg	3,025	200,000	.....	.....
Genoa	.....	.....	2,178	150,000
Totals	864,459	55,355,330	894,460	56,469,975

The Province of Murcia, which comprises the two districts of Cartagena and Mazarron, may be said to be the source of production of about half the entire output of lead and 70% of the silver from the Peninsula, the next in importance being the province of Jaen, which comprises the Linares district, followed by Ciudad Real, Cordova, and Almeria. All the other provinces of Spain jointly produce under 10,000 tons of lead per year. The province of Guadalajara yielded during 1896 some 40,000 ozs. of silver from dry ores.

Cartagena during 1896 produced about 100,000 tons of lead ore, averaging about 58% lead and about 1250 gms. silver per ton. The chief feature of the year was the successful development of several of the mines at a much lower level than they had previously been worked; at this increased depth the lodes have been found in a highly mineralized condition.

Mazarron.—This district during 1896 produced about 93,000 tons of ore, averaging about 56% lead and 1250 gms. silver per ton. Within the past few

months considerable difficulty has been encountered with regard to ventilation, and a number of fatal accidents have occurred owing to engineers and miners being overpowered by the noxious gases in some of the mines now working the lode at a depth of over 400 meters.

Province of Ciudad Real.—This is the latest field for lead-mining enterprise in Spain, and the mines are being worked at present with very good results. Almost every lode in this district is said to yield ore of payable quality.

The Horcajo mines, which have been worked by the present company since 1882, are especially worthy of a visit. The property consists of some 20 mines, covering an area of 11,129,602 sq. meters, and gives employment to over 1000 hands. There is a considerable quantity of water to be contended with, but there are 33 engines at the mines with a united force of 2249 horse power, and the pumps are capable of raising 7000 liters of water per minute from a depth of 600 meters. The lodes run from east to west and dip north; the ore is a fine-grained galena and is associated with a gangue of quartz and calcite; it is practically free from blende, pyrites, or barytes, and is consequently very easy to concentrate. Some of the workings at these mines are already at a depth of over 330 meters.

The California Manchega mines are situated in the Alcudia Valley in the quinto La Beradilla. These mines are said to have been worked by the Romans, and many interesting indications of their ancient workings have been discovered at a depth of over 120 meters in the Beradilla mines, and at a less depth in many of the other mines in the Alcudia Valley. The geological formation in and around the Alcudia Valley (as in almost the entire Ciudad Real district) is Silurian, intersected by numerous metalliferous veins extending from Mestanza to the most northeasterly part of the valley, some of which have been partially worked, but the large majority are virgin. The chief lode in the Beradilla district is that known as the Romana, which forms the principal basis of the workings; from this lode there are many ramifications from which mineral is being cut in paying quantity. At the Romana mine work is at present being carried on at a depth of 250 meters. The ore from this district is of unusually pure quality; assays made of different grades show the following results:

	Lead. Per Cent.	Silver. Grams per Ton.		Lead. Per Cent.	Silver. Grams per Ton.
First quality lump	84.30	380	Rubble	80.80	260
Second quality lump	81.11	250	Small	79.85	250

Although these mines last year produced only some 30,000 tons of ore, nevertheless a short description of them is given, since it is quite probable that with improved prices and better means of transport to seaboard the Ciudad Real district may shortly figure very importantly in the production of lead in Spain.

Recent Improvements in Lead Metallurgy.

By H. O. Hofman.

The following pages contain a brief review of what has appeared during the past year in the leading technical journals upon the subject of Lead Metallurgy, and some additional facts which have been gathered in a recent visit to the leading silver-lead works, showing the changes that have been made in the treatment of silver-lead ores. This review forms the continuation of similar summaries that appeared in preceding volumes of The Mineral Industry. The headings under which the collected material has been classified are those of the writer's *Metallurgy of Lead;* the abbreviations used throughout this article for titles of periodicals are here subjoined:

B. & H. Ztg., Berg- und Hüttenmännische Zeitung.
Ch. Ztg., Chemiker Zeitung.
Comp. Rend., Comptes Rendus Mensuels à l'Académie des Sciences, Paris.
Dingl. Pol. J., Dingler's Polytechnisches Journal.
E. & M. J., Engineering and Mining Journal, New York.
Freib. J., Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen.
J. Am. Chem. Soc., Journal of the American Chemical Society.
J. Chem. Soc., Journal of the Chemical Society, London.
M. I., Mineral Industry.
S. M. Q., Columbia School of Mines Quarterly.
Trans. A. I. M. E., Transactions of the American Institute of Mining Engineers.
Z. Angew. Ch., Zeitschrift für Angewandte Chemie.
Z. Anorg. Ch., Zeitschrift für Anorganische Chemie.

INTRODUCTORY.

Properties of Lead and of Some of its Compounds.—Roessler* found by experiment that lead sulphide, PbS, dissolved in metallic lead crystallizes on cooling in cubes arranged in step form; they resemble the crystals formed in the blast furnace. Lead selenide, PbSe, shows a behavior similar to that of the sulphide. Cuprous sulphide, Cu_2S, taking up a small amount of lead, separates out in the form of small octahedra, while the natural cuprous sulphide crystallizes in the rhombic system. Cuprous selenide, Cu_2Se, forms a reddish-yellow and steel-blue crystalline aggregate of octahedra and copper-colored feathery masses, showing rhombic crystalline planes.

According to Junge,† users of sheet lead and lead pipe frequently demand that they shall be rolled or drawn from lead desilverized by the Pattinson rather than by the Parkes process, saying that the former lead contains less zinc, although the very reverse is more likely to be the case.

High-grade market leads contain from 0.0002 to 0.0005% zinc, the higher figure referring oftener to Pattinson than to Parkes lead. Steam Pattinson plants have been erected in England to meet this demand for lead that has not been desilverized by means of zinc. In the United States, Missouri non-argentiferous lead is often preferred for acid chambers (chemical lead) to the lead produced by refiners using the zinc process. The unreasonableness of this demand is shown by the following analyses of six leading brands of lead desilverized by the Parkes process, and of two by the Pattinson process:

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* *Z. Anorg. Ch.*, 1895, IX., pp. 31-77.
† *Freib. J.*, 1895, p. 1; *Z. Angew. Ch.*, 1895, p. 702; *B. & H. Ztg.*, 1896, p. 33.

Locality.	Process.	Sb.	Cu.	Fe.	Zn.	Ag.	Bi.
England	Parkes	0.0042	0.0009	0.0014	0.0002	0.0005	0.0052
Italy	Parkes	0.0020	0.0010	0.0028	0.0006	0.0005	0.0009
Harz Mountains	Parkes	0.0063	0.0007	0.0005	0.0007	0.0005	0.0024
Rhenish Provinces	Parkes	0.0077	0.0005	0.0017	0.0005	0.0004	0.0013
Rhenish Provinces	Parkes	0.0045	0.0002	0.0003	0.0002	0.0005	0.0172
Rhenish Provinces	Parkes	0.0037	0.0007	0.0002	0.0002	0.0005	0.0040
Mulden Works, Freiberg	Pattinson	0.0005	0.0009	0.0003	0.0003	0.0007	0.0530
Halsbrücken Works, Freiberg	Pattinson	0.0007	0.0795	0.0003	0.0004	0.0007	0.0318

The Parkes lead, in addition to containing less zinc, is likely to run lower in silver and copper, but, unfortunately, bismuth is concentrated in it, while in the Pattinson lead it may be, at least in part, collected in the enriched liquid lead, and this the more easily when the lead is richer in silver. Thus, base bullion with 145.83 ozs. silver per ton and 0.06% bismuth gave rich lead with 583.32 ozs. silver per ton and 0.18% bismuth, and market lead with 0.44 oz. silver per ton and 0.02% bismuth. When the base bullion was richer in bismuth, the silver remaining the same, the rich lead retained more bismuth (0.29%), and this was also the case with the market lead (0.054%). The decrease of bismuth in a series of crystallizations carried out with the method by thirds is shown in the following table, the figures indicating the percentages of bismuth:

Number of kettle..	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.	14.
Lead before crystallizing	0.242	0.205	0.200	0.203	0.183	0.173	0.162	0.137	0.132	0.110	0.105	0.088	0.064	0.063
Resulting crystals	0.235	0.207	0.204	0.208	0.176	0.158	0.157	0.128	0.115	0.112	0.085	0.074	0.070	a0.054
Resulting liquid lead	0.290	0.247	0.243	0.242	0.181	0.217	0.212	0.183	0.174	0.140	0.130	0.101	0.092	0.065

(a) Market lead.

The effect of bismuth on the mechanical properties of lead and on its resistance to acids is not, however, harmful. Thus, lead alloyed with 2% bismuth could without difficulty be rolled into a sheet 10 ft. long, 6 ft. wide, and $\frac{1}{64}$ in. thick; strips of lead with 0.2% bismuth, being exposed to 60° B. sulphuric acid at the chamber temperature, showed no sensible change. Such amounts of bismuth as were used in the experiments are not found in market leads, and as they showed no harmful effect on the mechanical properties and the resisting power to sulphuric acid, the bismuth actually present can be safely neglected. In the sulphuric acid department of the Freiberg (Saxony) smelting and refining works, the Pattinson and Parkes leads produced there were used for a number of years in the same chambers and showed no difference in behavior. A smooth or rough surface seems to make more difference in the matter of resistance to corrosion than a slight variation in the chemical composition, the rough surface being more readily attacked. The probable reason is that condensed droplets do not adhere readily on a smooth surface, while on a rough surface they collect, and nitrous fumes begin to eat into the lead. Near a smooth, unattacked surface will be found a rough one with deep pitholes. Therefore users of sheet lead often stipulate for smooth surfaces in contracts for sheet lead. It has been observed that in concentrating pans Parkes lead is more readily attacked than Pattinson lead. The cause seems to be its smaller percentage of copper, which makes lead more readily fusible, and lead heated to near its melting point is more easily attacked by concentrated acid than when at a lower temperature. A fact that

still lacks a satisfactory explanation is that lead-lined vats used for storing 60° B. sulphuric acid of 40° C. are often eaten through, here and there, in a comparatively short time, while the rest of the lead remains unattacked.

As to the use of lead for corroding purposes, very small quantities of foreign matter have a decidedly deleterious effect. From 0.003 to 0.005% bismuth is harmful, although its hydroxide and carbonate are white; the amount of copper should not exceed 0.003% and that of silver 0.17 oz. per ton. The presence of 0.05% antimony retards corrosion, but has no effect on the color of the white lead, though 0.1% has a decidedly bad influence. It is believed by some that Parkes lead can be improved upon as to its silver and copper contents by additional pattinsonizing, and some English works treat in this way the refined Parkes lead from Mechernich and Tarnowitz, which are the highest-grade brands of Prussia. Tests made on a large scale showed the unreasonableness of this idea. Parkes lead before pattinsonizing contained 0.00027% copper and 0.022% antimony; after the operation it showed 0.00027% copper and 0.0005% antimony, a decrease in antimony; yet no difference in the white lead produced from the two kinds of lead could be distinguished. Pattinsonizing Parkes lead may be necessary in some cases in order to reduce the amount of bismuth present, so that it may come within the proper limit.

C. Richter's book, *Das Löthen des Bleies*, Leipsic, 1896, will be welcomed by the German-reading metallurgist having to work with solutions, acids, etc., as it forms a complete treatise on a subject that generally has to be learned by practice, and with much difficulty. See, also, in this connection, the article by J. E. Rothwell, on "Lead Burning," in THE MINERAL INDUSTRY, Vol. IV., p. 735.

Ternier* states that litharge forms rhombic as well as tetragonal crystals, and Le Chatelier finds that crystals of both systems are formed at the same time.

Sampling.—Brunton† discusses the theory and practice of ore sampling, his aim being to determine in the sampling of silver and gold ores the fineness to which an ore must be crushed before it may be cut down and still represent a true average of the whole. He begins by calculating the areas of screen openings from 120-mesh to 4-in. spaces, then takes a cube of corresponding size for each sieve and from this determines the weight of five leading minerals—pyrite, argentite, galena, silver, and gold. Starting with 0.1 assay ton equaling 0.0064 lbs. of 100-mesh stuff, each grain (or cube) of which is calculated to have a volume of 0.0000001664 cu. in., he states that the ratio between the size of grain (or cube) and weight of sample must be kept constant, if the sample is to represent a true average of the whole. In order to derive a formula from which such curves can be plotted as will permit taking off the size to which an ore of given tenor ought to be crushed before it may be cut down to a certain weight, the required maximum weight of each of the five minerals for the different sizes is calculated, as well as its volume, and the final values, corrected by certain experimental data. The final formula is:

$$D = 0.45 \times \frac{1}{\sqrt[3]{f.\,s.}} \times \sqrt[3]{W} \times \frac{1}{\sqrt[3]{\frac{k}{c}} - 1}$$

* *B. & H. Ztg.*, 1895, p. 452; *Ch. Ztg.*, 1895, No. 25, p. 280.
† *Trans. A. I. M. E.*, XXV., p. 826.

W equals weight of sample in pounds; k equals grade of richest mineral in ounces per ton; c equals average grade of ore in ounces per ton; s equals specific gravity of richest mineral; f equals a factor expressing the ratio of the actual weight of richest mineral in largest particles which will pass a screen of given size to the weight of the largest cubes of richest mineral which will pass the same screen; and D the diameter, in inches, of the holes in screen, or the nominal diameter to which the ore is crushed. In plotting the curves, D gives the ordinate, the ratio $\frac{k}{c}$ the abscissa, s and W are constants, and a value for f is assumed. A comparison of the sizes demanded by Brunton with those laid down by Reed* shows that the former insists on very much finer crushing.

Bridgman† constructed a simple, cheap assay-office sampler, which is a modification of his large machine. The divider makes 100 revolutions per minute and cuts a sample in two, the largest suitable ore being $\frac{1}{2}$ in.

Topham patented‡ an ore sampler which consists of a stationary inclined chute and a moving inclined spout fixed to the side of a truncated cone which is attached to a revolving vertical shaft. The ore fed through the stationary chute is discharged upon the revolving truncated cone, and as sample one part is taken off at intervals by the moving spout and carried downward beneath the platform which receives the rejected ore from the surface of the cone.

Hallett patented§ a mechanical ore sampler which consists of a four-sided hopper with a square discharge pipe beneath which is moved to and fro a table having side-flanges. A median line through the table forms the base of a triangle; the other two sides extending downward are attached at their meeting point to a horizontal shaft. The hopper has a pipe in the center through which the ore passes. The table by its oscillating movement divides the ore into two parts, delivering them alternately into the pockets on each side.

Two new samplers which are giving great satisfaction in Colorado remain to be mentioned—those of Vezin and of Brunton. The Vezin sampler consists of two hollow truncated cones joined at the base and suspended from a vertical shaft. The upper cone has a spout extending outward which with every revolution of the shaft passes under the stream of ore and cuts out the whole of it as a sample, while the rejected ore glides over the cone into a separate receiver. The sampling device of Brunton consists of a swinging quadrant divided by radial walls into three parts. It oscillates to and fro under a continuous stream of ore and takes out the sample by the central partition, discharging it to one side, while the rejected portion, taken up by the other two divisions, is deflected in an opposite direction. Both samplers are used in the Cripple Creek district, Colorado, and the satisfactory sampling of this difficult gold ore is probably the severest test to which a mechanical sampler can be put.

Assaying.—The department of refractory materials of the Freiberg (Saxony) smelting and refining works‖ has brought into the market a transportable muffle furnace somewhat resembling the one of the Battersea Crucible Company. The size of the muffle is not given, but the furnace is said to require per hour 18.7 lbs. lignite, 14.3 lbs. bituminous coal, or 10 lbs. coke. The temperature can

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* *S. M. Q.*, VI., p. 337. † *E. & M. J.*, June 6, 1896. ‡ U. S. Patent No. 563,101, June 30, 1896.
§ U. S. Patent No. 566,169, August 18, 1896. ‖ *B. & H. Ztg.*, 1896, p. 290.

be raised to the melting point of Seger cone No. 6 or 7, which corresponds, according to the recent researches by Hecht, to 1250 and 1270° C.

Miller and Fulton* have made a contribution toward the solution of the perplexing question of the assay of rich silver ores. They determined the assay value of a carbonate ore containing over 2200 ozs. silver and 0.28 oz. gold per ton and 28% lead; and of a sulphide ore, an argentite, with over 1700 ozs. silver and 241 ozs. gold per ton. Both crucible and scorification methods were used and the loss in slag and cupel accurately ascertained, first by assaying them and then mixing the respective second slags and cupels and reassaying these. The crucible results showed that (1) an excess of litharge but very slightly increases the silver in the slag; (2) the silver collected in a given lead button is independent of the amount of charge; (3) the silver obtained from a lead button increases up to 28 gms., when it again decreases, as the loss by volatilization in scorifying or cupelling larger buttons is greater than that caused by obtaining a smaller-sized button in the crucible; (4) the amount of silver absorbed by the cupel increases with the size of the button, and more silver is recovered from the cupel than from the crucible slag; (5) the highest uncorrected crucible assay gives the highest corrected assay.

The scorification results show that (1) 30 gms. lead are not sufficient to extract all the silver from 0.1 assay ton of ore; (2) the temperature has to be carefully regulated; (3) the silver absorbed by the cupel increases with the size of the button as with the crucible assay; (4) the silver recovered from the scorification slag shows no regularity; (5) the loss in silver in scorifying is very great.

The following table gives a summary of average percentages of losses in silver in ores Nos. 1 and 2:

ORE NO. 1—SILVER.

In	Average Loss in Slag.	Average Loss in Cupel.	Average Loss—First Correction.	Average Loss—. Second Correction.	Average Loss—Total Correction.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Scorification.....	2.804	1.398	4.202	0.540	4.742
Crucible.....	0.927	1.109	2.036	0.252	2.288

ORE NO. 2—SILVER.

Scorification.....	0.7255	0.8635	1.589	0.237	1.826
Crucible.....	0.1435	0.7120	0.855	0.070	0.926

ORE NO. 2—GOLD.

Scorification.....	.....	.....	0.310	.....	.....
Crucible.....	0.0515	0.0725	0.124	.....	.....

In comparing the two methods the authors conclude that (1) the crucible is preferable to the scorifier for gold ores and for impure silver ores, especially with basic gangue; (2) the uncorrected scorification assays for silver with impure ores give higher results than the uncorrected crucible assay; (3) the corrected crucible assay gives usually slightly higher results than the corrected scorification

*S. M. Q., XVII., p. 160.

assay. They review the data obtained by Mason and Bowman,* Stetefeldt,† Dewey,‡ Furman,§ and Rose.‖

Godshall¶ made an extended series of experiments to determine the loss of silver in cupelling. He used pure lead and silver and worked as in commercial assaying, where no corrections are made for absorption by the cupel and for volatilization. The points investigated were the effects of variable quantities of silver, of temperature, of weight of lead button, and of nature of cupel. The following table is a summary of the results, covering the first three points, the figures representing the loss expressed in percentages:

APPROXIMATE AMOUNT OF SILVER IN MILLIGRAMMES AND EXACT AMOUNT OF LEAD IN ASSAY-TONS

Position of Cupel in Muffle.	2 Milligrammes.					5 Milligrammes.					10 Milligrammes.					20 Milligrammes.				
	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.
Front	3.66	4.40	5.52	5.96	4.89	3.29	2.63	3.83	4.31	3.51	3.73	2.89	4.47	4.26	3.84	3.42	2.34	3.59	3.10	3.09
Second	5.71	3.94	6.35	7.65	5.91	4.18	3.84	5.13	5.10	4.56	4.15	3.85	5.49	5.95	4.86	3.41	3.70	4.19	5.15	4.11
Third	4.66	6.45	7.51	9.20	6.95	3.34	4.13	5.12	6.49	4.77	2.99	3.98	5.49	5.12	4.39	2.87	4.58	4.00	4.48	3.98
Average	4.68	4.93	6.46	7.60	5.92	3.60	3.53	4.69	5.30	4.28	3.62	3.57	5.15	5.11	4.36	3.23	3.54	3.93	4.24	3.73

Position of Cupel in Muffle.	50 Milligrammes.					100 Milligrammes.					200 Milligrammes.				
	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.	¼.	½.	¾.	1.	Av.
Front	2.14	2.46	2.33	2.89	2.45	2.11	2.04	2.10	2.28	2.13	1.71	1.64	1.62	2.07	1.76
Second	2.44	2.55	3.37	3.95	3.08	1.91	2.10	2.88	3.28	2.54	1.47	1.89	2.10	2.19	1.91
Third	2.22	3.03	3.35	3.27	2.97	1.85	2.34	2.58	2.83	2.40	1.36	1.79	2.03	2.04	1.80
Average	2.27	2.68	3.02	3.37	2.83	1.96	2.16	2.52	2.80	2.36	1.51	1.77	1.92	2.10	1.82

The absorption of silver of different cupels (size of mesh not given) is shown in the subjoined table, in percentages:

Cupel from	Silver Used in Milligrammes.					Lead Used. Assay-Tons.
	2.	5.	10.	50.	200.	
Everett Works	4.57	3.33	2.62	2.40	1.59	0.25
New York Assay Office	4.19	4.32	3.44	2.61	1.74	0.25
San Francisco Mint	4.25	3.63	2.40	2.68	1.56	0.25
Average	4.25	3.76	2.82	2.56	1.63	0.25
Everett Works	5.12	4.02	2.46	2.46	1.81	0.50
New York Assay Office	4.86	3.99	2.79	2.27	1.95	0.50
San Francisco Mint	4.84	3.47	2.08	2.26	1.76	0.50
Average	4.94	3.83	2.44	2.33	1.84	0.50

Low** describes the De Haen iodide assay for copper as carried out to-day in his laboratory at Denver, Colo. The procedure is as follows: Treat ½ gm. of ore in

* *J. Am. Chem. Soc.*, XVI., p. 313; *M. I.*, III., p. 424.
† *Trans. A. I. M. E.*, XXIV., p. 537; *M. I.*, III., p. 423.
‡ *J. Am. Chem. Soc.*, XVI., p. 505; *M. I.*, III., p. 423.
§ *Trans. A. I. M. E.*, XXIV., p. 735; *M. I.*, III., p. 423.
‖ *J. Chem. Soc.*, LXIII., p. 700.
¶ *Trans. A. I. M. E.*, XXVI.
** *J. Am. Chem. Soc.*, XVIII., p. 458; *E. & M. J.*, May 23, 1896.

a 250 c.c. flask with 5 to 6 c.c. nitric acid (sp. gr. 1.42); boil nearly to dryness, add 5 c.c. strong hydrochloric acid, boil again nearly to dryness, add 5 c.c. strong sulphuric acid, heat until sulphuric acid fumes pass off, cool, add 20 c.c. cold water, boil, filter into 2½-in. beaker and wash, the filtrate to measure 50 to 60 c.c. Add to the solution 5 c.c. strong sulphuric acid and two strips of aluminum foil, 1½ in. square and $\frac{1}{16}$ in. thick, with corners alternately bent up and down for ¼ in., and boil for about 7 minutes, when all the copper will have been precipitated and will not adhere to the aluminum; should this be the case, continue to boil. Transfer the solution and most of the precipitated copper with hot water wash-bottle to original 250 c.c. flask, leaving aluminum foil and adhering copper behind; allow copper in flask to settle and decant its supernatant liquor through filter, washing from two to three times. Add to beaker containing aluminum foil 5 c.c. nitric acid (1.42 sp. gr., diluted one-half), warm until adhering copper is dissolved, and pour through filter containing precipitated copper, catching filtrate in original flask. When drained, replace flask by beaker, heat flask until all copper is dissolved, add potassium chlorate and boil a moment to oxidize all arsenic to arsenic acid; place under funnel and wash beaker contents on to filter; wash the filter and boil the filtrate again to remove all red fumes. All the copper is now contained in the flask as nitrate. Add 6 to 7 gms. zinc acetate and 15 c.c. water, or 20 c.c. cold saturated solution; boil and cool to ordinary temperature. Precipitate the copper as cuprous iodide by adding 3 gms. of potassium iodide—

$$Cu_2(C_2H_3O_2)_4 + 4\,KI = Cu_2I_2 + 4\,KC_2H_3O_2 + 2\,I$$

and titrate the iodine set free with sodium hyposulphite (19 gms. $Na_2S_2O_3 + 5aq.$ to 1 liter) solution ($2\,Na_2S_2O_3 + 2\,I = 2\,NaI + Na_2S_4O_6$) until the brown color caused by the liberated iodine becomes weak, then add cold starch liquor (0.5 gm. starch boiled in a little water and diluted with hot water to 250 c.c.), and finish the titration (1 c.c. = 0.005 gm. Cu). The results are accurate when not interfered with by silver, lead, or bismuth. Lead and bismuth, however, give yellowish iodides which mask the end point. A failure of the returning blue color to disappear upon addition of a drop or two of sodium hyposulphite shows that free nitric acid is present and that the assay is spoiled.

SMELTING OF LEAD ORES.

New Processes.—The Morosco, Shields & Bishop Fusion Company* has a plant at the Bunker Hill Mine, Amador County, Cal., for extracting precious metal from finely crushed, dead-roasted, argentiferous and auriferous ore by passing it through a bath of molten lead and recovering the lead carried off by the ore, freed from precious metal by wet concentration. Large capacity, low cost, and high extraction are claimed for the process.

Shields patented an apparatus† to desilverize argentiferous ores by passing them through molten lead.

In the Ore Hearth.—Picher patented‡ a process for working complex lead and zinc ores. He proposes to use a furnace in which he can get a reducing as well as an oxidizing action, first volatilizing the lead and then at a higher temperature the zinc, each to be collected by a separate system of screens.

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* *E. & M. J.*, May 23, 1896. † U. S. Patent No. 552,624, Jan. 7, 1896. ‡ U. S. Patent No. 554,725, Feb. 18, 1896.

Petraeus patented* a method for treating galena ores. It consists of smelting in a hearth furnace, filtering and agglomerating the fume, and then smelting it with metal slag in a cupola, and again treating the second fumes like the first.

In the Blast Furnace—Lead Slags.—An editorial† discusses the percentage of zinc in slags. The basic or singulo silicate of iron, being readily fusible, is well suited to dissolve the infusible silicate of zinc; slags not too high in lime also readily take up zinc oxide—*e.g.*, SiO_2 34%, FeO 34%, CaO 17%, ZnO 7%; a slag of the composition SiO_2 35%, FeO 38%, CaO 13%, ZnO 12%, is said to have done good work. The slag commonly made in lead smelting at the Herzog Julius and Frau Sophien Works in the Lower Harz Mountains—SiO_2 16.90%, FeO 35.05%, ZnO 19.64%, $BaSo_4$ 10.24%, Al_2O_3 6.31%, CaO 6.05%, the rest, being MgO, Pb, Cu and S, is very peculiar. It runs hot from the furnace, giving off much smoke; on cooling the surface of a slag cone bursts open into small craters from which white zinc fumes are emitted, the zinc oxide formed incrusting the craters. In smelting zinc-bearing copper ores the range of acidity of slags is much greater than with lead ores. Blast-furnace slags made in matting according to the partial pyritic principle which gave economic results had the following composition:

SiO_2.	FeO.	CaO.	ZnO.
Per Cent.	Per Cent.	Per Cent.	Per Cent.
31	42	8.5	14.5
40	38	10.5	8.0

The statement of H. A. Keller‡ is quoted, that in Leadville, Colo., in 1892–93, in smelting matte, contrary to former custom, high-lime slags were preferred to high-iron slags when much zinc was present, the 1 : 1 slag with 26% lime having been common. A reverberatory-furnace slag made at Argo., Colo., in 1892 contained SiO_2 41%, FeO 28%, MnO 7%, Al_2O_3 3%, CaO 7%, MgO 0.7%, ZnO, 9%, Pb 0.5%, Cu 0.39%. Pearce§ gives the composition of a reverberatory-furnace slag made at a later date at the same works—SiO_2 36.52%, FeO 30.68%, ZnO 20.08%, MnO 4.46%, CaO 3.57%, PbO 1.50%, Al_2O_3 1.29%—total 98.10%. The slag was olive-green and highly crystalline.

Canby‖ records some interesting facts about the smelting of zinc-lead ores (Pb 8 to 15%, Zn 25 to 30%, Ag 25 ozs. per ton) in 1891 in Thomasville, N. C., in a circular water-jacket blast furnace, 3 ft. in diameter at the tuyères, the lead being tapped from the crucible. The slags had the following composition:

	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2	27.9	30.0	26.0	27.0	24.5	26.0	27.0	26.4	30.0
FeO	33.9	29.0	33.4	31.5	29.4	32.1	26.5	22.7	24.5
CaO	14.8	14.0	14.4	19.0	24.5	19.0	24.3	24.8	22.8
ZnO	16.6	15.5	19.8	15.0	14.5	15.0	14.1	21.0	17.7

They ran well, and contained less than 1% lead and 1 oz. silver per ton, very

* U. S. Patent No. 556,691, March 17, 1896. † *E. & M. J.*, August 29, 1896. ‡ *Trans. A. I. M. E.*, XXI., p. 74.
§ *E. & M. J.*, Sept. 12, 1896. ‖ *Ibid.*, Sept. 26, 1896.

little matte being made. The percentage of coke used ranged from 14 to 20%; sometimes fluorspar was used, which aided in carrying zinc into the slag.

Influence of Foreign Matter.—The Bartlett process of treating zinc-lead ores is described fully elsewhere in this volume. As an example of the kind of ore treated at the works of the American Zinc Lead Company, Cañon City, Colo., and the prices paid, Bartlett* gives the following data:

Zinc, 26.8%, allowed for	Nothing.
Lead, 18%, at 30c	$5.40
Silver, 21.2 ozs. at 65c	13.78
Gold, 0.05 oz. at $19	0.95
	$20.13
Less treatment charge	5.25
Net, per ton	$14.88

The loss in silver is said to be from 1 to 2 ozs. per ton. All the lead and 90% of the zinc are saved in the form of pigment.

The Angell-Ellershausen process for treating zinc-lead sulphides is described in the *Engineering and Mining Journal,* July 4, 1896, but the details are not clear.

Biewend† proposes to smelt blende in a blast furnace, decomposing the sulphide with iron; the slag to be made is a lime-alumina silicate similar to that of the iron blast furnace; sufficient fuel is to be used to reduce all carbon dioxide to carbon monoxide, and the zinc vapor is to be withdrawn some distance below the feed hopper and condensed in a reservoir filled with glowing fuel.

David and Le Doux patented‡ a process for smelting lead ore with salt cake and burnt pyrite.

The record of this subject would be incomplete without mentioning the paper on the manufacture of zinc vitriol in the Harz Mountains, by Bruno Kerl, which appeared in THE MINERAL INDUSTRY, Vol. IV., p. 83.

Roasting of Ores.—Brown§ gives an illustrated outline description of the three modifications of his single-hearth mechanical reverberatory roasting furnaces, the Horseshoe, the Elliptical, and the Parallel. Their general character is that the hearth has on either side a slotted wall which separates it from passages through which pass the carriages bearing stirrers, the main walls of the furnace forming the outer sides of the passages. The carriages are attached to a steel wire rope running on one side in grooved wheels. The main objection made to the original construction of the furnace that the slotted walls prevented free access to the hearth, to remove accretions or make repairs, have been overcome by placing on the heavy cast-iron door frames, which are 4 ft. apart, channel beams, held in place by duckstaves, and using them as skewbacks for the arch. The spaces between the door frames are bricked up lightly, so that when it is necessary to have access to the hearth they can be easily removed. By taking out the tiles forming the lower part of the slotted wall, an opening sufficiently large to do satisfactory work is made without doing any damage. The Horseshoe furnace is circular and occupies about four-fifths of the annular space; the other

* *E. & M. J.*, May 23; June 20; August 22; Sept. 22, 1896; see *Ibid.*, June 20 (Longmaid and Collins).
† *Z. Angew. Ch.*, 1895, p. 358. ‡ U. S. Patent No. 559,935, May 12, 1896. § *E. & M. J.*, July 4, 1896.

fifth (30 ft. long) is an open space in which one carriage cools while the other travels over the hearth. The Elliptical furnace consists of two straight-line furnaces joined at one end by a semicircular furnace, at the other by an open hearth which forms the cooling floor; the wire rope is on the inner side and is set in motion by an engine placed in the large space inclosed by the furnace. The ore on leaving the furnace is slowly moved over the cooling hearth by the stirrers and discharged into a trough having a screw conveyor. The Parallel furnace is a straight-line furnace; the ore leaving it is carried over a curved open hearth to a cooling hearth placed close to and parallel with the roasting hearth. The cooling area is about one-half that of the roasting area, and the wire rope runs along the outer wall of the furnace. Data are given showing the work the furnaces are doing in oxidizing blende, lead-bearing copper matte, and silicious gold ores, and in chloridizing silver ores. A paper covering the same ground was read by Brown before the Federated Institute of Mining Engineers (England), June 5, 1896.

Hixon patented* a two-hearth roasting furnace run on the O'Hara principle, having an improved apparatus for pulling the rabble-frames over the hearths. The frames are supported by vertical arms extending through longitudinal slots in the hearth to trucks with which they are firmly connected. The vertical arms can be connected and disconnected with a moving cable or chain in the truck conduits, thus pulling along the rabble-frames or dropping them, as may be necessary.

Stockford patented† an ore-roasting furnace. It is a vertical shaft with inclined shelves down which the ore runs slowly, while the flame travels in the opposite direction. The furnace appears to be constructed for ores free from lead.

Herreshoff patented‡ an improved stirrer for the MacDougal furnaces of the Nichols Chemical Works of New York. Other patents are those of Koneman,§ Sears,‖ and Wells.¶

While mechanical furnaces have proved a great success in the roasting of copper, silver, and gold ores, and new inventions to improve the work and to reduce the wear and tear are being patented, yet in the roasting of mixed or lead ores they have failed so far to fully satisfy the demands made upon them. Most lead smelters prefer to do their work in the long-hearth, hand reverberatory furnace. Slagging roasted ore, in order to eliminate the sulphur as much as possible and obtain the roasted ore in lump form, which was very common a few years ago, is falling into disuse on account of the inevitable loss in lead and silver; even sintering is being given up, and the ore is frequently withdrawn from the furnace in a pulverulent state. This has been made possible by the great improvements made in bricking ores (see Flue Dust).

General Arrangement of a Smelting Plant.—L. S. Austin publishes** a paper entitled "A Modern Silver-Lead Smelting Plant," in which he lays down the general principles that should govern the selection of a site for a modern smelting plant, and its construction and management. A smelting plant is commonly built on a hillside; there are, however, advocates of a level site who claim that

*U. S. Patent No. 555,951, March 10, 1896.
†U. S. Patent No. 569,361, Oct. 13, 1896.
‡U. S. Patent No. 556,750, March 24, 1896.
§U. S. Patent No. 560,412, May 19, 1896.
‖U. S. Patent No. 562,158, June 16, 1896.
¶U. S. Patent No. 569,901, Oct. 20, 1896.
** *Trans. A. I. M. E.*, XXVI.

the advantages of good ventilation, accessibility, and compactness outweigh the disadvantages of being forced to elevate all the materials. As to machinery, it is advisable to duplicate all the parts, to avoid stoppages, as all delays count heavily, because the general expenses must be about the same whether the plant is working to its full capacity or not. For the same reason it is advisable to have the power plant under one roof and in charge of a chief mechanic. In the receiving of materials (ore, fluxes, and fuel) and in discharging them at the places where they are to be used into suitable bins or covered piles, great progress has been made in modern plants and the handling has been reduced as much as possible. In disposing of waste slag, the system of the Omaha & Grant Works, Denver, Colo., is highly praised. The slag-car carrying the waste slag is raised and allowed to run by gravity to a distant point, where it turns on a curve and, still descending, comes back on a track forming the edge of the dump, where it is emptied; when empty it is run further down the inclined plane to the place where it is again to be filled. The inclined elevators used to bring materials from the slag-floor to the feed-floor are considered out of place and should be supplanted by vertical elevators. The requirements to be fulfilled in mechanical sampling are enumerated as follows: Only machines taking the whole of a stream of ore are to be used, and this should be done frequently from all parts of the stream, which must have a regular flow; before subdividing the first sample it should be mixed and crushed; the machinery should be simple, strong, and easy to clean. The tendency in blast-furnace construction is to increase the height of the furnace; water jackets which absorb much heat might be replaced by water-cooled brick boshes, as in the iron blast furnace; the space in front of the furnace ought to be covered with cast-iron plates to facilitate handling of pots; rotary pressure blowers might be advantageously replaced by piston blowers; the use of hot blast is advocated, the waste heat of the blast furnaces and roasting furnaces being utilized for this purpose, and in addition the air heated by extraneous fuel.

The Blast Furnace and its Accessory Apparatus.—The leading changes which the blast furnace and its management have undergone during the last three or four years having been briefly sketched in the preceding paragraph from L. S. Austin's paper, some proposed improvements in the accessory apparatus may be mentioned here.

Huber patented* a mechanical feeding apparatus. The gases issuing from the top of the charge are carried off by a hood with swinging doors. Over the furnace rails are laid on which the car containing the charge is run. The bottom of the car consists of sections which are turned by a lever from their normal horizontal to a vertical position when the charge is to be dropped.

Williams patented† another feeding apparatus, consisting of (1) a car having the form of an inverted V, the openings of the car on either side being closed by swinging doors to be raised by a lever; (2) several charge distributors in the throat of the furnace, also shaped like an inverted V, which are placed parallel to the width of the furnace. The car with the charge having been brought by hand or machinery over the throat of the furnace, the two doors closing the latter are lowered and the charge dropped from the bottom of the car. In descending

* U. S. Patent No. 552,372, Dec. 31, 1895. † U. S. Patent No. 554,562, Feb. 11, 1896.

it strikes the distributors and is thus evenly divided over the surface of the previous charge.

The third patent is that of Honold,* who weighs the charge in his feeding apparatus. On the throat is placed a metal frame with a hopper having a drop bottom. The frame rests on the two parallel arms of the balance, the lower end being connected with the furnace by a water-joint. The charge is weighed into the hopper and when it is to be dropped the lid of the hopper is closed and the bottom turned down,

Cunningham patented† a blast furnace in which the shaft above the usual water jackets consists of upright tubes inclosed in a casing, each tube being separately connected with the main inlet and outlet pipes.

Iles‡ replaces the I-beams, commonly used to bind the shaft pillars and support the collar on which the shaft rests, by water jackets.

Nesmith patented§ an apparatus for superheating the air for blast furnaces. The air passes through an intermediary chamber into which is introduced a finely divided crude oil, which is burned by consuming part of the oxygen of the blast furnace air. Thus a heated mixture, composed of air and the products of combustion of the oil, enter the bustle-pipe of the blast furnace. It seems doubtful if this arrangement will prove of any advantage, since, if the air is to be superheated by extraneous fuel, it might better be done in such a way as not to weaken it by diluting with products of combustion; and, besides, by blowing carbon dioxide upon incandescent coke the temperature of the furnace will be reduced.

Furnace Products—Base Bullion.—The sampling of base bullion by punching and sawing has in some instances been superseded by melting down the impure blast-furnace lead in a kettle, drossing it carefully, syphoning it in the same manner as in molding refined lead, and taking a running sample for every four or five bars. In this way the difficulty of imperfect sampling of drossy bullion is avoided and the dross returned to the blast furnace of the ore-smelter, where it belongs, and not shipped to the refiner.

Speiss.—Speiss, which used to be such a bugbear to smelters treating arsenical ores, has lost its serious aspect, as smelters have now learned by regulating their charge to avoid making it. The arsenic that is not volatilized is forced to enter the base bullion, which is a disadvantage to the refiner.

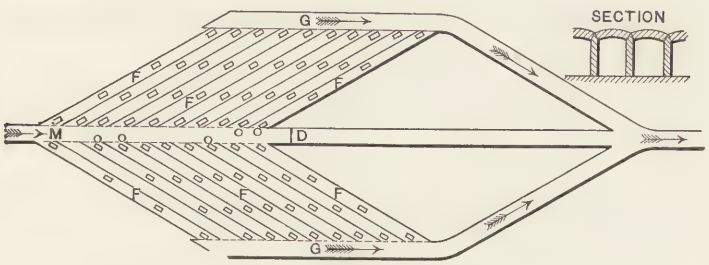
Matte and Slag.—Braden‖ describes in an illustrated paper the different methods in use at some of the leading copper and lead-smelting works for separating and handling slag and matte. Various forehearths and pots for collecting matte as well as matte molds are described and illustrated, and the granulating of matte previous to roasting is discussed. The disposal of slag by pouring over the edge of the dump and by granulating receives the consideration it deserves, as well as the returning of foul slag to the feed floor. The paper closes with a short notice of the manufacture of slag brick from blast-furnace slag near Santiago, Chile.

Field & Goetzmann¶ give an illustrated description of their elevator for granulated slag, in which the crossbar carrying the buckets extends through a

* *Dingl. Pol. J.*, 1895, CCXCVII., p. 229.
† U. S. Patent No. 558,680, March 3, 1896.
‡ U. S. Patent No. 557,127, March 31, 1896.
§ U. S. Patent No. 555,731, March 3, 1896.
‖ *Trans. A. I. M. E.*, XXVI.
¶ *E. & M. J.*, Jan. 18, 1896.

divided legging to the elevator chains, which thus do not come in contact with the strong cutting mixture of granulated slag and water, and last much longer than under ordinary circumstances.

Flue Dust.—Iles* critically reviews the different methods for collecting dust and fume produced in metallurgical operations. Long flues and large chambers have been generally considered to form a most effective means for collecting dust and fume. While they are satisfactory for dust, they do not successfully settle out fume. This collects especially on the roof and floor, but the velocity of the current passing through is too great to allow much fume to settle out. Of course, the cooler the flue (of sheet iron or of hollow terra-cotta tiles) the greater the surface exposed to the current of the gases, and the more tortuous the passages the more thorough will be the condensation, but in the end a large percentage of fume will be lost. The Freudenberg plates, which have proved so successful in European works, have in many cases been abandoned here, because of the insufficient length of the dust chamber and the lack of draught. Iles strongly advocates the system of Cowper, who retards the current of gas by placing at an angle



on either side of a main flue *M* (closed by door *D*) a number of parallel flues *FF*, ending in two flues *G*, which combine again into one gathering flue, forming the continuation of the original flue *M*. The entrances into the lateral flues are small, so that the gases after passing through them expand in the flues proper. Thus the air-current is divided into twenty parts and the velocity retarded. In the branch flues, which have a much larger section than the entrances into them, the current expands again, which retards the velocity a second time.†

Static electricity at first promised to give a successful means of clearing a dust and fume-laden atmosphere, but it was soon found that the fume could be settled out only in a quiet atmosphere and not in a moving air-current.

In order to condense fume, brushwood, coke, and screens have been placed in flues, chambers, and towers, but this has always been a failure. On the other hand, filtering through suspended bags (as practiced at the works of the Picher Lead Company of Joplin, Mo., the American Zinc Lead Company of Cañon City, Colo., and the Globe Smelting and Refining Company of Denver, Colo.) or through cloths stretched vertically zigzag over horizontal poles (as practiced at the works of the Omaha & Grant Smelting and Refining Company, Omaha, Neb.) has proved a great success.

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* *S. M. Q.*, XVII., p. 97.

† By referring to *Berg- und Hüttenmännische Zeitung*, 1883, p. 535, it will be seen that Freudenberg patented something similar after his first patents for suspended plates had been issued.

Steam has been repeatedly introduced into the gas-current, but with little good effect and with many bad results, as it helped to convert the sulphur dioxide of furnace gases into sulphurous and sulphuric acids, both of which readily attack metal, mortar, and brick. In wet condensation the projection of fine sprays of water into or across the current of gases was once advocated by Iles, but given up again on account of the corrosive effect. The spraying of water at the works of the St. Louis Smelting and Refining Company, Cheltenham, Mo., was very effective, although somewhat complicated. The chamber, 10 by 20 ft. and 12 ft. high, had two vertical partition walls dividing it into three compartments. The two larger ones were filled with 4-in. drain tiles, placed one upon another, and received a spray of water from the roof. The gases were made to ascend in the smallest one after having deposited their dust in a large flue, the necessary draught being furnished by a 90-in. Sturtevant fan. On arriving near the top of the chamber they descended first through one of the large chambers, then ascended again in the other and passed through a long wooden box which was divided by a number of vertical partitions extending from the roof to near the water, which covered the floor. In the middle of the chambers thus formed were water-sprays. The gases being forced to travel close to the surface of the cooling water, about all the fumes were condensed.

Another method of wet condensation, viz., sucking the fumes through water, has always been a failure, the main difficulty being to thoroughly wet the fume. The late F. C. Blake replaced the water by other fluids, but with no practical results.

Williams patented* an apparatus for separating dust from fume, consisting of a sheet-iron cylinder mounted on columns. It is closed at the bottom by a conical hopper with discharging door, and has at the top an outlet for the gases. At the side, below a horizontal wire screen, is the air inlet. In the cylinder is suspended a sheet-iron funnel with a long neck. The mouth is above the wire netting partition; the neck is below and reaches to near the discharge opening of the hopper-shaped bottom of the outer cylinder. The area of the inlet pipe is equal to the annular space surrounding the top of the funnel. Dust and fume-laden gases enter below the wire partition, pass up through it, and are deflected downward before they ascend through the pipe in the roof. The coarse dust, separated out by the screen, and the finer dust dropped by the change of direction of current, are collected in the hopper of the sheet-iron cylinder and discharged at intervals.

Moffet patented† an apparatus for filtering lead fumes. It consists essentially of two parts, a chamber for the reduction of sulphur trioxide to dioxide and a straining or filtering contrivance to separate out the fume. The chamber has a fireplace for bituminous coal near the bottom; the sulphur trioxide of the furnace gases coming in contact with the products of incomplete combustion is to be reduced to less harmful sulphur dioxide. The strainer consists of a horizontal flue, closed at the bottom by two hoppers, across which are placed two frames connected by a rod and covered with perforated sheepskins, the wool facing the air current, and cleaned by quick reciprocal movements. The gases from the furnace are drawn by a fan through the reducing chamber, then forced through

* U. S. Patent No. 554,563, Feb. 11, 1896.

† U. S. Patent No. 554,945, Feb. 18, 1896.

a long pipe with hoppers attached into the filtering flue. The coarsest dust collects in the reducing chamber, the finer dust in the hoppers of the cooling and settling pipe, and the finest dust and fume in the hoppers before the filters.

Iles patented* an apparatus for the settling of flue dust and the condensing of fume, in which the dust-laden gases from a blast furnace pass from a main flue alternately through two parallel flues, receiving sprays of water in the direction of the air current, and then through a series of chambers to the stack. Dust and fume pass with the water into reservoirs, and the clarified water through filters into a flume and storage tank.

Rappelye patented† a wet fume condenser much resembling the one described in most older text-books. It consists of a chamber having a vertical division wall reaching to near the floor. The bottom is filled with water, the fumes are sucked by an inclined partially submerged Archimedian screw from one compartment through the water into the other; each compartment has water sprays in the roof.

In the bricking of flue dust great progress has been made. Flue dust alone used to be mixed with lime and molded into brick. Now fine ores, from 2 in. in size down to gold-mill concentrates, are mixed by machinery with a small percentage of lime and more or less flue dust, and molded by brick machines under considerable pressure. The machines of The Chisholm, Boyd & White Company, Chicago, Ill., are found at many works. The bricks are air-dried and added to the blast-furnace charge, either raw or after having been burned in an ordinary brick-kiln. When exposed to heat in the blast furnace, the lead appears to hold the ore particles together so that the bricks arrive very little broken up at the smelting zone of the furnace. By thus mixing and bricking with machinery, instead of by hand, the operation has become so cheap that to-day most fine material is bricked. By charging little or no fines, the smelting power of the blast furnace has been greatly increased, which more than makes up for the cost of bricking, to say nothing of the diminished loss in metal by dusting and volatilization. Another machine, which is being successfully used in making briquettes of slack coal, is said to be doing satisfactory work in bricking flue dust and fine ores at the works of the Illinois Steel Company, South Chicago, Ill. It is the one patented by the Acme Mineral Nugget Company, of Philadelphia.‡ Six men are said to be able to turn out in 10 hours 300 tons of agglomerated flue dust nuggets ready to be charged into the blast furnace.

The question of rendering harmless the sulphurous gases from roasting furnaces has not yet become of such importance in this country as in Europe.

Nissenson and Neumann§ discuss the effects of sulphur dioxide on vegetation and the means employed to render the fumes harmless. The authors, connected with the Stolberg (Prussia) smelting and refining works, find that the usual chemical tests give unsatisfactory evidence regarding the harm done by the fumes emanating from their works. Metal found in the dust collected on a plant, or in the plant itself, may be due to other causes than the fumes of the metallurgical works, and an excessive percentage of sulphur in a plant may have its origin in another source than the sulphurous gases produced by the works.

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* U. S. Patent No. 553,989, Feb 4, 1896.
† U. S. Patent, No. 557,880, April 7, 1896.
‡ *E. & M. J.*, March 14, 1896.
§ *B. & H. Ztg.*, 1896, pp. 145, 153.

DESILVERIZATION OF BASE BULLION.

Parkes' Process.—The two great advances made in the desilverization of base bullion by means of zinc are those attained by the Howard stirrer and the Howard press for liquating zinc crusts. The stirrer is a mechanical device for stirring the zinc into the lead, the kettle being covered to exclude the air. Its main advantages are that the operation takes less time, and very little zinc is oxidized, thus requiring a smaller amount of zinc, and making the liquation of the crust much more perfect. The mechanical stirrer has also made it easy to increase the size of the desilverizing kettle, whose capacity has been in some instances doubled—*i.e.*, 60 tons. The press* removes, without the application of heat, the unalloyed lead carried along with the zinc crust. It can be conveyed to and from the desilverizing kettle, and raised and lowered.

Tommasi describes† a method and apparatus for the electrolytic desilverization of argentiferous lead. The plan very much resembles that of Keith,‡ abandoned years ago, the main difference being that the cathode is a disk partly immersed in the electrolyte making from one to two revolutions per minute. The spongy lead deposited on the rotating cathode is removed by stationary brushes.

PARTING DORÉ SILVER.

To complete the record of improvements mention must be made of the paper by Titus Ulke on this subject, which appeared in THE MINERAL INDUSTRY, Vol. IV., p. 343.

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*U. S. Patent No. 563,769, July 14, 1896. †*Comp. Rend.*, CXXII., p. 1476; *E. & M. J.*, August 29, 1896.
‡Hofman: *Lead*, p. 296.

MAGNESITE.

THE occurrence, production, preparation, and uses of this mineral were fully described in Vol. II. of THE MINERAL INDUSTRY, pp. 455–463. Since that time its use has been steadily extending along the old lines, but no new applications and no new supplies have been discovered.

The total production of magnesite in the United States continues to be from Child's Valley, California, though it exists at several other places in the same State. The mineral is of good quality, but the deposits are too remote to permit shipments to the Eastern States where is the chief market for it. The Californian production has not much chance, therefore, of increasing beyond the requirements of the Pacific coast. Up to the present these comprise the material for furnace-linings in the rolling-mills at San Francisco, the manufacture of Epsom salt and magnesia which is carried on in San Francisco on a small scale, and as a bleaching agent in the manufacture of paper from wood-pulp.

The Child's Valley deposit lies about 12 miles east of Rutherford. It is a fissure-vein, striking north 40° west and dipping 55° east, which has been traced on the surface several miles. Three mines have been located upon this vein (or series of veins), namely, the Berthenia, in Soda Valley, showing a vein 3 ft. wide, upon which but little work has been done; the Snow Flake, 2½ miles from the Berthenia, showing a vein 5 ft. wide; and the Cleveland about 1 mile south of the Snow Flake.

The Berthenia and Snow Flake mines are producers, the latter being the more important. It is opened by four adits, each about 100 ft. long. The rock is calcined in an ordinary limekiln to save freight in shipment.

In Fresno County a vein of hard, white magnesite 10 ft. wide is said to exist on King's River, 4 miles below Mill Creek. In Santa Clara County a deposit on Coyote Creek, 4 miles east of Madrone, has been worked intermittently.

PRODUCTION OF MAGNESITE IN CALIFORNIA.

Year.	Crude.		Calcined.		Crude Equivalent of Calcined. Short Tons.	Total Crude.	
	Short Tons.	Value at Mine.	Short Tons.	Value at Works.		Short Tons.	Value.
1892.....	608	$4,256	397	$7,940	794	1,402	$9,814
1893.	263	1,841	440	8,800	880	1,143	8,000
1894.	450	1,600	460	6,500	920	1,370	7,864
1895.	520	2,520	840	12,180	1,680	2,200	14,700
1896.	489	2,300	789	11,135	1,578	2,067	13,435

The most important of the foreign sources of magnesite are those of Greece, where the deposits are operated by the "Société des Travaux Publics et Communaux." It owns several mines high up in the hills, about 4 miles from Krimasi, a small port on the east coast of Eubœa and 2 miles from Limne, a port on the west side. The principal lode worked near Krimasi has been proved for 420 meters in length and 20 to 30 meters in thickness (horizontally), the lode dipping at an angle of about 77°. So far it has only been worked to a depth of about 40 ft. The stone is very hard, and is mined with dynamite which is manufactured at Athens. The lode gives a general average of 88% magnesium carbonate, which is sorted up to a product containing 95% to 98% $MgCO_3$. A good quality of stone showed SiO_2, 0.45%; Fe_2O_3, 0.25%; alumina, 0.10%; carbonate of lime, 1.40%; carbonate of magnesia, 97.80%. Inferior grades carry more silica.

MANGANESE.

THE consumption of manganese in the United States apparently decreased largely in 1896. The subjoined table is, however, misleading in this particular, since it makes no account of stocks at beginning and end of the year; and a part of the abnormally high imports of 1895 were undoubtedly used in 1896. The table includes the production of all classes of manganese ore in the United States, with the exception of the argentiferous manganese-iron ore of Leadville, Colo., and Tombstone, Ariz., which is used as a flux by the lead-smelters. The production of Colorado given in the table is manganese-iron ore, assaying from 25 to 30% Mn, which is shipped to the steel works at Pueblo, Colo., and Chicago, Ill. The production of Michigan and Wisconsin is manganiferous-iron ore, assaying from 4 to 10% Mn. The production of New Jersey is franklinite residuum, which assays from 10 to 14% Mn. The production of the remaining States is chiefly pyrolusite, assaying over 40% Mn.

PRODUCTION OF MANGANESE ORE IN THE UNITED STATES. (IN LONG TONS.)

Year.	Production.											Imports.		Consumption.	
	Arkansas.	California.	Colorado	Georgia.	Michigan and Wisconsin.	New Jersey.	Tennessee.	Virginia.	West Va.	Totals.		Tons.	Value.	Tons.	Value.
										Tons.	Value.				
1892	6,708	150	4,000	2,000	147,431	32,381	.....	6,079	.....	198,749	$518,141	58,364	$830,046	257,113	$1,348,187
1893	2,000	450	4,700	500	110,648	30,602	.....	1,500	.....	150,400	347,749	67,717	860,832	218,117	1,208,581
1894	3,000	250	5,000	1,000	198,466	27,667	921	1,438	66	237,808	484,332	44,655	432,561	282,463	916,893
1895	2,500	880	7,968	2,614	112,265	42,726	.....	1,556	.....	170,500	328,107	86,111	747,910	256,620	1,076,017
1896	3,038	318	9,072	2,538	110,317	35,655	.....	1,588	.....	162,526	339,083	31,489	250,468	194,015	589,551

The price of manganese ore delivered at Bessemer, Pa., is determined by the Carnegie Steel Company, according to the following schedule, which is based on ores containing less than 8% SiO_2 and 0.10% P:

Tenor in Mn.	Price Per Unit.		Tenor in Mn.	Price Per Unit.	
Per Cent.	Fe.	Mn.	Per Cent.	Fe.	Mn.
Over 49	6c.	28c.	37 to 40	6c.	24c.
46 to 49	6c.	27c.	34 to 37	6c.	23c.
43 to 46	6c.	26c.	31 to 34	6c.	22c.
40 to 43	6c.	25c.			

For each 1% SiO_2 in excess of 8% a deduction of 15c. per ton is made, and 1c. per unit of manganese for each 0.02% P in excess of 0.1%. Settlements are based on samples dried at 212° F.

Alabama.—According to a paper recently read by Mr. W. M. Brewer before the Alabama Industrial and Scientific Society, only a few deposits of manganese ore have been so far discovered in Alabama. These are confined chiefly to Cherokee County, and have never been producers on a commercial scale. A few sample carloads were shipped from Pleasant Gap station, on the East Tennessee, Virginia & Georgia Railroad. This was mined in the vicinity of Rock Run, in Cherokee County. There are surface indications of the occurrence of deposits in sections of the same county, in close proximity to the brown iron-ore deposits; but little prospecting has been done in recent years.

Arkansas.—Attention has been attracted to the deposits in Polk County through the construction of the Kansas City, Pittsburg & Gulf Railroad into the region, and they are thought to be more extensive and of better quality than was believed at first. The production reported in 1896 was chiefly from the Cushman mines.

California.—Manganese deposits occur in Alameda County, 8 to 15 miles southeast of Livermore, in the northern part of Del Norte County; on the Mailliard Ranch, 7½ miles west of San Rafael in Marin County; in San Joaquin County, 10 miles southwest of Tracy; in Santa Clara, 3 miles south of De Forest, and 25 miles southeast of Livermore; in Sonoma County, 7 miles northwest of Cloverdale. The mines in Alameda, San Joaquin, and Sonoma counties have been producers. The Californian ores are chiefly pyrolusite, varying a good deal in manner of occurrence.

New Jersey.—The manganese production of this State is entirely franklinite residuum, recovered from the zinc oxide furnaces after the zinc has been driven off as far as economical. This residuum is smelted to spiegeleisen at three works, their combined product in the five years, 1892–96, having been 9420, 9615, 8385, 14,308, and 12,403 long tons, respectively.

Virginia.—Since the closing of the Crimora mines, in Augusta County, in 1892, the production of this State has been small.

MANGANESE IN FOREIGN COUNTRIES.

As appears in the first statistical table American consumers of manganese ore are dependent chiefly upon foreign countries for their supply. Importations are made from Russia, Greece, India, Cuba, Colombia, and Chile.

Colombia.—The deposits of Panama were described by E. J. Chibas in a paper read before the American Institute of Mining Engineers, February, 1897, of which the following is an abstract: They are situated south of Viento Frio, a village on the Caribbean Sea and about 45 miles northeast of Colon. A railway has been built from the mines to the port of Nombre de Dios for shipment of the ore.

The ore occurs as braunite, pyrolusite, and psilomelane. Its general composition is represented by the following analysis of some samples taken from La Guaca mine: Manganese peroxide, 16.98%; manganese sesquioxide, 59.30%; silica, 18.77%; iron oxide, 0.90%; phosphorus, 0.031%; alumina, 0.12%; lime, 0.98%; magnesia, 0.30%; copper oxide, 0.09%; moisture, 0.38%; combined water and loss in ignition, 2.64%; sulphuric acid, ash, 0.10%—total, 100.591%. The amount of metallic manganese is 52%. The silica is rather high, and a small

discount is made in the price of the ore for every unit in excess of 8%; but by exercising due care in selection, a considerable amount of ore has been shipped containing a much lower percentage. The amount of phosphorus is insignificant, and less than the allowable limit for a first-class ore.

The analyses by Lehman & Glaser of four sets of picked samples from the Nispero mine gave the following:

	I.	II.	III.	IV.
	Per Cent	Per Cent.	Per Cent.	Per Cent.
Metallic manganese	63.74	62.92	62.76	59.98
Iron oxide	0.28	1.13	0.98	1.19
Silica	0.25	0.39	5.42	4.71

The amount of phosphorus was below 0.05%. Ores of this class, although they have occurred in small pockets of uniform purity, are more often found associated with lower grades of ore (higher in silica), and it becomes necessary to exercise great care to prevent the mixture of the two grades. In most of these deposits the highly silicious ore can be easily recognized at sight, as the silica is visible, especially on fresh fractures, in the form of minute white specks; but when the ore is wet by rain the white specks cannot be seen, and unless a piece is broken the only means of distinguishing the grade of ore is the weight, the most silicious being considerably lighter than the least silicious. Small shades of difference cannot, of course, be distinguished in this manner.

The first cargo of 900 tons analyzed 53% Mn and 13% silica; the next one, of 2560 tons, analyzed 47.5% Mn and 20% silica. Two small cargoes that followed were also high in silica; but when the nature of the ore was better known, and the analyses of several cargoes were at hand, it became easier to set a standard, and several high-grade cargoes have been shipped since. One of 1400 tons, which arrived in Baltimore in February, 1896, is said to have been the richest cargo of manganese ore ever sold in the United States. The average of the various analyses made by A. S. McCreath and Booth, Garrett & Blair was as follows: Metallic manganese, 57.502%; silica, 4.18%; moisture, 2.734%; phosphorus, 0.054%. The next cargo, of 1800 tons, was of nearly the same grade. The average of the analyses made by the same chemists gave: Metallic manganese, 56.788%; silica, 5.82%; and moisture, 1.415%. Then followed a cargo of 2600 tons, the average analysis of which, as determined by A. S. McCreath and Lehman & Glaser, was: Metallic manganese, 54.561%; silica, 9.88%; and moisture, 5.372%.

At present no washing is necessary. The ore is shipped as it comes from the mines. But a large portion of the highly silicious ore unfit for shipment can in the future be rendered merchantable by the establishment of a proper system for crushing, washing, and jigging.

The total exports of manganese ore from Colombia are estimated as follows: From 1872 to 1875, 1500 tons; from May, 1895, to December, 1895, by the Caribbean Manganese Company, 4550 tons; during the same period by Brandon Bros. & Co., over Caribbean Manganese Company's Railroad, 1600 tons; total for the latter half of the year 1895, 6150 tons; from January, 1896, to Novem-

ber, 1896, by Caribbean Manganese Company, 9065 tons; grand total to November, 1896, 16,715 tons.

At present (December, 1896) 1500 tons are loading at Nombre de Dios upon a steamer which will leave before the end of the year. This will make the total production for 1896, to December 31, 10,500 tons, and the grand total for the whole region to the same date, 18,215 tons. Of this amount 16,715 tons have been shipped since the organization of the Caribbean Manganese Company. All the ore shipped by the Caribbean Manganese Company has been from the Nispero and the Soledad, excepting 200 tons from the Viento Frio and Carraño properties. The ore shipped by Brandon Bros. & Co. was from La Guaca mine.

The manganese ore occurs in the form of pockets imbedded in clay, resulting from the decomposition *in situ* of the original inclosing rock. This rock, in places, is only partly decomposed, and its sedimentary origin is revealed, although in some localities metamorphism seems to have turned it to red and yellow jasper. The clay has been eroded in many places, and, leaving some of the pockets uncovered, has given them the appearance of isolated boulders lying on the surface. These boulders vary from a few pounds to 500 tons. In one case, at La Guaca mine, over 1000 tons of good ore were taken out of a single boulder. The rock underlying the manganese seems to be serpentine.

Germany.—Last year 43,613 tons of manganese ore were produced from the mines in the Bonn district as compared with 39,882 tons in 1895.

India.—Manganese ore is found in large quantities, in pockets close to the surface, along the East Coast Railway, in the Vizianagram State. Another hopeful locality is Jubbulpore, and it is also found in the Bombay Presidency and in Lower Burma. The Madras Presidency is the only one exporting the ore at present, the quantity shipped in the fiscal year 1895–96 being 22,758 tons, of which 19,358 tons went to Great Britain and 3400 tons to the United States.

Macedonia.—Manganese ore of good quality is mined in Macedonia. The mines are situated near the coast, do not require expensive machinery, and are worked at good profit. The ore is exported from Stratoni, in Kassandra, Gulf of Salonica.

Russia.—The Russian manganese mines are situated in the Ourals, government of Ekaterinoslav and in Trans-Caucasia. The richest deposits in the latter territory are in the Sharopan country, government of Kutais, having been discovered in 1879. The beds, which are nearly horizontal, cover a vast area and average 7 ft. in thickness. The content of manganese in the sorted ore does not fall below 50%, while the phosphorus does not exceed 0.16 to 0.17%. The mines are connected with Poti, on the Black Sea, by a railroad 110 miles long, and the conditions for their exploitation are so favorable that in spite of many natural obstacles the export amounts to 160,000 metric tons annually. The present value of the ore is $1.20 to $1.44 per ton f. o. b. Tchiatoura. The ore is shipped principally to Great Britain, Germany, the United States, and France. In 1895 there were shipped to Great Britain, 62,178 metric tons; Germany direct, 5913 metric tons; Germany via Holland, 51,498 metric tons; France, 148 metric tons; Belgium, 1951 metric tons; United States, 42,833 metric tons.

The French consul at Tiflis reports that notwithstanding the floods that destroyed all railway communication, and entirely suspended the exportation of manganese ore from the Caucasus during the first four months of 1896, the returns for the whole year are practically the same as those of the previous one. This result was due to the accumulated stocks, and to the improvement in the railway service from Tchiatoura to the mines. The increase in freight, which always occurs at the end of the summer when the export of grain is in full activity, was felt with special intensity last year. The price, which averaged 11s. on ships for England and Germany in the spring, rose in the summer to 15s., falling again to 12s. at the end of the year.

Of the 180,000 tons of ore estimated from the preliminary returns to have been exported from Poti, about 10,000 tons were absorbed in the interior of Russia, 10,000 tons went to America, and about the same amount to France. The remainder was divided in nearly equal parts between England and Germany, the former kingdom still heading the list of purchasers. The demand at the present time is very active and the outlook for the current year is most promising. The development of the steel works in Ekaterinoslav offers an easy outlet for an increased product. The miners of Sharopan have a good deal to complain of, however, in the freight rate to Tchiatoura, which is 11 kopeks a poud, the distance being only 38 versts.

Of the other Russian manganese ore deposits only those of Ekaterinoslav deserve special mention. Manganese ore beds were discovered in 1885 and the annual production amounts now to about 80,000 tons. The ore hitherto mined has been of inferior grade, containing only about 25% manganese, but recently considerable deposits of high-grade pyrolusite ore have been developed near Nikopol. The ore is found at a depth of 6.4 to 32 meters, of an average thickness of 1.42 meters, while the maximum thickness is 2.49 meters. The average content of manganese appears to be 44%, and the deposits are stated to be favorably situated for export, being only about a mile distant from a navigable branch of the Dnjepr, by which ore can be transported to the railway at Alexandrowsk. Some business has already been done from them in supplying works in the south of Russia.

WORLD'S PRODUCTION OF MANGANESE ORE. (IN METRIC TONS.)

Year.	Austria-Hungary.	Bosnia.	Canada.	Chile.	Colombia.	Cuba.	France.	Germany.	Greece.
1891.....	5,409	8,847	231	35,017	.....	22,341	15,343	40,335	13,453
1892.....	5,868	7,944	104	50,000	.....	18,000	32,406	32,861	11,716
1893.....	5,411	7,403	193	50,000	.....	13,922	38,080	40,798	5,250
1894.....	5,055	6,588	67	(*a*)	.....	(*a*)	32,751	43,702	9,319
1895.....	4,352	(*a*)	113	(*a*)	6,025	(*a*)	(*a*)	41,327	7,250

Year.	Italy.	Japan.	New Zealand.	Russia.	South Australia	Spain.	Sweden.	United Kingdom.	United States.
1891.....	2,429	3,249	1,172	113,090	861	6,993	9,079	9,628	22,757
1892.....	1,243	5,027	529	199,195	715	16,910	7,832	6,175	52,239
1893.....	810	14,169	324	141,526	2,467	1,460	7,061	1,357	40,387
1894.....	760	(*a*)	543	157,241	176	430	3,359	1,837	40,032
1895.....	1,569	(*a*)	213	163,770	49	10,162	3,117	1,293	58,531

(*a*) Statistics not yet published.

Manganese Ore in Spain.

By Reginald W. Barrington.

Considerable animation has been displayed during the past year in the production of manganese ore, especially from the district of Huelva, which has turned out nearly 100,000 tons of ore. The mines in this neighborhood were largely worked in the early sixties, when the ore fetched a high price in the market for chemical purposes, especially for the manufacture of chloride of lime. During the last 15 years, however, the Huelva manganese mines have had a hard time, owing to the severe competition of manganese of richer quality from other countries, especially Russia, Japan, and Chile.

The revival in the Huelva manganese trade is owing to the fact that in the mines from which the peroxide of manganese was formerly produced for chemical purposes there has quite recently been discovered (that is to say, within the last two years), at a deeper level than where the peroxide is found, large quantities of carbonate of manganese, containing from 30 to 40% Mn, which finds a ready market with manufacturers of spiegeleisen.

The following table shows the export of manganese ore from Huelva during the first seven months of 1896 and the countries to which the ore was shipped:

1896.	Destination.	Metric Tons.	Metric Tons.	1896.	Destination.	Metric Tons.	Metric Tons.
January	Belgium	5,076		April	Belgium	4,212	
"	England	3,250		"	Holland	1,264	
"	Holland	1,335	9,661	"	France	1,065	6,541
February	Belgium	3,552		May	Belgium	3,716	
"	Holland	1,425		"	Holland	1,172	
"	France	1,150	6,127	"	France	150	5,038
March	Belgium	2,132		June	Belgium	2,948	
"	Holland	2,119		"	France	600	3,548
"	England	1,218		July	Belgium	4,381	
"	France	900	6,369	"	France	1,683	6,064
					Total		43,348

The export of manganese ore from Huelva in 1895 was only 38,000 tons, being less than half the production of 1896.

At the present time the only mines producing this ore are those situated within an easy distance of the railroad, but if the present demand continues there is no doubt that many others will begin, and it is anticipated that the production from the Province of Huelva during the present year will be double that of 1896. One new mine alone it is expected will yield an annual output of at least 30,000 tons.

Rich deposits of manganese ore have also been found in the Provinces of Oviedo, Asturias, and Ceruel, the ore from which districts is said to be of excellent quality, containing 50% Mn; and if the railway companies can be induced to reduce their present tariff, the ore will soon be put on the market.

MICA.

THE domestic supply of mica was derived as heretofore from India, Canada, North Carolina, and New Hampshire. There were no developments in South Dakota, where there is good mica which cannot be worked profitably on account of the high freight rates to market. The Indian mica, which is mined chiefly in Bengal and Madras, comes to New York through Calcutta and London, and is the principal supply of the American market.

The uses for mica remain the same, i.e , for stoves and electrical purposes. The former demand is supplied chiefly by the North Carolina mineral; the latter by the Canadian "amber" mica and the Indian. For electrial purposes a good deal is prepared in the form of "micanite," which is mica sheet built up from small pieces of the mineral. These can be made of any size and thickness, and also can be molded. It is, of course, sold at a lower price than sheet mica, since in its manufacture are utilized small, irregular pieces which would otherwise be useless. These pieces are united in layers by a cementing material, probably a shellac composition, the exact nature of which is kept secret, as is also the method of manufacture. Micanite is manufactured by the Mica Insulator Co. at Schenectady, N. Y. Similar products are made by other concerns, but these cannot use the name "micanite," which is protected by patent.

The mica industry at the present time is dependent upon the electrical industry, in which the more part of the production of the mineral is consumed. A small amount of the most transparent, flexible, and cleavable mica, obtainable in sheets more or less large, is used in stove and furnace doors, but this is a demand which does not vary much from year to year. In the mica for electrical purposes the chief requirement is non-conductivity, which implies a low iron content. The Indian and Canadian micas have proved best adapted to this purpose, while the North Carolina mineral is used chiefly for stoves, the sheets of inferior transparency and the spotted ones produced there being ill suited for electrical purposes.

It may be said, therefore, that the demand for mica at present fluctuates in accordance with the demand for electrical machinery and appliances, in which it is a most valuable insulator. On account of its expense many attempts have been made to find a substitute, but none satisfactory has yet appeared. And, indeed, owing to the more rigid requirements of the fire insurance companies, mica is coming to be used more and more in places where it was not formerly applied.

Micanite and the similar artificial mica sheet are probably nearly as good an insulator as the pure mineral, and large quantities of them are used, but nevertheless there are many purposes for which the pure cut mica seems to be preferred. Micanite sells for $1.75 per pound.

For the first part of 1896 the mica business was prosperous; in the latter part it was dull; but at the opening of 1897 active demand set in again and the outlook for the year is regarded as favorable.

The price of mica of given quality varies according to size of sheets. The established schedule for cut India and amber micas, selected especially for electrical use, is as follows:

Size. Inches.	Price per Lb.	Size. Inches.	Price per Lb.	Size. Inches.	Price per Lb.	Size. Inches.	Price per Lb.	Size. Inches.	Price per Lb.
1 x 4	$0.90	1¾x 5	$1.75	2¼x 7½	$5 00	3 x 3¼	$2.75	4 x 7	$9.50
1 x 4½	1.15	1¾x 5½	2.15	2¼x 8	5.50	3 x 3½	3.25	4 x 8	9.75
1 x 5	1.50	1¾x 6	2.75	2¼x 8½	6.25	3 x 3¾	3.75	4 x 9	10.00
1 x 5½	1.90	1¾x 6½	3.25	2¼x 9	6.75	3 x 4	5.00	4 x10	11.00
1 x 6	2.40	1¾x 7	4.25	2¼x 9½	7.50	3 x 4¼	5.25	4 x11	12.00
1 x 6½	3.00	1¾x 7½	4.50	2¼x10	8.00	3 x 4½	6 00	4 x12	13.00
1 x 7	4.00	1¾x 8	4.75	2¼x11	8.50	3 x 4¾	6.50	4½x 4½	7.50
1 x 7½	4.50	1¾x 8½	5.50	2¼x12	9.50	3 x 5	7.00	4½x 5	8.25
1 x 8	5.75	1¾x 9	6.00	2½x 2½	.85	3 x 5½	7.50	4½x 5½	8.75
1 x 8½	5.25	1¾x 9½	6.75	2½x 2¾	1.00	3 x 6	8.00	4½x 6	9.25
1 x 9	5.75	1¾x10	7.25	2½x 3	1.25	3 x 6½	8.25	4½x 6½	9.50
1 x10	7.00	1¾x11	8.00	2½x 3¼	1.35	3 x 7	8.50	4½x 7	9.75
1¼x 4	.90	1¾x12	9.00	2½x 3½	1.50	3 x 8	8.75	4½x 8	10.00
1¼x 4½	1.15	2 x 2	.60	2½x 3¾	1.60	3 x 9	9.00	4½x 9	10.25
1¼x 5	1.50	2 x 2¼	.60	2½x 4	1.75	3 x10	10.50	4½x10	11.25
1¼x 5½	1.90	2 x 2½	.65	2½x 4¼	2.00	3 x11	12.00	4½x11	12.00
1¼x 6	2.40	2 x 2¾	.70	2½x 4½	2.50	3 x12	12.50	4½x12	13.00
1¼x 6½	3.00	2 x 3	.75	2½x 4¾	3.00	3¼x 3¼	3.50	5 x 5	8.50
1¼x 7	4.00	2 x 3¼	.80	2½x 5	3.50	3¼x 3½	4.00	5 x 5½	9.00
1¼x 7½	4.50	2 x 3½	.90	2½x 5¼	3.60	3¼x 3¾	4.50	5 x 6	9.50
1¼x 8	4.75	2 x 3¾	1.00	2½x 5½	3.75	3¼x 4	5.25	5 x 6½	9.75
1¼x 8½	5.25	2 x 4	1.20	2½x 6	4.25	3¼x 4¼	5.50	5 x 7	10.00
1¼x 9	5.75	2 x 4¼	1.25	2½x 6½	4.50	3¼x 4½	6.00	5 x 8	10.25
1¼x10	7.00	2 x 4½	1.40	2½x 7	5.00	3¼x 4¾	6.50	5 x 9	10.50
1½x 2	.60	2 x 4¾	1.50	2½x 7½	5.25	3¼x 5	7.25	5 x10	11.00
1½x 2¼	.60	2 x 5	1.75	2½x 8	5 50	3¼x 5¼	7.50	5 x11	12.00
1½x 2½	.60	2 x 5½	2.40	2½x 8½	6.50	3¼x 5½	7.75	5 x12	13.50
1½x 2¾	.60	2 x 6	3.00	2½x 9	7.00	3¼x 6	8.50	5½x 5½	9.25
1½x 3	.60	2 x 6½	3.50	2½x 9½	7.50	3¼x 7	8.75	5½x 6	9.75
1½x 3¼	.65	2 x 7	4.50	2½x10	8.00	3¼x 8	9.00	5½x 7	10.25
1½x 3½	.70	2 x 7½	4.75	2½x11	8.75	3¼x 9	9.50	5½x 8	10.50
1½x 3¾	.75	2 x 8	5.00	2½x12	10.00	3¼x10	10.50	5½x 9	10.75
1½x 4	.90	2 x 8½	5.75	2¾x 2¾	1.25	3¼x11	12.00	5½x10	11.50
1½x 4¼	1.00	2 x 9	6.25	2¾x 3	1.75	3¼x12	12.50	5½x11	12.00
1½x 4½	1.15	2 x 9½	7.00	2¾x 3¼	2.00	3½x 3½	4.50	5½x12	13.50
1½x 5	1.50	2 x10	7.50	2¾x 3½	2.25	3½x 3¾	5.00	6 x 6	10.00
1½x 5½	1.90	2 x11	8.25	2¾x 3¾	2.50	3½x 4	6.00	6 x 6½	10.25
1½x 6	2.40	2 x12	9.25	2¾x 4	3.00	3½x 4¼	6.25	6 x 7	10.50
1½x 6½	3.00	2¼x 2¼	.60	2¾x 4¼	3.25	3½x 4½	6.50	6 x 8	10.75
1½x 7	4.00	2¼x 2½	.65	2¾x 4½	3.75	3½x 4¾	7.00	6 x 9	11.00
1½x 7½	4.50	2¼x 2¾	.70	2¾x 4¾	4.25	3½x 5	7.50	6 x10	11.50
1½x 8	4.75	2¼x 3	1.00	2¾x 5	5.00	3½x 5½	8.25	6 x11	12.00
1½x 8½	5.25	2¼x 3¼	1.15	2¾x 5¼	5.25	3½x 6	8.75	6 x12	14.00
1½x 9	5.75	2¼x 3½	1.30	2¾x 5½	5.50	3½x 6½	8.75	7 x 7	10.75
1½x 9½	6.50	2¼x 3¾	1.45	2¾x 6	6.00	3½x 7	9.00	7 x 7½	10.75
1½x10	7.00	2¼x 4	1.60	2¾x 6½	6.25	3½x 8	9.25	7 x 8	11.00
1½x11	7.75	2¼x 4¼	1.75	2¾x 7	6.75	3½x 9	9.50	7 x 9	11.25
1½x12	8.75	2¼x 4½	2.00	2¾x 7½	7.25	3½x10	10.75	7 x10	12.00
1¾x 3	.70	2¼x 4¾	2.25	2¾x 8	7.75	3½x11	12.00	7 x11	12.50
1¾x 3¼	.75	2¼x 5	2.75	2¾x 8½	8.75	3½x12	13.00	8 x 8	12.00
1¾x 3½	.85	2¼x 5¼	2.85	2¾x 9	9.00	4 x 4	7.00	8 x 9	12.50
1¾x 3¾	1.00	2¼x 5½	3.00	2¾x 9½	9.50	4 x 4½	7.25	8 x10	13.00
1¾x 4	1.10	2¼x 6	3.40	2¾x10	10.00	4 x 5	7.75		
1¾x 4¼	1.15	2¼x 6¼	3.65	2¾x11	11.00	4 x 5½	8.50		
1¾x 4½	1.25	2¼x 6½	3.90	2¾x12	11.50	4 x 6	9.00	All larger, special prices.	
1¾x 4¾	1.40	2¼x 7	4.75	3 x 3	2.50	4 x 6½	9.25		

From the above list there is a discount of 50, and 10, and 5 per cent on India mica, and 60, and 10, and 5 per cent on the amber.

Trimmed mica which will cut to assorted sizes is quoted as follows (without discount):

Width.	Length.	Indian, per Lb.	Canadian, per Lb.
4″ to 8″	8″ to 10″	$2.50	$1.75
3 " 6	6 " 9	1.75	1.25
2 " 4	5 " 8	.....	.75
2 " 3½	4 " 6	.50	.85
1 " 3	2½ " 4	.25	.18

PRODUCTION OF MICA IN THE UNITED STATES

States.	1895.						1896.					
	Ground.		Sheet.		Total.		Ground.		Sheet.		Total.	
	Lbs.	Value.	Lbs.	Value	Lbs.	Value.	Lbs.	Value.	Value	Lbs.	Lbs.	Value.
New Hampshire..	195,000	$8,521	3,200	$3,250	198,200	$11,771	300,000	$12,900	5,400	$5,780	305,400	$18,680
North Carolina ...	545,000	23,435	3,000	3,150	548,000	26,585	577,000	24,811	2,600	2,625	579,600	27,436
Totals.....	740,000	31,956	6,200	6,400	746,200	38,356	877,000	$37,711	8,000	8,405	885,000	$46,116

India.—The mica industry is confined to the Hazaribagh district, and is gradually gaining in importance. Most of these mines are situated in the Koderma Government reserved forests, and the rest are scattered over portions of Gadis-Gawan, Domchanch, Khesmi, Mushnudih-Byria, Dorunda, and Satgawan. The mica mines in the Government estates and in the Dorunda and Satawgan estates, which are under Government management, have been let out in accordance with the rules framed by the Government. The greater number of the mines situated in the private estates are worked by Messrs. Christien & Company. The total output of all mica mines worked in the district in 1895 is reported to be 8019 maunds,* against 4849 maunds in the previous year, and the total number of workmen employed in the mines 5409, against 3718 in 1894.

* The maund is equivalent to 82 lbs.

MINERAL PAINTS.

THE most important mineral pigments are embraced in the following list:

Red.—Indian red, Tuscan red, Venetian red (Fe_2O_3), true vermilion (HgS), antimony vermilion (Sb_2S_3), minium or red lead (Pb_3O_4)

Brown.—Umber (Fe_2O_3, MnO_2, etc.), Van Dyke brown (Fe_2O_3, C), manganese brown (Mn_3O_4).

White.—White lead (2 $PbCO_3$, PbH_2O_2), lead sulphate ($PbSO_4$), zinc white (ZnO), zinc sulphide (ZnS), lithophone ($BaSO_4$, ZnO, ZnS), barium sulphate ($BaSO_4$), blanc fixe ($BaSO_4$, precipitated), gypsum ($CaSO_4$), strontium white ($SrSO_4$), whiting ($CaCO_3$), magnesite ($MgCO_3$), and china clay.

Yellow and Orange.—Chrome yellow (2$PbCrO_4$), Chinese yellow (PbO, $PbCrO_4$), zinc chrome ($ZnCrO_4$), realgar (As_2S_3), cadmium yellow (CdS), yellow ochre (Fe_2O_3, Al_2O_3, SiO_2, etc.), and siennas (Fe_2O_3, H_2O, Mn_3O_4).

Green.—Chrome green (Cr_2O_3), copper green (CuA), mineral green, malachite ($CuCO_3$, $Cu(OH)_2$), cobalt green (ZnO, CoO, P_2O_5, etc.), manganese green (BaO, MnO_2, etc.), emerald green, Paris green (7 $Cu_2C_2H_3O$, 3 $CuAs_2O_4$), and Brunswick green, which is a compound of barytes, chrome yellow, and Prussian blue.

Black.—Lampblack, graphite black, slate, and numerous asphaltum and coal-tar products.

Blue.—Ultramarine (SiO_2, Al_2O_3, Na_2O, S), Prussian blue, Brunswick blue, Chinese blue ($Fe_8C_{18}N_{18}$), cobalt blue or smalts (Al_2O_3, CoO), Bremen blue (CuH_2O_2), and copper blue (CuO, CO_2, H_2O).

Miscellaneous.—Asbestos is used in making fireproof paints.

The production of many of these pigments has been included in other statements of production, *e.g.*, barium white under "barytes," vermilion under "quicksilver," and graphite, asbestos, asphaltum, etc., under the respective captions.

The pigments included in this review, of which statistics of production have not been given elsewhere, are iron oxides (commonly referred to as "metallic paints"), ochers, ultramarine, umber, sienna, and slate, Venetian red, and zinc oxide. The entire amount of zinc oxide made in the United States is derived directly from ores and is not included in the spelter report. The make of white lead, on the other hand, is obtained entirely from pig lead, which is counted in the statistics of that metal. This is, indeed, the most important channel for consumption of lead. Since the average white lead contains 86% Pb, it may be

assumed that about 82,000 tons of pig lead were consumed in white lead manufacture in 1896. Besides this a large amount is used for litharge, minium, and orange mineral, all of which are used as pigments.

The most important mineral paints made in the United States are the whites, of which white lead, zinc white, and barium sulphate rank in importance in the order mentioned. A good deal of lithophone, which consists of 67% $BaSO_4$, 8% ZnO, and 25% ZnS, is imported, but none is made here. No strontium white is made in the United States.

PRODUCTION OF MINERAL PAINTS IN THE UNITED STATES.

(In tons of 2000 lbs.)

Products.	1895.			1896.		
	Tons.	Value.		Tons.	Value.	
		Totals.	Per Ton.		Totals.	Per Ton.
Iron oxides	20,903	$235,727	$11.28	21,900	$261,152	$11.92
Ochers	15,851	147,440	9.30	17,465	169,475	9.70
Ultramarine	2,500	500,000	200.00	1,900	380,000	200.00
Umber, sienna, and slate	4,850	45,900	9.43	6,990	71,255	10.19
Venetian red	2,980	67,700	22.72	5,898	91,386	15.50
Vermilion	118	118,190	1,002.45	96	94,677	986.22
White lead	92,939	8,768,656	95.00	95,068	7,802,267	82.61
Zinc oxide	22,690	1,588,300	70.00	16,799	1,259,925	75.00
Totals	162,831	$11,471,913	$70.45	166,116	$10,130,137	60.98

The Manufacture of White Lead.—Mr. William Tatham, in the *Journal of the Franklin Institute,* reviews the well-known methods of preparing white lead, showing the disadvantages of each. He considers Bradley's patented process to be the most successful of the precipitation methods. In this a basic lead acetate solution, containing 11% of basic salt, is treated with carbonic acid at a temperature of 120° F., while flowing in a thin sheet over a series of shelves. But the essential point is that the process must be stopped when one-half of the basic lead salt has been converted to white lead. The product contains from 30 to 37% of hydrated lead oxide, and is amorphous. Mr. Tatham proposes to stir finely powdered litharge into a solution of normal lead acetate until the mixture crystallizes. By exposing the crystals to carbonic acid gas a basic carbonate is formed, while neutral lead acetate remains in solution. A solution containing 14% of normal acetate is treated with 14 lbs. of litharge to each cubic foot of the solution. Coarsely powdered litharge is used at first, but the final additions are made with litharge that has been passed through bolting cloth of 200 meshes to the square inch. The process is carried out in a steam-jacketed copper vessel provided with a stirring apparatus. After 5 or 10 minutes the mass becomes curdy, and then a stream of carbonic acid gas is passed into the vessel. The crystals melt and are in part converted into white lead, which is separated from the lead acetate solution by the use of a filter press.

Red Lead and Orange Mineral.—In the manufacture of these pigments the color is the property of first importance. The composition of red lead is approximately 26.4% PbO_2, 69.7% PbO, and 3.5% of impurities. The PbO is partly in combination with the PbO_2, and exists partly as a mere admixture. Durand

Woodman has studied the subject and thinks* the above analysis would be more correctly stated: Pb_3O_4, 75.6%; PbO, 20.5%; impurities, 3.5%.

Chrome yellow,† or neutral chromate of lead, and *chrome red,* or basic chromate of lead, always contain a certain number of adulterants, of which the principal are barium sulphate, gypsum, chalk, barium carbonate, and sulphate of lead. For some time chrome yellows of a very rich color have been found on the market which contain as much as 65% sulphate of lead. The makers say that pure chromate of lead cannot present this richness of tint, whatever be its method of manufacture.

According to Liebig, chrome yellow may be prepared by treating sulphate of lead with a warm solution of neutral chromate of potash. The yellow tint thus obtained is lighter or darker in proportion to the amount of undecomposed sulphate of lead the product contains. Two combinations of sulphate of lead with the chromate are known. The first is got by treating a solution of bichromate of potash with the necessary quantity of sulphuric acid to decompose it, then precipitating it from the mixture with a salt of lead. The second is obtained in the same way, except that the proportion of sulphuric acid is doubled. In the moist state the latter combination is much less brilliant than the former, but after drying it has an extremely fine sulphur-yellow tint.

The chromate and sulphate of lead being insoluble, it is not easy to determine rapidly if these products contain sulphate of barium and gypsum. The difficulty is still further increased when other adulterants, such as chalk and carbonate of barium, are also present.

The various methods of analysis are discussed at length in the paper from which this excerpt is taken.

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* *Journal of the American Chemical Society*, XIX., No. 4, April, 1897.

† From a paper by H. Amsel in *Zeits. für Angewandte Chemie*, and *Oil, Paint and Drug Reporter.*

NICKEL AND COBALT.

THE Lancaster Gap mine remained closed, as did also those of Oregon and Nevada, and there was no production of nickel from domestic ore in 1896, except possibly a trifling amount from Mine la Motte ore. New discoveries of nickel ore in the United States were reported on the Columbia River, in Washington, about 12 miles west of Colville, and at the headwaters of the Cowlitz River in the same State, but presumably neither of these occurrences was of any importance.

Nickel, nickel oxide, and nickel sulphide are produced in the United States from Canadian matte by the Oxford Copper Company, Bayonne, N. J., and the American Nickel Works, Camden, N. J. The American Nickel Works remains the only producer of cobalt oxide.

UNITED STATES NICKEL STATISTICS.

Year.	Production.						Imports.(b)		Exports.	
	From Canadian Ore.				From Domestic Ore.					
	Ni. in Sulphide, Oxide, etc.		Metallic.		Metallic.					
	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1892.....	2,059,697	$720,894	*nil*	.....	96,152	$57,691	4,487,890	$428,062	.....	$197,763
1893.....	2,558,836	.....	331,049	.. ..	25,893	12,429	12,427,986	386,740	....	378,577
1894. ..	(a)	(a)	(a)	(a)	*nil*	.....	9,286,733	310,581	1,235,588	247,568
1895. ...	1,503,216	300,643	1,175,445	$293,861	*nil*	... .	20,355,749	629,910	1,061,285	239,897
1896.....	2,096,890	419,378	1,600,049	400,012	*nil*	.....	102,412	32,532	2,756,604	606,833

(a) The total product of fine nickel, and nickel in sulphide, oxide, etc., was 4,223,115 lbs., valued at $818,869.
(b) Includes nickel oxide, nickel alloys, in which the nickel is the chief element of value, and all forms of nickel, except manufactures.

UNITED STATES: PRODUCTION AND IMPORTS OF COBALT OXIDE.

Year.	Production.	Imports.		Year.	Production.	Imports.	
	Pounds.	Pounds.	Value.		Pounds.	Pounds.	Value.
1891.....	7,200	35,483	$60,630	1894.....	6,763	24,020	$29,857
1892.....	7,869	32,833	60,067	1895.....	6,400	36,155	39,839
1893..... ..	8,422	28,164	41,105	1896.....	12,825	27,189	36,212

The production of nickel in other countries in 1896 was small, the Société " Le Nickel " having had its mines and works in New Caledonia closed during part of the year, which policy was adopted in view of the large stock of ore and metal on hand. A small quantity of nickel was produced in Norway, Sweden, and

Germany, and in England the new works at Birmingham, using the Mond process, have been producing about 12,000 lbs. per month, the raw material being Canadian matte.

In Canada no fine nickel or oxide is produced, the output of the mines being shipped in form of matte. The principal producers are the Canadian Copper Company, which smelts about 500 tons of ore per diem, Vivian & Co., of Swansea, and the Chicago Nickel Company.

THE WORLD'S PRODUCTION OF NICKEL, IN METRIC TONS.

Year.	New Caledonia.(a)		Canada.(b)	Norway.(c)	United States.(d)	Total.
	Production.	Exportation	Production.	Production.	Production.	
1892	8,300	.....	1,095	97	44	9,536
1893	6,913	4,561	1,807	90	12	8,822
1894	6,124	4,010	2,226	90	*nil*	8,440
1895	.....	.....	1,764	.....	*nil*	.....
1896	.....	.....	3,500	.....	*nil*	.....

(a) From *L'Annuaire de la Nouvelle-Calédonie*; the nickel contents of ore production reported there are calculated as 10%, which is probably too high. (b) Nickel contents of ore produced as reported by the Canadian Geological Survey. (c) Production of metal. (d) Production of metal from domestic ore.

PRODUCTION OF NICKEL WORKS IN FRANCE, GERMANY, AND THE UNITED STATES.

Year.	France.			Germany.			United States.(a)			Norway.
	Met. Tons.	Value.	Per M. Ton.(b)	Met. Tons.	Value.	Per M. Ton.(b)	Met. Tons.	Value.	Per Met. Ton.(b)	Met. Tons.
1892	1,244	$1,236,400	$994	1,238	$1,713,556	$1,385	978	$778,585	$796	97
1893	2,045	1,237,560	605	1,402	1,703,286	1,215	1,333	812,158	609	90
1894	1,545	1,237,600	801	997	1,154,757	1,158	1,916	818,869	427	90
1895	.....	.....	.....	1,143	1,111,074	972	1,215	594,504	489	.....
1896	.....	.....	.....	.....	.....	.....	1,677	819,390	488	.....

(a) Includes production from domestic and Canadian ores.
(b) The production and values are taken from the respective government reports, except those for the United States, which are from returns furnished to THE MINERAL INDUSTRY by the producers. The discrepancy in the unit values is due undoubtedly to the various statisticians reckoning the production at different stages of advancement. In the case of the United States, for instance, the more part of the product is turned out in the form of oxide and sulphide, of which the nickel contents are, of course, less valuable than fine nickel.

The world's production of nickel is not easy to arrive at accurately. The chief sources of supply are, of course, New Caledonia and Canada. The New Caledonia ore goes to England, France, and Germany for reduction, and the Canadian chiefly to the United States, though a small amount also goes to England. There are unfortunately no statistics of the production of metallic nickel in England. We are obliged, therefore, to fall back upon the production of ore as a basis. The Canadian Geological Survey bases its estimate of the production of nickel in that country upon the average nickel tenor of the matte produced, which figures are used in the above table. In the case of New Caledonia statement is made only of the tonnage of ore produced and exported. This ore averages from 7 to 10% Ni. In estimating the total nickel contents, we have for convenience assumed that it averages 10%, which is undoubtedly too high. Moreover, in estimating production of a metal in this manner, allowance should be made for loss in smelting, which is presumably done in the case of the Canadian figures. It is not unreasonable to believe, therefore, that the nickel production of New Caledonia as reported in the accompanying table is fully 20% too high.

The Oregon Nickel Deposits.—The nickel deposits of Oregon were described by W. L. Austin in a paper read before the Colorado Scientific Society, January 6, 1896. These deposits occur near Riddle's, in the southern part of Douglas County, which is only about 750 ft. above tide-water, 226 miles from Portland and not over 60 miles from the coast. The first discovery of ore was made in 1864, but no exploration was undertaken until 1881, and really little was done until after 1891. Since that time four companies have been operating in the field, namely, the Oregon Nickel Mines, the International Nickel Mining Company, the Anglo-American Nickel Company, and the South Oregon Nickel Mining Company.

Piney Mountain, on which the mines are located, consists of a peridotite rock, which is composed essentially of olivine and enstatite, with chromite and magnetite as accessory minerals. This rock has been given the name "saxonite." Nickel silicate ore (genthite) is found in veins intersecting this "saxonite."

The amount of work yet done on Piney Mountain is insufficient to demonstrate the commercial value of the ore deposits. Numerous open cuts and shallow shafts have been dug, but nowhere has a vertical depth of more than 50 ft. been reached. The extent of the veins is considerable, the silicious nickeliferous croppings on the summit of the mountain being in places 80 ft. wide, while nickel-bearing veins are distributed over a large area. The best showing of massive genthite is in a vein at a depth of 10 ft., where there is a breast 2 ft. wide; 30 ft. lower the vein is more silicious, carrying only 7% Ni, but the breast is 6 ft. wide.

At the present time there are about 3000 tons of ore piled up at various openings on the mountain, which is said to average 5% Ni; but as this ore has come entirely from surface workings, and has been obviously leached to a great extent, it is reasonable to expect that a better grade will be obtained at more depth from the surface. Most of the soil on the mountain assays 0.5 to 2.0% Ni. Manganese, cobalt, and copper are wholly lacking in this ore, and the absence of oxide of iron in the vein filling is also noticeable.

So far no attempt has been made to work the Oregon ores, though two or three carloads have been shipped away for experimental purposes. The low specific gravity of the nickel-magnesian silicates (2.20 to 2.58) precludes wet concentration. By a method of dry, magnetic concentration, however, an ore with 3% Ni has been concentrated by W. Q. Brown to the following product: Silica, 55.58%; alumina and ferric oxide, 6.88%; magnesia, 18.72%; nickel protoxide, 18.88%; water, 0.39%. The salvage of the nickel content of the crude ore was 85%. The metallurgy of these ores is not a difficult problem, but the geographical situation limits the choice of method to those which do not require expensive fuels or heavy chemicals, because freight rates over the only existing line of railway are excessively high. Austin considers that the best method of reduction will probably be a smelting to matte in conjunction with nickeliferous pyrrhotite, which can be obtained from other deposits in Southern Oregon, bessemerizing the matte, and refining by the alkaline-sulphide process.

The Nickel Market.—The uses for nickel have not yet become so extensive as a few years ago it was predicted they would be, and up to the present time the consumption in steel-making has not been very large. The market in 1896 was therefore very quiet, but, although the demand was light, prices were maintained

at about the same level as in 1895. At the end of the year the quotation was 33 @ 36c. per lb. for ton lots.

Early in the current year (1897), however, it was reported that the military nations of Europe had determined to use nickel-steel in ordnance construction, and to replace existing guns with new ones. The announcement of a large appropriation for this purpose by the French Government led to an extraordinary movement in the shares of "Le Nickel," which rose from 195 to 310 francs one week, and fell to 215 the next.

Literature.—To the literature of the metal the most important contributions have been "Metallurgy of Nickel," by H. Moissan and L. Ouvrard, published by Gauthier-Villers et fils, 1896, which describes only the dry processes—the wet are things of the past; the "Nickel Deposits of Oregon," by W. L. Austin, a pamphlet of 27 pp.; "Bessemerizing Nickel Matte," by H. W. Edwards, *Journal Soc. Chem. Ind.*, Feb. 1896, pp. 96–99; the "Separation of Nickel and Copper in Matte and the Recovery of the Contained Precious Metals," by Titus Ulke, *Eng. and Min. Journal*, Jan. 30, 1897, in which various electrolytic methods of nickel-winning are described. These bid fair to become as important in the metallurgy of nickel as they have already become in that of copper.

*The Hybinette-Ledoux process** depends upon the stronger affinity of sulphur for manganese and copper than for nickel. When, therefore, manganese is added to molten copper-nickel matte the nickel sulphide is reduced and metallic nickel settles to the bottom, while the mixed sulphides of copper and manganese float on top. The manganese is added in the form of binoxide, together with a suitable amount of coke. The reactions that occur are expressed thus:

$$MnO_2 + C = 2CO + Mn;\ Mn + NiS = Ni + MnS.$$

Practically not enough manganese is added to effect a complete separation, and in the first step of the process part of the nickel goes down as sulphide, nickel sulphide being insoluble in manganese sulphide. The top is then removed and the bottom retreated with manganese. The tops are treated for copper and for recovery of the manganese, which is used over again. The process has been tried by Ledoux & Co. at the works of Balbach & Co., at Newark, N. J., on a considerable scale, and results are said to have been satisfactory.

Electrolytic Refining of Nickel.†—Fine nickel is now produced electrolytically at the works of the Balbach Smelting and Refining Company, Newark, N. J., in plates $\frac{3}{8}$ in. thick. The metal is so tough and elastic that it is difficult to chisel it, and the shearing of sheets 1 mm. thick is as troublesome as the manipulation of tempered steel. The plates neither break nor crack under hammer or punch.

In the commercial electrolysis of nickel, even with anodes containing as high as 96% of nickel, the anodes readily fall to pieces, owing to their uneven solution, and a large quantity of anode scrap, often as much as 50% of the charge, is produced. This is a serious drawback to any electrolytic process, as the scrap must, of course, be remelted into anodes and retreated. The difficulty increases when matte anodes are employed. At the Balbach works, Orford anodes containing about 95% nickel, 0.55% copper, 0.75% iron, 0.25% silica, 0.45% carbon, and 3%

* U. S. Patent, No. 579,111, March 16, 1897.

† Abstract of an article by Titus Ulke in the *Eng. and Min. Jour.*, Jan. 30, 1897.

sulphur are used, and over 1000 lbs. of metallic nickel are produced daily. The metal, which is deposited on cathodes, about 20 by 30 in. in size, does not carry over 0.03% arsenic and 0.02% sulphur, and is otherwise exceptionally pure. German-silver makers, who generally work by rule-of-thumb methods, seem to prefer metallic nickel to copper-nickel alloys, and nickel-steel makers prefer the metal to nickel oxide, because of its greater convenience. The demand for metallic nickel, therefore, is rapidly increasing.

The Canadian Copper Company is now erecting a large experimental electrolytic plant at Cleveland, O., for treating its Bessemer matte. The latter averages 43.4% Cu, 40.0% Ni, 0.3% Fe, 13.8% S, with 7 ozs. of Ag, 0.1 to 0.2 oz. Au, and 0.5 oz. Pt per ton, or a total value of precious metals per ton of about $15. This matte can be cast into anodes, and dissolves readily in dilute sulphuric acid. To treat 1000 lbs. daily, it is estimated 20 baths 22 by 30 by 60 in. will be required.

The following process for treating such bessemer matte was recently proposed by Mr. Ulke: Cast anodes are placed opposite sheet-copper cathodes in tanks provided with air circulation like those used in copper refining, and then subjected to electrolytic action.

The electrolyte can be prepared by dissolving granulated bessemer matte in an excess of dilute sulphuric acid, so that the electrolyte will contain about 8% of free acid. It will be well to place ½% of ammonium sulphate in the solution to hinder the electro-deposition of the arsenic as much as possible, and a little hydrochloric acid to retain all of the silver in the slimes. Copper and nickel will dissolve at the anode, while copper deposits at the cathode, and the gold, silver, and platinum will settle to the bottom of the tank as a slime, together with the anode scrap.

1. *The Copper Deposition* will proceed regularly until the percentage of copper in the solution becomes very small as compared with the nickel present. Then the solution will be siphoned over to another tank, and the remainder of the copper thrown down by sodium sulphide, or by passing the solution through a filter of nickel sulphide. The solution of nickel sulphate is then freed from iron, after which it (1) may be treated with soda to secure nickel carbonate and eventually nickel oxide, or (2) it may be concentrated in the usual way to obtain nickel vitriol, or (3) the solution, after being made alkaline may be electrolyzed to secure fine nickel. The last method concerns us here.

2. *The Electro-deposition of the Nickel.*—For this purpose the nickel solution is kept neutral or slightly ammoniacal. Insoluble anodes, of carbon or lead, which require very strong currents to overcome polarization, may be used, or we may use soluble anodes—for example, Orford anodes containing 95% Ni and 0.5% Cu. With such soluble anodes, the nickel deposited on the sheet-nickel cathodes will be free from copper for all practical purposes until a time when the latter metal, dissolved at the anodes, accumulates in the solution to a very appreciable extent. The copper may then be removed from the solution by precipitation, as described above. However, nickel containing only a trace of copper can be used for many purposes in the trades, although nickel for making nickel-steel should be free from both copper and sulphur. With insoluble anodes, the nickel deposited will naturally be of great purity. According to H. Fontaine, 5.27 grams Ni are deposited per ampere-hour from nickel sulphate. In the electrolytic decomposition of the double sulphate of nickel and ammonia, it takes 0.89 of a horse-power hour to

decompose 1 kgm. of the electrolyte, and 2.33 horse-power hours to deposit 1 kgm. of Ni. In neutral nickel baths, with soluble anodes, the electrolyte becomes more and more alkaline, if acid be not added from time to time in order to neutralize the excess of free alkali formed. With insoluble anodes, the reverse effect has been observed, *i.e.*, the bath becomes more acid in time. Fontaine therefore suggests the use of both anodes of crude nickel and of carbon in the same bath, so that their combined effect will be to preserve the neutrality of the electrolyte.

3. *Slime Treatment.*—When sufficient precious slimes have accumulated, they may be removed from the copper electro-depositing tank, screened from anode scrap, boiled with dilute sulphuric acid and air to remove impurities, melted into doré bars and cheaply treated by the Moebius process to produce fine silver. The anode residue in the Moebius tanks may be dissolved in aqua regia, filtered from the small amount of silver chloride present, the gold precipitated from the filtered solution by ferrous sulphate, and the platinum recovered from the remaining solution by precipitation with sal ammoniac. Thus fine gold and platinum can be secured cheaply.

The Electrolytic Determination of Nickel and Copper.

By Titus Ulke.

The following method of assaying crude nickel anodes was some time ago adopted by agreement between the Orford and the Balbach companies as the basis of nickel settlements, and has not yet, to my knowledge, been described elsewhere. The methods given in the standard text-books on assaying are either not adapted to this material or are too tedious and complex for commercial use.

In the simple method described below, the absence of metals whose oxides are soluble in ammonia, zinc for instance, is taken for granted, and the little cobalt present is counted as nickel. The combined percentage of copper and nickel in the sample is ascertained by one determination, and by an independent determination the percentage of copper is arrived at. The difference between these is the percentage of nickel in the sample. To determine the combined weight of the nickel and copper weigh up two or three samples of the anode nickel (1 gm. each) and dissolve with dilute nitric acid in a covered Griffin beaker $3\frac{1}{2}$ in. in diameter and 4 in. high, using about 10 c.c. water and 5 c.c. nitric acid of 1.42 sp. gr. Digest at low heat so as to avoid violent attack; then, when nearly all is dissolved, add 5 c.c. of c. p. sulphuric acid of about 60° B. Place on cooler part of sand-bath and evaporate until white fumes of sulphuric acid come off. Cool the beaker gradually, drench its contents with cold water, and half-fill with warm water. Replace on the bath and heat until the anhydrous sulphates are dissolved. To the solution, preferably cold, add enough ammonia solution (about 25 c.c.) to redissolve the nickel oxide and bring to a quiet boiling to coagulate the ferric hydrate and any arseniate and antimoniate of iron present. The alkaline solution, after the precipitates have settled, is decanted on a 9-cm. Schleicher & Schull No. 589 filter (of porous texture); finally, the precipitate is brought on the filter and is washed, the filtrate being received in the beaker or cylinder in which the

electrolysis is to be performed. The precipitate of ferric hydrate is twice redissolved in dilute H_2SO_4 (50 c.c. acid per liter) and reprecipitated with NH_4OH, the filtrate being added to the nickel solution, two reprecipitations being sufficient when the iron present is small, *i.e.*, not over 1%. Add 25 c.c. strong ammonia to the solution containing the two or three filtrates and electrolyze.

At the Balbach Works cells in parallel are used, but no difficulty is experienced in depositing 1 gm. of nickel in one night, even when electrolytic copper determinations are made simultaneously with the nickel assays, the only precaution being that thick anode spirals are used for the nickel and thin anode spirals for the copper determinations. The strength of the dynamo current employed is regulated by means of 50 and 110-volt lamps. Good deposits of nickel of a silvery white appearance, and presumably free from other metals, excepting copper and cobalt, are obtained.

The determination of the copper is carried out as follows: The copper constituting less than 1% of the metals in the Orford anodes, 5 or 10 gms. of drillings are weighed out and treated with 25 c.c. HCl and 10 c.c. of aqua regia, containing 50% of HNO_3 in a spacious Erlenmeyer flask. Let the action begin in the cold, and finish on the sand-bath. Evaporate to almost complete dryness, and cool. When the beaker is cold enough to withstand addition of liquid, add 25 c.c. HCl, and evaporate until all the metals present have been dissolved, then add about 300 c.c. hot water, and pass H_2S into the solution. After the precipitate has settled, decant on a small filter, the filtration being accelerated by using a suction syphon. M. Liebschutz, chemist at the Balbach Works, avoids all danger of breaking the filter by protecting the tip of the filter by a small disk (2 in. in diameter) of cheese-cloth, instead of by the usual platinum cone. The filter with the precipitate is now placed in a small dish, dried, calcined slightly, digested with HNO_3, evaporated, and the nitrates calcined in order to burn and destroy the organic matter present. The residue is taken up with HNO_3 and washed into a beaker or electrolyzing cylinder. Now add 20 c.c. HNO_3, 5 c.c. H_2SO_4, and enough water to make up about 300 c.c. solution, and electrolyze in the usual way. Deduct the percentage of copper found from the combined percentage of nickel and copper as above. Results generally check very closely. Mr. Liebschutz, however, informs me that he believes the results are possibly too high by two or three-tenths of a per cent., owing probably to an occlusion of nitrogen by the nickel. The writer is inclined to think this error is due rather to the formation of a little nickel oxide on the cathode, as is shown by the loss in weight when the electro-deposited nickel is heated in a current of dry hydrogen gas.

NITRATE OF SODA.

NITRATE of soda, $NaNO_3$, which, when pure, contains 63.5% of nitrogen pentoxide, and 36.5% of soda, is found at several points in the arid regions of the Western United States, but the sole source of the world's supply of this mineral at the present time is the district of Tarapaca, in Northern Chile, and in the adjacent parts of Bolivia, where the pampas, situated at an elevation of about 3000 ft., is covered over a large area with a stratum several feet in thickness. Associated with the soda niter are gypsum, common salt, and sulphate of magnesium. These deposits are exploited almost entirely by British companies, which have constructed railways to them for the cheap transportation of the product. The latter is exported chiefly from the port of Iquique. In fact the exportation of nitrate is the main business of that place, and the welfare of its inhabitants depends on it. The production of nitrate is regulated by a combination of the proprietary companies.

NITRATE OF SODA SHIPMENTS, CONSUMPTION, STOCKS, AND PRICES FROM 1889 TO 1896. (*a*)

(In tons of 2240 lbs.)

	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.
Shipments from South American ports to all parts for 12 months ending Dec. 31	930,000	1,028,000	759,000	784,000	939,000	1,073,000	1,218,000	1,068,000
Afloat for Europe Dec. 31	380,000	514,000	321,000	215,000	358,000	393,000	506,000	347,000
Stocks in United Kingdom ports	30,000	24,000	23,000	20,000	9,100	17,300	21,500	24,000
Stocks in Continental ports Dec. 31	160,000	172,000	191,000	185,000	92,900	101,700	134,500	324,000
Consumption in United Kingdom for 12 months ending Dec. 31	105,000	119,000	120,000	117,000	100,000	114,500	117,500	105,000
Consumption in Continent for 12 months ending Dec. 31	564,000	666,000	703,000	685,000	682,000	749,500	789,500	805,000
Consumption in the United States for 12 months ending Dec. 31	80,000	104,000	100,000	100,000	104,000	100,000	110,000	115,000
Consumption in the World for 12 months ending Dec. 31	749,000	889,000	923,000	902,000	886,000	964,000	1,017,000	1,025,000
Visible supply Dec. 31 (including the quantity afloat for Europe and stocks in United Kingdom and Continent)	570,000	710,000	535,000	420,000	460,000	512,000	662,000	695,000
Price Dec. 31, per cwt.	8s. 4½d.	7s. 7½d.	9s. 1½d.	9s. 1½d.	9s.	8s. 10½d.	7s. 7½d.	7s. 10½d.

(*a*) From W. Montgomery & Co.'s Circular.

PETROLEUM.

THE production of petroleum in the United States increased enormously in 1896, especially in Ohio, New York, Pennsylvania, and West Virginia. At the close of the year it was estimated that there were in stock 10,000,000 bbls. of Pennsylvania oil, and about 22,500,000 bbls. of Indiana and Ohio oil, respectively, an increase and decrease from the figures at the beginning of the year. This is without doubt due to the fact that the Indiana-Ohio oil is displacing the Pennsylvania.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.

(Barrels of 42 gallons.)

Year.	Pennsylvania and New York.	West Virginia.	Ohio.	Indiana.	Colorado.	California.	Kentucky and Tennessee.	Other States. (a)	Total United States.
1892.....	28,422,377	3,810,086	16,362,921	698,068	824,000	385,049	6,500	135	50,509,136
1893.....	b30,541,740		16,134,485	2,232,303	730,000	600,000	9,000	1,700	50,349,228
1894.....	b30,622,336		13,891,795	2,606,505	803,000	600,000	1,000	2,700	48,527,336
1895.....	b30,406,693		18,415,631	2,386,100	530,000	1,215,000	1,500	7,019	52,961,943
1896.....	b33,455,571		23,311,853	2,566,000	500,000	1,550,470	1,500	11,000	61,396,394

(a) Illinois, Kansas, Texas, Missouri, Indian Territory, and Wyoming. (b) The crude from West Virginia and Southwestern Pennsylvania being run into common pipe lines, the production cannot be given separately for each State.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (IN GALLONS.)

(1 = 1000 in quantities and values.)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffine.		Residuum.		Totals.	
1892.....	104,013	$4,660	16,351	$1,033	586,406	$31,488	33,805	$5,071	329	$31	740,905	$42,313
1893.....	114,609	3,926	16,249	933	705,675	31,283	34,763	4,888	461	28	871,757	41,058
1894.....	114,269	4,617	14,832	904	726,727	29,799	38,975	5,137	119	10	894,922	44,463
1895.....	116,108	6,286	12,922	1,000	686,006	43,540	47,876	6,289	170	15	863,082	57,131
1896.....	118,133	6,032	13,641	1,123	758,076	49,704	51,705	6,770	521	28	942,076	63,657

In December prices, which for the greater part of the year had ranged well above $1, fell to 93c. per bbl. for Pennsylvania oil. This decline is attributed to the greater use of Ohio oil, and especially to the increasing competition of the Russian product in foreign markets. Moreover, there is now a considerable production of oil in Sumatra and Java, which is supplying some of the markets of the East.

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Yearly Average
1892.....	$0.62⅜	$0.60¼	$0.57⅛	$0.57⅞	$0.57⅜	$0.54⅛	$0.52½	$0.55	$0.54⅜	$0.51⅜	$0.52	$0.53¼	$0.55⅝
1893.....	.53½	.57⅜	.65¼	.68¾	.58¾	.60¼	.57⅝	.58⅞	.64⅝	.70¾	.73⅞	.78¼	.64
1894.....	.79¾	.80⅝	.82	.84¼	.86	.89⅜	.83⅛	.81	.83	.83	.83	.91½	.83⅞
1895.....	.99	1.04¾	1.09¾	1.79	1.74¼	1.53⅜	1.46⅝	1.26⅛	1.22⅜	1.24¼	1.48⅝	1.42	1.35⅞
1896.....	1.45¾	1.39	1.33½	1.22⅜	1.10¾	1.14¾	1.09½	1.05	1.12⅝	1.15¼	1.15½	.97⅞	1.19

American petroleum only a few years ago constituted the entire supply in the markets of India, China, and Japan. It has been to a very considerable extent replaced in those countries by Russian oil, and the trade is now being further divided by the active competition of new fields. Langkat oil now holds almost as great a share of the Chinese trade as the Russian product, and the other newly opened fields of Borneo, Sumatra, and Java are making rapid progress. Although the Japanese oil wells have not done as well as was expected, they furnish a considerable part of the home supply. It looks very much as if the far East would, in a few years, cease to be a customer for American oil, and would take its supplies entirely from the new fields which are now in course of development.

PETROLEUM IN FOREIGN COUNTRIES.

Austria-Hungary.—The exportation of illuminating oil from Austria-Hungary in 1896 was 231,062 quintals (about 150,000 bbls.), or 192,435 quintals (about 120,000 bbls.) more than in 1895. The export of crude oil increased 4000 quintals, or about 2500 bbls. Of benzine there were exported 18,283 metric tons as against 8722 metric tons in 1895.

The Austro-Hungarian Petroleum Trust has now practically collapsed; the recent trade agreement between the refineries expires May 1, 1897, and owing to the existing conditions there is little probability that it will be renewed. A number of the refineries have cut prices and encroached upon the territory and allotments of others. The question of securing an outlet for the surplus Galician oil has become a matter of prime importance. The markets most contended for are Germany, Switzerland, and Italy.

In order to promote the import of Galician oil into Hungary, the Hungarian government has made such concessions in regard to freight rates and duty that all the Hungarian refineries, even those at Fiume and Orsova, now procure their crude oil from Galicia, which, owing to the low freight rates, is much cheaper than the American or Russian. The importation of American crude oil has ceased altogether, while that of Russian has declined to one-third of the former quantity. Even the only refinery in Bosnia, at Brod, which is the nearest to the Russian oil field, consumes Galician oil exclusively.

Canada.—The Canadian oil field, of which Petrolea is the center, covers an area of 2½ by 18 miles, and has about 8000 wells in active operation, averaging 100 bbls. a year each. The well that yields a barrel a day is considered good pay. Many of them do not produce half that amount and yet return profit on the money invested. Each well costs about $500, the drilling alone amounting to $150 to $175. The total number of men engaged in the industry is 633, whose wages average $1.50 per day.

The amount of crude oil distilled by Canadian refineries in 1895 was 25,223,785 imperial gals., in addition to which 2,213,636 gals. were used for fuel, making a total of 27,437,424 gals. The products were as follows: Illuminating, 10,924,826 gals., valued at $1,237,328; lubricating, 2,400,404 gals., valued at $205,591. The proportion of illuminating oils extracted from the crude increased 4.64% from 1892 to 1895, while the proportion of lubricating oil decreased 2.84%, and that of all other oils remained very nearly the same; but the average of all oils extracted from the crude raised by 2.53%. At the same time the quality of the illuminating oils has been so much improved that they are now little, if at all, inferior to the best American. The quality of the crude remains as formerly, with its high percentage of sulphur. The Canadian production of petroleum in 1896 was 726,822 bbls. (of 42 imperial gals.), valued at $1,155,646.

Petroleum is said to have been discovered at Kisbueena Creek, East Kootenay, British Columbia, in such quantity that it is believed the industry will attain considerable importance.

Germany.—The Consolidated German Petroleum Company has completed its first well in the Luneburg forest, from which it obtained 100 bbls. a week. The well is 211 ft. deep, and cost about $170 to drill. Other wells are to be drilled.

The Alsatian Petroleum Company, of Amsterdam, has bought the wells and refinery formerly owned by the Rudolf-Biblisheim Walburg Company in Alsace, and intends to enlarge the scope of operations. This concern is the second largest oil producer in Alsace, being exceeded only by the Pechelbroun works, which have been operated very successfully since 1889. The purchase includes 30 wells, with a capacity of 300 bbls. a day.

Great Britain.—Imported, in 1896, 3,842,373 bbls. of illuminating oil, of which 2,995,096 bbls. came from the United States and 847,277 bbls. from Russia. The importation in 1896 was 173,221 bbls. more than in the previous year.

India.—The production of petroleum in Burma in 1894 was 50,612,291 liters, valued at 11,000,709 rupees; and in 1895, 59,128,517 liters, valued at 15,42,591 rupees.

Italy.—According to the French consul-general at Florence, Italy, it has been established by official exploration that the oil zone of the Apennine Mountains extends from Bologna to Florence. The first traces of oil were encountered at a depth of 475 ft., but the wells drilled so far have been poor producers. The intention is now to sink them to a depth of 1800 ft.

Java.—The production of petroleum in Java has greatly increased during the last few years, so that Javanese oil is commencing to compete with the Russian and American for the home trade. The largest producer in Java is the Dordrecht Petroleum Company, which owns a number of wells in Central and Eastern Java. It was organized in 1887, with a capital stock of 350,000 florins. In 1890 it paid a dividend of 6% and since then 15, 42, 36, and 62% per annum. The company operates two refineries, one at Surabaya, the other at Samarang, the former having a capacity of 450,000 to 500,000 gals. per month, the latter a capacity of 330,000 gals. The Surabaya refinery is able to

increase its production to 1,000,000 gals. per month. The oil is shipped in tank cars to the principal towns in the eastern regencies and there emptied into tin cans. The company has built a tin-can factory at Surabaya and a case factory at Bandjermassin, on the island of Borneo. The Samarang refinery will soon produce 600,000 gals. per month, and it is expected that the production will be increased further as soon as the Javanese oil is introduced more extensively in the regencies of Samarang, Pekalongan, Tegal, Cheriboe, and Indramayoe. The company sells its oil 30 Dutch cents per case cheaper than Russian oil can be procured, or for about 3.40 florins, including all charges. The quality of the oil is inferior to the imported.

Peru.—The production of petroleum in Peru is increasing and these fields will probably become an important source of supply. The principal is in the department of Piura, where the area of the petroleum-bearing strata has been estimated as high as 7200 sq. meters. The crude petroleum contains 84.9% carbon, 13.7% hydrogen, and 1.4% oxygen. The average depth of the wells is 800 ft., the maximum being 1200 ft. Out of 49 wells drilled since 1892 all but five have been productive. Crude oil is being used on the locomotives of all the Peruvian railways, and in numerous manufacturing establishments. Domestic consumption is now more than two-thirds supplied by domestic production.

Roumania.—The petroleum fields of Roumania are now attracting much attention. There seems to be no doubt as to their extent and value. From Turn-Severin on the western frontier of the kingdom the petroleum zone can be traced skirting the foothills of the Carpathians northeast toward Bukovina and Galicia. The width of the zone is from 10 to 15 kilometers. According to several authorities, however, petroleum occurs in the whole of the country to the Danube, which would embrace a zone 100 kilometers wide. The petroleum is found in the paludinous Pliocene and Miocene formations.

Petroleum has been worked in Roumania in more or less primitive manner for 25 years, at five principal centers, where oil has been discovered comparatively near the surface. These are:

1. Olt Valley, in the vicinity of Rimnic-Valcea.
2. Dimbovitza and Talomitza Valleys, in the vicinity of Tirgu-Vestei.
3. Prahova and Teleagen Valleys, north of Ploiesti.
4. Buzeu Valley, north of Buzeu.
5. Trotush and Tazlau Valleys, between Tirgu-Oena to the south and Bacan to the north.

So far only the first oil-bearing stratum has been worked to any extent. In the five centers there are about 50 borings and 800 hand-dug wells. The deepest successful boring is about 300 meters, and while the most of the hand-dug wells range from 60 to 120 meters none go below 200 meters. The output of these wells and borings in 1894–95 amounted to 80,000 tons. Since then it has been increasing. The borings produce 20% and the hand-dug wells 80% of the total. The production is chiefly consumed at home, only about 15% being exported. The principal operators are Mr. Hernia, at Campina, Mr. Monteorn at Monteorn, Mr. Grigorescu at Glodeni and the Roumanian Petroleum Company at Moinesti and Soluntz. These produce about 33% of

the total. At present there are 70 kilometers of pipe-line in the kingdom; the railway rolling-stock includes 111 tank-cars for the transportation of oil. In the last two years, however, numerous improvements have been made, which favor the development of the petroleum industry, and considerable foreign capital has already been enlisted in it. The bridge over the Danube at Cernavoda enables direct railway communication with Constanza, on the Black Sea, and that port is now being dredged to a depth of 8 meters. The improvements at the Iron Gates of the Danube gives an easy outlet into Hungary, Austria, and Servia. Consequently Roumanian oil producers are in a position to compete with those of Russia as soon as the exploitation of the industry is undertaken on a larger scale.

At present there are numerous small refineries in Roumania, but none of them is well equipped or scientifically operated. The products put on the market are:

Class.	Density.	Percentage of Crude Oil.	Price—Francs.
Benzine	650 to 720	15	20
Kerosene, water-white	812 to 820	32	14 to 15
Kerosene, standard	820 to 228	18	12
Lubricating oil	900 to 910	15	20 to 25
Residue	.....	15	3
Loss	.....	5	.....
Total	.....	100	.....

At the last annual meeting of the Roumanian Petroleum Company it was resolved to increase the capital stock from 2,400,000f. to 10,000,000f., and to push drilling operations in the different oil fields more actively. It was also decided to build a case factory and a petroleum dock at Constanza, on the Black Sea, and several docks at different points along the Danube River. Ten iron tank steamers of 300 to 1200 tons capacity will be available for the export trade at the opening of navigation.

Besides petroleum, ozokerite and natural gas occur in the Roumanian petroleum region. On some of the claims there are evidences of ozokerite in large quantities.

Russia.—The exports of oil from Russia in 1896 are reported as follows: Residuum and crude oil, 199,300 bbls.; lubricating oil, 726,200 bbls.; distillate, 846,700 bbls.; illuminating oils, 5,481,400 bbls., making a total of 7,253,600 bbls. Of this total Batoum shipped 5,243,500 bbls. and Novorossisk the remainder.

The production of petroleum in the Baku district showed a falling off in the first part of the year, but from July onward increased rapidly, one big gusher after another having been opened. One of the latest, drilled by Tagiev & Co., in the Bibi-Eibat field, near Baku, spurted 160 ft. in the air, producing daily about 100,000 bbls.

In 1895, 269 new wells were drilled, the average depth of all being 540 ft. In six years 1371 wells have been drilled. The average number now in operation is 622, and out of an annual production of 38,000,000 bbls., 26,000,000 are pumped, the remainder flowing freely.

Many new discoveries of petroleum were made in Russia in 1896 and much important exploration was undertaken. By order of the government, geological investigations are being made in the Province of Volhynia, on the Galician frontier, and in the districts of Kremenez and Starokonstantinov petroleum has been discovered. Naidbeko & Co. have discovered petroleum near Petrovsk, on the Caspian Sea.

The Grosni field, in the Caucasus, is to be connected by pipe-line with Poti, on the Black Sea, a distance of 200 miles. The petroleum of this field yields but 17% illuminating oil; the residuum is of 0.925 sp. gr.

Belgium capitalists have asked permission from the government to drill wells in the valley of the Petchora River, in the extreme northeast of European Russia. The existence of oil in this region has been known since 1725, but little exploration has yet been done, though what there is done is promising. The Petchora petroleum contains 55 to 60% illuminating oil, which is not far behind the Pennsylvanian (containing 75%), and nearly twice as good as the Caucasian (containing 30 to 33%). The oil sand is reached at a depth of 250 to 300 ft. against an average of 900 ft. in Caucasus. If the developments in the Petchora region prove successful the field will be a strong competitor with American and Caucasian petroleum, since crude oil can be delivered in almost any port of northwestern Europe at a lower figure.

On the coast of the Black Sea, near the port of Anapa, there is an oil field which is considered very promising, according to recent investigations. As regards location, this field is much better situated than Baku, Grosni, Petrowsk, and the other oil fields on the Caspian Sea, on account of its proximity to the Black Sea.

In the district of Dagestan, in the Caucasus, which was recently opened by the Petrowsk and Derbent Railway, oil deposits have been discovered. The new field is siutated about 8 miles from the port of Petrowsk, on the Caspian Sea, and comprises an area of about 10,000 acres. The oil found there is much lighter than the Grosni oil.

South Africa.—Rich oil deposits are reported to have been discovered in the Mdimba Range, 8 miles from Bremerdorp, in the Swazi district of the South African Republic. The oil is almost colorless and the soil so thoroughly saturated with it that it can be collected in shallow wells.

Sumatra.—The Royal Netherlands Petroleum Company completed a well at Langkat, Sumatra, which surpassed any drilled so far on that island. Its daily production is 4800 bbls. while none of the older wells produced at any time more than 500 bbls.

Scotland.—The Scotch shale oil industry has been preserved only by the invention of improved processes of production, and recovery of by-products. Some shales are rich in ammonia and paraffine wax, while others are rich in burning oils. Chemical science has greatly assisted in the more complete extraction of the different products, some of which had been treated as of little value. In the manufacture of shale oil the great object is to minimize the yield of incondensable gases, and secure the liquid and solid hydrocarbons as completely as possible; this is effected in part by fractional distillation, the temperature ranging from 600 to 800° F.

At the Clippens Works, five miles south of Edinburgh, a ton of oil shale produces on an average, dependent on the class of shale used in distillation, 25 gals. of crude oil, 2½ gals. of crude spirit, and 20 lbs. of sulphate of ammonia. About 4000 tons of shale are treated per week. These works are remarkable for the brilliant quality of the burning oil produced. The best (the sunlight oil) is shown by analyses made by Mr. J. F. King, gas examiner to the city of Edinburgh, to give 10% more light than the Russian oil, and 50% more than the American. The flash point of samples examined gave American oil, 70° F.; Russian, 84° F.; Clippens, 128° F. A summary of the various products made at these works is as follows: Sunlight oil for domestic use; naphtha, used for outside lamps and in the manufacture of linoleum; gas oil from which the gas used in railway carriages is made; lubricating oil, for machinery generally; paraffine wax, from which candles are made, about 60 tons of wax being refined every week; coke, left after distilling crude oil, utilized for yachts chiefly; and sulphate of ammonia, used as a fertilizer.

The Scottish-American paraffine combination was disrupted in January, 1896. Prices, which had previously been as follows: Hard scale, 1⅞d., soft scale 1¼ @1$\frac{5}{16}$ d., wax 120°, semi-refined, 2$\frac{7}{16}$@2⅜d., fell $\frac{3}{16}$d. per lb. on each sort. As the year progressed, however, there were increased difficulties in making sales to the candle manufacturers, who are the chief consumers of these paraffines. The Candle Association had collapsed with the other, and the market was oppressed by heavy importations of American wax; consequently the quotations on Scotch paraffines had to be lowered in September to 1$\frac{7}{16}$@1¾d. for hard scale, 1½@1⅝d. for soft scale, and 1$\frac{5}{16}$@2⅛d. for wax, 120°, semi-refined. Candles suffered in like manner, and as low as 2s. 6d. per dozen pounds was taken for the common sort, though 2s. 8d. to 2s. 9d. is ordinarily a low figure.

Burning oil, which rose to 6¾@7d. per gal. in 1895, fell to 6¼@6½d. in May, 1896, which price was confirmed in August in fixing the winter supply. Business in this branch was fairly good throughout the year. Lubricating oil, however, gained in price, closing at the end of the year at £5@£5 5s. for 865°, £5 15s.@£6 5s. for 885° and £6 10s.@£7 for 905°, a gain of 10s. per ton for the year.

The Broxburn Company paid a dividend of 7½% in the spring, the Pumpherston one of 5%, and the Oakbank one of 5%; while Young's Paraffin and Mineral Oil Company carried forward £20,000, and the Linlithgow Company also made a profit. The Holmes Company (crude oil and ammonia sulphate only) suffered a loss on the year's operation. In the autumn the Dalmeny Company paid 20%, and the Clippens had a small amount to the credit side.

THE WORLD'S PRODUCTION OF PETROLEUM. (*a*) (IN METRIC TONS.)

Year.	Austria.	Hungary.	Canada.	Germany	India.	Italy.	Japan.	Russia.	United States.
1891.....	123,006	726	105,660	15,315	20,455	1,131	7,400	4,756,417	7,978,923
1892.....	120,000	20	109,080	14,527	29,084	2,548	9,825	4,896,179	7,000,982
1893.....	122,000	14	111,700	13,974	31,079	2,652	13,276	5,790,692	6,388,318
1894.....	131,930	2,061	116,000	17,232	44,506	2,853	.....	5,191,552	6,158,119
1895.....	130,000	2,083	114,653	17,051	52,032	3.594	.....	.....	7,215,896

(*a*) From the official reports of the respective countries.

The Petroleum Fields of Wyoming.

By Wilbur C. Knight.

Petroleum was discovered in that part of the Rocky Mountain region now known as Wyoming about a quarter of a century prior to the great oil excitement in Pennsylvania. Bonneville* was the first to make the announcement, but evidently no one placed credence in his statement; economic geologists paid no attention to his discovery, and as late as 1890 the territorial geologist of Wyoming stated that the first discovery of oil had been made only about a quarter of a century before.†

The acquisition of oil territory by private location was not thought of until the advent of the Union Pacific Railroad, and then in a limited way only, on account of the hostilities of the Indians who controlled the greater portion of the oil territory. At the completion of the Union Pacific Railroad some attempt was made to develop the oil fields in Uinta County, and oil was collected in a small way and sold to the railroad for lubricating purposes. From 1869 to 1880 little or no attention was paid to oil in the State. Most of the oil springs were open for location, but the districts were not named and but few of them known. In Crook County a company collected oil and sold it to the Black Hills miners until the railroad reached the Hills and furnished Eastern lubricating oil cheaper than it could be freighted in by team from Wyoming points. The Graff Oil Company, now known as the Murphy Oil Company, commenced work on the Little Popo Agie River in 1884 and drilled three producing wells. This company freighted some crude oil to the Union Pacific Railroad, but Eastern oil soon replaced it. These wells, which have been rated at 200 bbls. each, have been packed for about 12 years, awaiting an opportunity to utilize the oil.

The oil industry of the State was placed on a paying basis in 1894, when the Pennsylvania Company at Casper completed its wells in the Salt Creek Basin, and commenced making regular shipments of petroleum. This company has gradually increased its output, and in 1896 it surpassed all previous records. Between 1886 and 1896 there were many oil companies organized in Wyoming, and several wells were attempted, but with the exception of those mentioned above and a single well in Crook County, no wells of any importance were drilled. As a rule the people furnishing the money to do the work were unacquainted with the industry, and did not select good sites to drill; they made no survey for depth nor allowance for bad ground to drill through. They have not drilled deep enough to reach the oil sand, and they cannot be induced to drill deeper, even when an oil spring in close proximity is furnishing a constant flow of oil.

The great commercial importance of the oil fields in Wyoming is not fully realized by the people living in the State, much less by those who are not acquainted with the great fields and their numerous oil springs. It is even possible that the development may be delayed until the Eastern supply of oil is less than the demand. But in any event the day is rapidly approaching when oil territory in any part of the United States will be eagerly sought for, and then the industry in Wyoming may be as great as it has been in Pennsylvania. Under

* Irving's *Bonneville*, p. 142.

† *Report of Territorial Geologist*, 1890, p. 63.

existing circumstances progress is necessarily very slow. There are no railroads nearer than 50 miles to the producing wells. The freight rates, including the haul by wagon, will not allow the product to be sold at any of the great commercial centers in competition with Eastern oil. On account of its superior quality the Salt Creek oil commands a very high price, crude oil selling at $9 per bbl. f.o.b. at Casper, and the company finds it under these exceptional circumstances a profitable business. There are other factors that tend to retard development. There are no pipe lines constructed in the State. The supply of oil from the Florence field of Colorado practically meets the demand in the Rocky Mountain region, and this makes it almost impracticable for the Wyoming product to find a market, since it can compete with the Colorado oil only in a few places.

The oil fields of Wyoming are numerous and very extensive. There are 18 known at present, and there have been no locations made except where oil springs

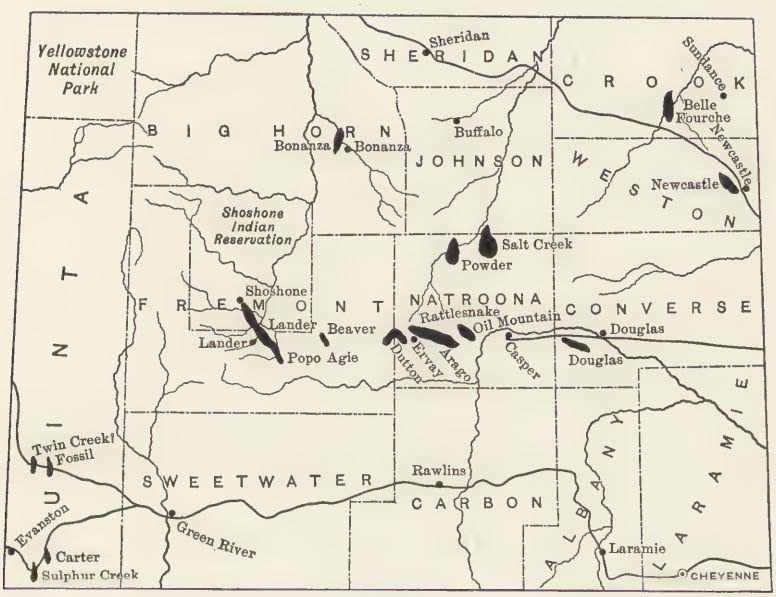


FIG. 1.

or oil-saturated sandstone have been found. As a rule these fields are found along anticlinal folds and are generally miles in extent. On the northeast flank of the Rattlesnake Range, in Natrona County, there is a continuous oil field for 25 miles. Along the foot of the Wind River Range there is an oil field or fields of about the same length. These are the largest, but there are many fields from 3 to 10 miles in length. They are not very wide, as a rule, since the dip of the rocks makes it impossible to drill deep enough to reach the oil sands at any great distance from the axis of the anticlinal. The accompanying map (Fig. 1.) gives the location of the oil fields.

The oil is of all possible grades. In color it ranges from a honey yellow to jet black. In gravity it varies from 0.865 to 0.995. Most of the oils are lubricating, but many produce considerable gasoline and kerosene, and some of them contain as high as 40% of illuminating oil. They are also noted for their low

cold test and great viscosity. The Salt Creek oil is fluid at —17° C.; viscosity at 20° C. is 15.74.*

The accompanying table presents data of the oil fields; it gives the name of the district, county, geological horizon, number of producing wells, production by barrels per day, natural occurrence, color of oil, flashing point, and specific gravity for each.

A TABULATED STATEMENT GIVING GENERAL INFORMATION ON THE OIL FIELDS OF WYOMING. (*a*)

Name of Oil Field.	County.	Specific Gravity.	Baume.	Flashing Point. Fahr.	Color of Oil.	Geological Horizon.	No. Producing Wells.	Product Barrels per Day	Natural Occurrence.
Salt Creek (*b*)..	Natrona and Johnson	0.9100	24.5°	221°	Green.....	Lower Fox Hills or Upper Pierre.	6	60-70	Springs
Powder.....	Johnson.....	0.9241	22.0°	311°	Green.....	Dakota.....	.....	.....	Springs
OilMountain (*c*)	Natrona.....	0.9100	24.5°	234°	Green.....	Dakota.....	.....	.....	Springs
Argo (*d*).....	Natrona.....	.....	.....	.....	.....	Dakota.....	.....	.....	Springs
Rattlesnake(*e*).	Natrona.....	0.9950 to 0.9873	12.1°	.....	Black.....	Fox Hills.. Niobrara... Benton .... Dakota	.....	.....	Springs
Dutton (*f*).....	Natrona and Fremont	0.9220	22.5°	.....	.....	Eocene .... Niobrara... Benton .... Dakota	.....	.....	Oilsand
Beaver.....	Fremont	0.9650	15.2°	280°	D'k brown.	Dakota?.....	.....	.....	Springs
Popo Agie (*g*)..	Fremont	0.9000	25.0°	90°	Black.....	Carboniferous.	3	600	Springs
Lander (*h*).....	Fremont	0.8565	33.0°	117°	Green.....	Fort Benton...	.....	.....	Springs
Shoshone (*i*)....	Fremont	.....	.....	.....	Black.....	Carboniferous.	.....	.....	Springs
Bonanza (*j*) ...	Big Horn....	0.8544	34.9°	133°	Green.....	Dakota.....	.....	.....	Springs
BelleFourche(*k*)	Crook	0.9285	21.3°	211°	Green.....	Dakota.....	1	5	Springs
New Castle.....	Weston.....	0.9200	22.7°	259°	Green.....	Benton?.....	.....	.....	
Douglas (*l*).....	Converse....	0.9210	22.6°	.....	.....	Fox Hills?.....	.....	.....	Oilsand
Twin Creek....	Uinta.....	.....	.....	.....	Black?....	Laramie?.....	.....	.....	Springs
Carter.....	Uinta.....	.....	.....	.....	.....	Eocene?.....	.....	.....	Springs
Sulphur Creek.	Uinta.....	.....	.....	.....	Black?....	Laramie?.....	.....	.....	Springs
Fossil.....	Uinta.....	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) This table has been published in *Bulletin I., Oil Series*, School of Mines, University of Wyoming, 1896, and in *American Manufacturer and Iron World*, May, 1896. As given here it contains some additional data and corrections. (*b*) Regular shipments of oil. (*c*) Oil darkens on exposure at spring. (*d*) Same nature as the Rattlesnake. (*e*) Oil exposed on the surface of water in spring soon sinks. (*f*) Oil distilled from sand. (*g*) Wells packed since being drilled. (*h*) Rich in kerosene. (*i*) The same as Popo Agie oil. (*j*) Rich in kerosene. (*k*) Darkens on exposure at springs; no production. (*l*) Oil distilled from sand.

The geological range of the producing oil sands is the greatest, so far as I am able to learn, that has ever been reported. The lowest oil horizon is the Upper Carboniferous† and the highest is Wasatch Eocene. The thickness of formations between these two horizons varies in Wyoming from 10,000 to 15,000 ft. Most of the oil is found in groups of the Upper Cretaceous, and resembles in that respect the petroleums of Europe and Asia more than the oils found in the eastern part of the United States. There is, however, a marked resemblance between the Paleozoic oil fields of the Eastern States and the Mesozoic oil fields of Wyoming. The oils of the East are found beneath the Carboniferous coal measures, and in Wyoming they occur beneath the Laramie coal measures of the Cretaceous. The accompanying ideal geological section (Fig. 2) will give a clear idea of the oil-bearing horizons known in Wyoming. The oil storage is found to be in sandstone, magnesian limestone, and conglomerate. In the majority of cases the oil is stored in sandstone.

* *Petroleum Series No. 1*, School of Mines, University of Wyoming. † This may be Permian.

As to the origin of petroleum in Wyoming but little can be said of the three prominent theories offered to account for the formation of petroleum; the one proposed by Mendeleeff seems to be the most fitting. The oil horizons are, as a rule, almost devoid of either animal or vegetable remains; and in fact where there was the greatest amount of life in existence, so far as the geological record proves, we find the least oil. The Jurassic beyond a question contained the greatest bulk of animated life of any of the formations in the Mesozoic, and not a trace of oil has been found in these rocks; the Laramie contained the greatest development of vegetable life, and it is not absolutely certain that oil is found in these beds. The recent researches of Moissan tend to make Mendeleeff's theory all the more reasonable.

Only a part of the districts have been studied sufficiently to enable a description to be given. Those in the central and northern part of the State have received the most attention.

At the foot of the Wind River Mountains, in Fremont County, on the eastern

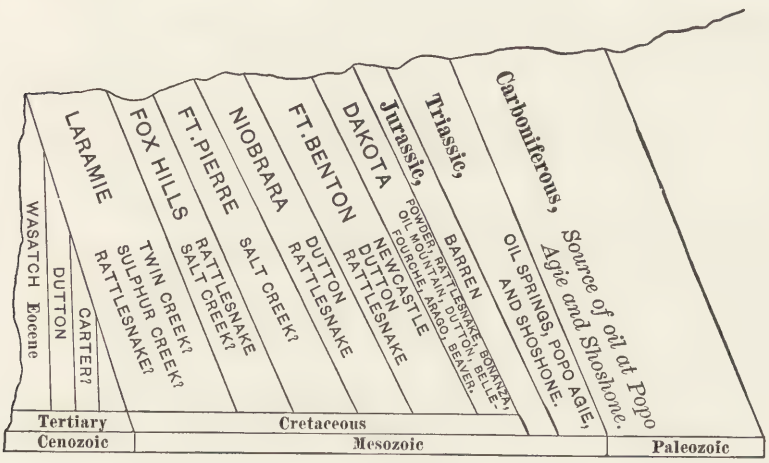


FIG. 2.

slope, and only a few miles away, there is an anticlinal that extends northwest from Beaver Hill, near Hailey, parallel with the mountain, to a point a few miles north of Fort Washakie, a distance of over 40 miles. The core of this anticlinal, so far as it is exposed, is Triassic red sandstone. Along this anticlinal are three oil fields: the Popo Agie at Dallas, 10 miles southeast of Lander; the Lander field, 5 miles north of Lander; and the Shoshone, 1½ miles east of Fort Washakie. The Popo Agie district has been called the Shoshone, but on account of the confusion of names with the Shoshone Indian Reservation and the oil spring on the reserve, the name was changed to conform to the name applied to this district. The oil is found in the Upper Carboniferous, but appears in springs in the Lower Triassic sandstone on the east side of the anticlinal. In this district the axis of the anticlinal has been deeply cut away by the action of the water of the Little Popo Agie River, producing an anticlinal valley. A geological section of the district is shown in Fig. 3.

On the east side of the river the red Triassic sandstone rises precipitously

nearly 500 ft. and dips to the east from 12° to 17°. Above the Triassic is a continuous series of Jurassic and Cretaceous. On the west side the bands change rapidly in dip from the axis of the anticlinal, the dip changing from a few degrees to 70° in a few hundred feet. The rocks on the west have been eroded away so that in the synclinal valley lying west of the oil springs only the lower groups of the Cretaceous remain. The great angle of the dip on the west side of the anticlinal makes it undesirable oil territory on account of the great depth that must be drilled to reach the oil sand. There are four wells in this district: three drilled by the Murphy Company that range in depth from 300 to 800 ft., and one by the Mullin Company that is down 1600 ft. The Mullin well has an artesian flow of water and oil, and has not been properly cased, so that the oil produced cannot be estimated. The Murphy wells will produce, according to various unofficial tests, 200 bbls. each per day if pumped. The oil is black in color, contains a good proportion of kerosene, and also a fair percentage of lubricating oils. All of the Murphy wells are leaking badly, and 10 to 15 bbls. of oil are going to waste each day. The oil is stored in two bends of the river which

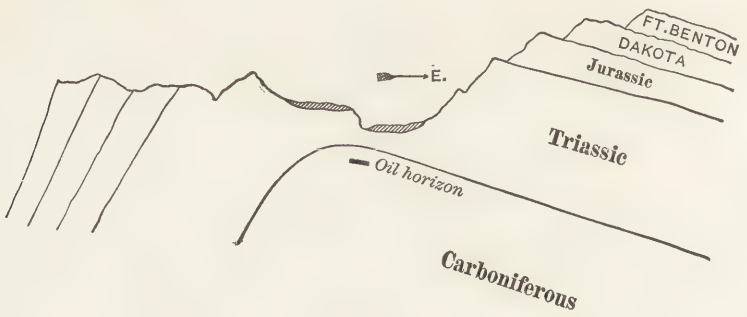


FIG. 3.

have been cut off for that purpose, making two oil ponds aggregating 1500 ft. in length and about 50 ft. in width. In this way there have been several thousand barrels of oil stored.

The Lander* district is located on the same anticlinal, but 14 miles northwest of the Popo Agie. The country in this vicinity is rolling and cut by deep gulches, but mostly covered with loess. The eastern side of the anticlinal is partially exposed for some distance down to and including most of the Fort Benton shales, which dip at a high angle to the northeast. Near the base of the group the oil horizon occurs. For several hundred feet the shale is saturated with oil, and at one place there is a spring that yields about a gallon per day, though there has been no development, since it is located on the Indian reservation. The oil is very light, and as it runs from the spring it evaporates without leaving any hardened residuum or asphaltum. When placed in the hollow of the hand it has a light straw-yellow color and can be wiped off without leaving a stain.

The Shoshone district is located on the same anticlinal as the Popo Agie and the Lander, 10 miles northwest of the Lander district. The conditions here are

*For a more complete description of the Popo Agie, Lander, and Shoshone oil fields, see *Bulletin II., Oil Series*, School of Mines, University of Wyoming, January, 1897.

similar to those of the Popo Agie. On the east side of the anticlinal, but some distance away from the axis, the Triassic red sandstone dips from 12° to 15° to the northeast. On the west the Triassic, Jurassic, and Cretaceous rocks are tilted nearly perpendicularly. The oil spring is in an alluvial flat, about one-half mile north of the Little Wind River. The oil flows very slowly with a stream of water, and has built up a bed of asphaltum that is about 150 by 300 ft. in extent and from 1 to 4 ft. in depth. The oil is black and very viscous, in every particular resembling the Popo Agie oil, and beyond a question it comes from the same horizon. On account of the viscid nature of the oil no tests have ever been applied. This district is also located on the Indian reservation, and has not been developed.

In years to come this anticlinal extending from Dallas to Fort Washakie will probably show a line of oil derricks. The width of this oil belt seems to be limited only by man's ability to drill deep enough to reach the oil sand. It must also be stated that as a whole this belt is an important gas horizon. Gas accompanies the oil in every instance, and the gas pressure at the Murphy wells is so great that the casings had to be weighted down as soon as the packers were put in; even with this precaution the gas forces its way up around the casings, and carries with it large quantities of oil.

The Beaver oil basin is located 40 miles east of Lander, in Fremont County, at the termination of an anticlinal that extends southeast, and only a few miles away is covered by Eocene Tertiary. In this vicinity the Cretaceous rocks stand almost perpendicular, but on account of the slight exposure it is difficult to locate the oil horizon with certainty. It is probable, however, that the oil in this district is found in the Dakota sandstone. Springs occur in three places that are about half a mile apart, and slight beds of asphaltum occur at two of these springs. At one of the springs an oil well was started but never completed. In one place where a prospect hole, 4 by 5 ft. inside of timbers, was dug, gas escapes with such rapidity as to cause the water filling the hole to boil vigorously. The gas is chiefly marsh gas, and burns with a bright flame when ignited. The extent of this field is less than is usual in Wyoming. The oil is a very superior lubricant, and judging from the flow at one of the springs the district will be very productive.

Ten miles west of Ervay, in Natrona County, and extending into Fremont County, is the Dutton oil basin. This field also lies at the termination of an anticlinal similar to the Beaver oil field, except that in the Dutton the termination is quite broad and exposes a Triassic core. The anticlinal two miles southeast of the northern termination is two miles wide. There are no oil springs at the present time from which oil could be collected, but oil is found in small quantities at many springs, and bands of oil sand are very extensive. The oil occurs in the basin in the Dakota, Fort Benton, Niobrara (Cretaceous), and Wasatch Eocene. There has been no development with the exception of prospector's 10-ft. holes, and nothing is known of the nature of the oil except as it has been distilled from the oil sand, but the conditions are favorable to the production of a large yield of lubricating oil. Gas is also an important product accompanying the springs of water on which some oil is found.

The northeastern slope of the Rattlesnake Mountains is one great oil field. It

has been divided into two districts—the Rattlesnake and the Arago.* The Rattlesnake occupies that part of the range north and west of Garfield Peak, and the Arago occupies the southeastern portion of the range. One description will include the two districts. The structure of the Rattlesnake cannot be satisfactorily explained, but the northern side has every appearance of the flank of an anticlinal. The Archean is possibly exposed, and above the Archean is a series as follows: Cambrian, Silurian, Carboniferous, Triassic, Jurassic, and Cretaceous. The sedimentary rocks dip from 20° to 35° to the northeast and form a series of high ridges parallel to the range. A section is shown in Fig. 4. Oil is found in the Dakota, Fort Benton, Niobrara, and Fox Hills groups. Oil springs are very numerous in the Dakota sandstone and break out at intervals for a distance of 25 miles. The oil above the Dakota is seldom found in springs, but usually as oil sand, which in the Fox Hills has a thickness of not less than 60 ft. Wells have been attempted at several points along this belt, but none of them seem to have reached the oil horizon. The oil taken from the springs in the Dakota

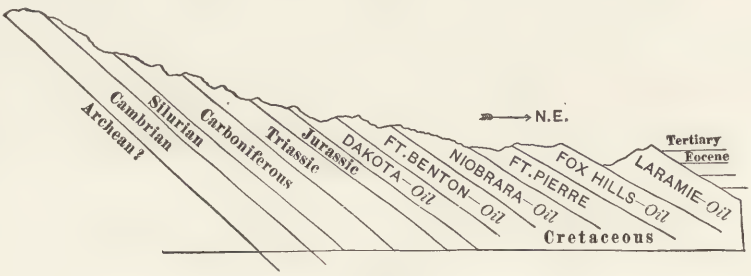


FIG. 4.

sandstone is dark-colored and very heavy. It comes to the surface with water, but soon becomes so heavy that it sinks. The oil tested has a sp. gr. of 0.995.

Oil Mountain district is in Natrona County, 25 miles west of Casper. As in the other districts referred to, the oil is found associated with an anticlinal fold. The Oil Mountain uplift extends from a point a few miles west and north of Bessemer, north and west toward the Big Horn Mountains. The anticlinal is quite sharp, and with the exception of the region in the vicinity of the oil spring near the Oil Mountain peak, it has been eroded away, leaving the surface to the north rough, rolling upland. The oil is probably found in the Dakota sandstone, but comes up through a fracture in the Fort Benton shales. One well was drilled and oil struck, but it was cased off to drill deeper for a second sand. The extent of this district cannot be approximated, but the width of the oil-producing territory will necessarily be limited, owing to the dip of the rocks. One singular feature of this field is that the Rattlesnake district is separated from this by a synclinal valley, and the oil found in the same group of rocks has no resemblance, the Oil Mountain oil being green when it comes to the surface of the water at the spring, and the Rattlesnake dark-brown or black. The Oil Mountain oil has a sp. gr. of 0.910 and the Rattlesnake of 0.995.

The Powder River district is 35 or 40 miles almost due north of Oil Mountain.

*This name has been given to me as Argo by some and Arago by others. So far I have been unable to trace its origin.

This district has not been studied, but the oil is found in springs along an anticlinal fold in Dakota sandstone. Some wells have been drilled, but on account of caves and other misfortunes there is no production of oil. The oil is a dark-green color as it is taken from the spring, and is a lubricating oil much resembling that of the Salt Creek district.

The Salt Creek district* is located 50 miles north of Casper, and is the only district in the State now producing oil for the market. The oil is found in sandstone, which is probably Fort Pierre, exposed by the erosion of an anticlinal fold. In this district three companies have started wells, the Pennsylvania Company having completed six which produce oil, the Wyoming Lubricating Company having drilled one to the depth of 1200 ft., and the French Syndicate having also drilled a well to considerable depth. Neither of the last two wells has yet struck oil. The depth of the completed wells at Salt Creek varies from 800 to 1300 ft., and the production by using pumps is 10 bbls. per well per day. The oil is a remarkable natural lubricant, having a sp. gr. of 0.910, flashing test of 221° F., viscosity at 20° C. of 15.74, cold test fluid at —17° C. It does not contain any tar or acid. On account of its superior quality the company has been able to place it on the market in competition with manufactured products from Colorado and the East. The extent of this district cannot be accurately given, but it is quite large, having a length of at least 6 miles and a width quite as great, owing to the slight dip of the formation.

On the west slope of the Bear Lodge Mountains, in Crook County, and lying along the Belle Fourche River, is the Belle Fourche oil district. It has been studied only in a preliminary way, and no detailed account can be given. The field is apparently quite extensive, and the oil is found in Dakota sandstone and possibly in the Fort Benton. Several wells have been drilled, most of them not deep enough to reach the oil sand, but possibly they have not been sunk in the best location. The oil is dark-green in color as it seeps into the basins at the springs, but darkens rapidly upon exposure. The springs are very productive and indicate a strong flow. One very shallow well has furnished 5 bbls. per day when pumped. Little or no development work has been done during the past six years.

In Big Horn County there have been several oil fields reported, but that at Bonanza is the only one, so far as known, that is of importance, and it is 175 miles northwest of Casper. At the foot of the Big Horn Mountains, and parallel to them, in the vicinity of Bonanza, there are three anticlinal folds extending for a few miles only. Along the central fold in the Benton shales there are numerous places where oil oozes out of fractures in the shale. The source of the oil is unquestionably the Dakota. There has been no development work of any consequence done. The oil is collected in pits sunk in the shale, and one well has been attempted. The oil resembles the oil of the Lander district and will yield a large percentage of kerosene as well as lubricating oil. The district is not large, but should furnish a strong flow.

West and north of Newcastle, in Weston County, there is an extensive ridge of Fort Benton shale, extending many miles to the northwest, beneath which there

* *Engineering and Mining Journal*, Jan. 25, 1896, and *Bulletin No. 1, Petroleum Series*, School of Mines, University of Wyoming, June. 1896.

is a well-developed band of Dakota sandstone that extends northeast to Cambria, and dips to the southwest. Oil springs are found in the bands of upper Dakota sandstone in several places, and there are also many bands of oil-saturated sandstone. Considerable money has been spent in drilling in this vicinity, but the drillers have not been successful in casing off the water and handling caving ground. One well produces oil and water. Work will be carried on extensively during 1897. The oil is a heavy lubricating variety of a high grade.

Twelve miles southwest of Douglas, in Converse County, a heavy band of oil sand was discovered in 1894. The geological horizon is probably Fox Hills. The oil distilled from the sand was lubricating, resembling other good lubricating oils found in the State. The rocks in this vicinity have been tilted to the northeast by the Laramie Mountain uplift, and the Fox Hills bands extend westward along the range, bending to the north a few miles west of Casper. A company commenced a well in this district in the fall of 1896.

The districts of Fossil, Carter, Twin Creek, and Sulphur Creek are all located in Uinta County, and are near the railroad. In the vicinity of Sulphur Creek and Carter Wells there are numerous oil springs. At Carter two wells have been drilled, but neither reached the oil sand. Work was commenced at Carter in 1869 and was resumed in 1884, but since that time no work has been done. The exact geological horizon cannot be stated, but it is safe to say that the most of the oil is found in the Upper Cretaceous. The oil is these districts is found along the axes of anticlinals running almost north and south.

The districts of Fossil and Twin Creek are almost north of Sulphur Creek on the Oregon Short Line Railroad. Two wells have been drilled in these districts, but on account of the soft nature of the rock penetrated, the wells caved and were soon filled with clay. One of these wells struck a heavy flow of gas and oil, and when the cave caused the drillers to stop, the clay was forced through the casing in a consistence fit to make brick, and would sometimes rise several feet before breaking off. In these districts springs are numerous, and many of them yield a gallon or more of oil each per day. The geological horizon is supposed to be Eocene, but has not been absolutely determined. No samples of Uinta County oil nave been tested or analyzed so far as known.

PHOSPHATE ROCK.

THE phosphate industry in the United States was less prosperous in 1896 than in the previous year. This was due to the falling off in the foreign demand, which is being supplied more and more by the Algerian mines. With these the Americans cannot apparently compete successfully on equal terms, though, owing to the interference of the French government, conditions were not very cheerful for the Algerian producers last year, and they were urging reductions of railway rates, etc., in order to be able to continue to compete with the Americans. However, there is a still more serious competitor in the Thomas slag, an artificial phosphate, which is every year displacing the native mineral to a greater extent. As a consequence of these changes in conditions, phosphate rock, 58 to 60% calcium phosphate, sold in Charleston from $3.25 down to $3.00, and river rock as low as $2.85, the lowest prices being quoted at the close of the year.

Under the existing conditions the comparatively small decline in the Florida shipments is misleading. Advices from the producers indicate a general suspension of operations, on the ground of no profit in working, whence we infer that the shipments have been to a large extent from accumulated stocks.

The State Phosphate Inspector of South Carolina, in his report of operations for the year ending August 31, 1896, names the following companies and individuals as engaged in mining and shipping rock during the year: Coosaw Company; Farmers' Mining Company; Beaufort Phosphate Company; W. Y. Tripp; James Reid; Carolina Mining Company, and John C. Nelson. The total amount of rock mined during the year ending August 31, 1896, is estimated at 123,708 tons, as against 196,777 tons for the previous year. The total number of tons of rock shipped during the year was 121,602 tons, being a decrease of 52,797 tons from the previous year. The amount of State royalty was $60,854.

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES.

(In tons of 2240 lbs.)

Year.	South Carolina.	Florida.	North Carolina.	Ohio.	Tennessee.	Totals.	Value.	
							Total.	Per Ton.
1892	548,396	354,327	6,000	.....	.....	908,723	$3,354,098	$3.68
1893	556,883	426,432	7,500	.....	.....	990,815	3,467,853	3.50
1894	493,800	558,990	9,000	.....	17,384	1,079,174	3,291,481	3.05
1895	515,734	530,356	7,500	.....	45,329	1,098,019	3,296,757	3.00
1896	314,360	499,065	7,418	439	42,911	863,754	2,591,262	3.00

Algeria.—While the American phosphate producers have been suffering severely from foreign competition it does not appear that the Algerian have been any more prosperous, although they (the more important being English) have been attacked violently in the French senate and press, on the ground that the mineral wealth of the colony was being squandered by the concessions granted. It was proposed, therefore, that an export duty should be levied on rock exported by English companies to foreign countries. Sir Lambert Playfair, the British consul-general, in a report on Algeria, showed the injustice of these attacks.

An English house had taken over a concession to a Frenchman, M. Wéterlé, near Souk Abras, and lost £48,000 in the speculation; a French firm lost even more in attempting to work the deposits in the valley of the Chéliff; so when M. Chapelle discovered the deposits at Tebessa he could find none of his countrymen disposed to exploit them, and finally interested Mr. Crookston of Bône, to whom belongs the credit of developing the industry.

The concession granted was for 18 years and there was no possibility of Mr. Crookston extracting more than 60,000 tons per annum. The actual cost of mining and shipping 60% phosphate rock from Bône is 34.3f. per ton, and at present prices only the highest grade can be worked profitably. The lower grade deposits have been temporarily abandoned.

The total production of the Tebessa district rose to 600 tons per diem toward the close of 1895; in 1896 it fell as low as 300 tons at certain times. Of the companies operating in the district the Société Français entirely suspended operations; the Constantine Phosphate Company of Leith reduced its output to 100 tons per diem; and the Crookston to 200 tons.

The consul-general concludes as follows:

"Thus it is seen to what insignificant proportions this enormous natural treasure is now reduced, and how exaggerated have been the expectations regarding it. It cannot be doubted that this industry, once so promising, has been shriveled up, mainly owing to the political campaign that was directed against it, and to the subsequent action and interference of the State."

The shipments from Bône, which are entirely the yield of the Tebessa district, are given in the subjoined table. The apparent increase in 1896 is due to the dispatch of accumulated stocks, which had been very large.

SHIPMENTS OF ALGERIAN PHOSPHATES FROM THE PORT OF BÔNE.

Shipped to—	Metric Tons.		Shipped to—	Metric Tons.		Shipped to—	Metric Tons.	
	1895.	1896.		1895.	1896.		1895.	1896.
France	34,835	34,661	Austria	8,645	4,240	Switzerland	1,550	.....
Italy	16,395	20,170	Belgium	3,015	1,500	Japan	.....	3,500
Germany	7,710	17,490	Spain	555	250	Other Alg'n ports.	420	1,324
England	37,870	54,005	Holland	1,000	2,900	Total	111,995	140,040

M. Mouffet, in a letter in *La Politique Coloniale* of Paris, reports the discovery of deposits of phosphate of lime at Kayes, in Senegal, French Africa. He says that outcrops of high-grade rock occur over a considerable extent of country.

Phosphate Mining in Tennessee.

By Lucius P. Brown.

The chief feature of phosphate mining in Tennessee in 1896 was the change from the long wagon-hauls and underground work of the Blue Rock field to the easy transportation and open-cut mining of the Mt. Pleasant territory. The latter was opened during July, and up to that time the most serious problem confronting the Blue Rock miners was, as it has always been, transportation. The Swan Creek Phosphate Company had practically solved this by building a narrow-gauge road to its mines, and the Duck River Phosphate Company had in contemplation a line (since completed) from its mines to the mouth of Swan Creek, whither a branch had been built by the Nashville, Chattanooga & St. Louis Railway from Centerville, presumably with the ultimate purpose of continuing the line up Swan Creek, so as to afford a common outlet to the mines in the lower portion of the valley.

Since the discovery of the higher grade and more cheaply mined phosphates of Mt. Pleasant the mining has been done mostly in that district. For a few months following the beginning of work at Mt. Pleasant, the production of rock in the State decreased considerably. The Blue Rock miners were unable to meet Mt. Pleasant prices, and the Mt. Pleasant miners were not in a position to ship much rock; besides which consumers were probably a little slow to take hold of a rock entirely new to them. Time brought no improvement in prices, and consequently very little Blue Rock is now being shipped, and that little only from the most advantageously situated mines. The only mines now being operated in the Blue Rock district are those of the Southwestern Phosphate Company, J. H. Daniel & Co., and the Duck River Phosphate Company.

Although the opening of the Mt. Pleasant field has well-nigh closed the Blue Rock mines, it still seems probable that those which are situated very near a railroad will be able to mine at least a few thousand tons yearly. The rock has certain qualities that commend it very highly to manufacturers, and in admixture with Mt. Pleasant rock it makes a very desirable acid phosphate.

The Mt. Pleasant Field.—In January, 1896, Judge S. Q. Weatherby, of Lewis County, found on a farm near Mt. Pleasant a rock resembling the weathered Blue Rock, which analysis showed to be a very high-grade phosphate. A few months later a little exploration was done, and about July 15th work was regularly begun in the field. The town of Mt. Pleasant, whence the field takes its name, lies in Maury County, 11 miles southwest of Columbia and 56 miles from Nashville. It is situated upon the Nashville, Florence & Sheffield Division of the Louisville & Nashville Railroad, and has therefore good railway connections. The workable phosphate deposit is comparatively small in area, but the proportion of available rock to area of the field is large. The field seems to be an irregular quadrilateral in shape, extending 6 or 8 miles east and west, and 3 or 4 north and south. The railway cuts through the western end of the field.

The Blue Rock, as noted in Vol. III. of The Mineral Industry, belongs to the Devonian age. The Mt. Pleasant beds, on the other hand, are Lower Silurian,

belonging to the "Capitol Limestone," a member of the Nashville (Cincinnati) formation. The Capitol Limestone is found scattered all over the "central basin" of Tennessee, and wherever it is found some phosphates are discovered, but no workable deposits have so far been disclosed except near Mt. Pleasant. In many places, indeed, the phosphate degenerates into an impure sandstone, containing only a few per cent of phosphoric acid.

As to the origin of the phosphates, about all that can be said is that the deposits of Mt. Pleasant seem to be the residue left after the natural leaching of certain highly phosphatic limestones by carbonated waters, leaving behind the less soluble phosphates.* The limestone immediately underlying the phosphate is rich in phosphates, containing sometimes as much as 50% bone phosphate, and an outer, weathered coating, sometimes 3 or 4 in. thick and of a pearl-gray color, contains about 10% more. This limestone is several feet thick.

There are two varieties of the Mt. Pleasant rock, the soft and the hard. The former is the type of the field, since the latter is insignificant in quantity, so far as yet discovered. The soft rock, when dry, is of a color varying from grayish or reddish brown to creamy white, and occurs in layers or plates of varying thickness and size, but rarely more than 2 or 3 ft. in diameter. These plates are packed closely together into a more or less regular stratum. The stratum is not always uniform in thickness, but often sinks into depressions in the underlying limestone, and "chimneys" of the latter sometimes reach the surface of the overburden. This overburden varies very much in depth, but averages about 3 ft., except in the eastern part of the field, where it becomes probably so deep as to preclude successful mining in very many places. In texture the phosphate rock is loose and porous, grinding easily when dry. Its specific gravity is about the same as that of the Blue Rock, but in the mass it is much lighter, and a cubic yard of ordinary rock in the bank will yield about 1400 lbs. of merchantable rock.

The "hard rock" of Mt. Pleasant is a hard, compact material, with a structure almost oölitic, yellowish white in color, and not porous. It is so far found at only one point, where it seems to replace the soft rock in the beds, and where a few hundred tons have been taken out. It occurs, like the Blue Rock, in a well-defined, solid stratum about 2 ft. in thickness. Occasionally rose-colored streaks are found in it, which are often arranged in concentric circles, the center of which, on being broken open, is found to be a blue phosphate indistinguishable by the eye from certain varieties of the Swan Creek rock. The rosy rings are apparently due to the oxidation of pyrite contained in this unaltered blue center. The "hard rock" is uniformly high in bone-phosphate and low in oxides of iron and alumina, but so small in amount as to cut no figure on the market.

The Mt. Pleasant phosphate is not essentially different in chemical composition from the other rock-phosphate deposits of the world. It contains no pyrite, little fluorine, and is usually high in bone-phosphate. Since iron and alumina are present in the rock as it comes from the mine, in rather large amounts, washing or some other method of cleaning is usually necessary to reduce them to the fig-

* Dr. J. M. Safford, *American Geologist*, October, 1896, to which journal refer for a more extended account of the geology of the region.

ures demanded by the trade. They probably exist as oxides or silicates, and are brought into the rock from the soil above, suspended in water, and are deposited in the jointings between the plates. Large lumps of the rock, carefully cleansed from adhering matter, show little iron and alumina.

The subjoined analyses, while mainly made for commercial purposes, show well the character of the rock, as well as of the limestone accompanying it.

	Average of more than 100 Samples of all sorts.	"Composite" Sample of Raw Rock.	"Hard Rock"	Average of 29 Samples of Shipments, Picked Samples, etc.	Unaltered Limestone.
Moisture	.84%	1.05%	.60%	.84%	.27%
Phosphoric acid	34.77	34.69	37.00	35.94	22.59
Bone-phosphate, calculated on dry basis	76.47	76.42	81.14	79.02	49.38
Iron oxide	4.46	2.39	2.84	4.29	1.45
Alumina		3.36			
Insoluble matter		4.05			1.78
Carbonic acid		(*a*) 1.42			(*d*) 18.11
Lime (CaO)		46.76			
Oxide of magnesium		(*b*) .59			
Organic matter		2.64			
Sulphuric oxide		(*c*) 1.68			
Fluorine and undetermined		1.39			

(*a*) Equivalent to lime carbonate, 3.18%. (*b*) Equivalent to magnesium carbonate, 1.24%.
(*c*) Equivalent to lime sulphate, 2.84%. (*d*) Equivalent to lime carbonate, 41.16%.

The mining of the Mt. Pleasant phosphate is all open-cut work. The overburden, up to 8 or 9 ft. thick, is removed by hand-labor or wheeled scrapers, to expose a good working face. The rock is then got out with pick and crowbar, no explosives being used, and the larger lumps, together usually with what fails to pass through an ordinary "ballast-fork," are carried away in wheelbarrows or trams. The tracks for the latter are laid out as in South Carolina land-rock mines.

Some rock has been shipped just as it comes from the mines, but this practice no longer obtains, and now it is all cleaned and dried. As in the pioneer days of Florida phosphate mining, so here, the best practice for these operations has not yet been settled upon. At present the rock is dried usually by piling under suitably constructed sheds on a cribwork of cord-wood, which is then fired, and dries the rock thoroughly. The dry rock is then passed through a rock-crusher, and over a screen to bins or cars. Miners without a plant break with hammers and handle the rock a couple of times with ballast-forks. The work of mining, etc., is as yet of so experimental a nature that no reliable figures of cost on board cars can be given.

The chief companies in the field, with their head offices and developments, are as follows:

Blue Grass Phosphate Company; offices in Nashville; mines west of Mt. Pleasant; developments consist of storage and kiln sheds, spur-track, and considerable open-cut work.

T. C. Meadows & Co.; offices at Mt. Pleasant; own the only deposit of "hard rock" so far discovered; developments consist of storage sheds at mines and at Mt. Pleasant, with Aultman-Taylor crusher, screens, Sturtevant mill, and engines. This company grinds its screenings, and sells to small mixers, and for direct use.

Tennessee Phosphate Company; head office at Louisville, Ky.; mines about a

mile north of Mt. Pleasant, on the railroad; developments consist of sheds for dry-kilns adjoining a crushing, elevating, and screening plant, consisting of a large Gates crusher, with elevators, and a rotary screen, the screened rock from which goes directly to cars. The work of installing a washer plant, with rotary dryers, etc., is now going on; in addition to this the company has storage bins for 3500 tons of dry rock, and a large working-face in its mine, with admirably arranged tram-tracks in mine and from mine to mill. This company is arranging its works with a view to a very large output.

Hardy & Bryan; office at Mt. Pleasant; mine adjoins property of Tennessee Phosphate Company on the south; buildings consist of storage and kiln sheds, and a spur-track has been built. They have shown up a large body of easily mined rock, and contemplate the erection of crusher plant, etc.

Columbian Phosphate Company; offices at Columbia, Tenn.; mines are a little south of those of Hardy & Bryan, and adjoin those of the Blue Grass Company on the north.

As will be noted from the analyses given above, the rock from this field compares very favorably in content of bone-phosphate with the "hard rock" of Florida. Although the raw rock contains more iron and alumina than the Florida product as now shipped, it is certain that the percentage of these can, in the cleaned rock, be lowered to the figures demanded by the trade. Besides this, the Tennessee rock is so soft that it can be ground in almost any mill (in fact, the writer knows of a common buhrstone mill now successfully in use), and this item of cost to the factories is very much reduced. When properly handled the rock makes a light gray acid phosphate with a very low percentage of "insoluble" matter.

It seems at present simply a question of freights as to whether the Mt. Pleasant miners will be able to sell their rock abroad and in seacoast markets. The supply of rock is apparently large enough to answer any demands upon it for some years to come; and while low prices of phosphates at present do not favor a rapid expansion of mining operations, it is nevertheless probable that the growth of the industry will be steady.

The production of phosphate rock in Tennessee during 1896 was as follows: January, 4165 tons; February, 4136; March, 4060; April, 2875; May, 3970; June, 4100; July, 4304; August, 6322; September, 4954; October, 4394; November, 5692; December, 8870—total, 57,846 tons.

PLATINUM.

THE production of platinum in the United States in 1896 was 200 ozs., nearly all of which was derived from California. The principal source of the world's supply of this metal continues to be the Ourals. The price of platinum steadily advanced during the year, and at present it is quoted at $13.50@$14.00 per oz. on large contracts.

California.—Platinum is found in the beach sands near Crescent City, Del Norte County, and also in the auriferous sands of Smith River, 4 miles east of Crescent City. In Humboldt County nearly all the beach sand contains more or less; at the Union mine the gold saved contains about 5% platinum, and at the Little River mine 50%. At LaPorte and along Spanish Creek in Plumas County platinum occurs in the auriferous gravels, but more abundantly still in the tributaries of New River, especially Devil's Cañon Creek in Trinity County; also on the north and east forks of Trinity River, and near Taylor Flat on French Creek, where some large nuggets are found.

Colombia.—According to Robert B. White,* the Novita Vieja mine, situated about a mile from the river Tamana, one of the principal tributaries of the San Juan, which falls into the Pacific Ocean a few miles north of the Bay of Buenaventura, consists of a bed of cement or conglomerate, which has been laid bare by old alluvial workings over a length of two miles or more, and a width of one-half a mile. Above the cement there existed beds of gold-bearing gravels, sands, and clays, which were washed off by ground sluicing in the time of the Spaniards, and as these old workers had no means of treating the cement they left it as bed-rock. The cement bed is from 6 to 12 ft. thick. It forms the flat bottom of a valley with a fall of 2%, which joins the valley of the river Cajon or river Negro. In 1878 a series of prospects showed a produce on the average of 1 oz. per ton, of which half was gold and half platinum.

Baker & Co., New York, exhibited a big nugget of platinum, which was said to be the second largest in the world. It was irregular in form, measuring roughly 3 by 2¾ in. and weighing nearly 2 lbs. No analysis was made of it but from its specific gravity it was estimated that it was about 85% pure. The nugget came from the interior of the United States of Colombia, but its precise source is unknown.

Hungary.—It has lately been reported that small veins carrying platinum have been found in syenitic rocks in Hungary.

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* *Engineering and Mining Journal*, Feb. 20, 1897.

New South Wales.—Apparently an important discovery of platinum has been made in New South Wales, in the country around Fifield, where the occurrence of the metal was recorded as early as 1887. Nothing was done there, however, until 1893 when alluvial gold was also discovered and a rush to the district set in. The important lead of platinum was found subsequently. This lead, according to J. B. Jaquet, Government Geological Surveyor, runs north and south for more than a mile, varying in width from 60 to 150 ft. It is overlaid by 60 to 70 ft. of loam. The platinum and gold occur in small, well water-worn grains, in crevices in the bed-rock and the gravel lying immediately thereon. Nuggets as heavy as 5 dwts. have occasionally been found.

In the treatment of this gravel it is first puddled in machines worked by horses and then washed in ordinary sluice-boxes. The gold is extracted by amalgamation with mercury, crude platinum being left behind. The latter realizes at the present time upon the field 24s. per oz. It contains about 75% of platinum, the remainder being chiefly platinoid metals. An analysis by Mr. J. C. H. Mingaye, analyst and assayer to the Mines Department, was as follows: Platinum, 75.90%; iridium, 1.30%; rhodium, 1.30%; palladium, traces; osmiridium, 9.30%; iron, 10.15%; copper, 0.41%; gold, none; lead, traces; silicious matter, 1.12%—total, 99.48. In treating parcels of dirt from various claims upon the lead, the quantity of platinum per load varied from 5 to 12 dwts., and the quantity of gold from 1 to 3 dwts., while the total value of the precious metals per load varied from $2.25 to $9.25.

Dry seasons have prevailed since the discovery of the field, and its development has been much retarded in consequence. The washing of dirt has sometimes been completely suspended for many months at a time on account of the shortage of water. At the present time 7000 loads of wash-dirt are dumped around the various shafts awaiting treatment.

About 1200 ozs. of crude platinum have already been sent away from the field, and including the deposits in the immediate vicinity of Fifield, and outside the Fifield-Platina lead proper, the gold won has amounted to about 1800 ozs.

Small quantities of drift yielding a payable quantity of platinum, associated with gold and tin (cassiterite), have been mined about 10 miles northeast of Fifield, near the village of Burra Burra.

Russia.—The production of platinum in Russia continues to be almost entirely from the placers of the Oural, but new deposits of considerable richness are said to have been discovered in the bed of the river Tura, government of Tomsk, Western Siberia. The quantity reported to the Imperial authorities for six years past has been as follows: 1890, 2834 kgms.; 1891, 4242; 1892, 4575; 1893, 5094; 1894, 5209; 1895, 4416.

For 1896 the output is estimated at 4800 kgms. It is believed that real production is above that reported, the miners themselves stealing a part, while another part is concealed to avoid the payment of tax. Some authorities estimate that 25%, at least, should be added to the official statement.

The smelting, refining, and working of the platinum are done chiefly in Germany, whence Russia imports all the manufactured platinum she requires.

PYRITES.

IRON pyrites is produced extensively in the United States as a source of sulphur for sulphuric acid making, the chief sources of supply being the Davis mine, near Charlemont, Mass., and a few mines in Louisa and Prince William counties, Va., and in Gaston County, N. C. Recently deposits of pyrites were opened near Hot Springs, Ark., and a small amount of ore was produced by them in 1896, but their output is not yet important.

The Arkansas pyrites assays from 48 to 52% sulphur, 40% iron, and 10% zinc, together with a small amount of copper and traces of gold and silver. It is claimed to be free from arsenic. The ore occurs in fissure veins, standing nearly vertically, intersecting a sandstone country-rock. As opened at the surface the veins are wide, consisting chiefly of quartzose vein-matter, divided by streaks of pure pyrites. One vein has been traced on the surface a distance of more than a mile. So far two deposits have been explored, but neither of them extensively.

Besides the domestic pyrites American acid makers import a large amount, chiefly Rio Tinto (Spanish), ore which contains a little copper. Diminishing supplies are also obtained from Capelton, Canada, where copper pyrites is mined, and from Pilley's Island, Newfoundland, which sends an iron pyrites—copper-free—containing at one time nearly 50% sulphur, but now running much below this figure.

The production, imports, and consumption of pyrites in the United States are stated in the following table:

THE PYRITES INDUSTRY IN THE UNITED STATES.

(In tons of 2240 lbs.)

Year.	Production.		Imports.		Consumption.	
1892	106,250	$357,000	210,000	$630,000	316,250	$987,000
1893	95,000	285,000	194,000	582,000	289,000	867,000
1894	107,462	466,466	146,023	590,905	253,485	1,057,371
1895	107,371	342,587	190,436	673,812	297,787	1,016,669
1896	117,782	297,853	199,678	1,140,571	317,460	1,438,424

This does not include a small amount of gold-bearing pyrites which is used for acid-making in Denver, Colo., the residues being smelted for their gold contents. The utilization of waste fumes for acid-making has not yet become general in

the United States, though at La Salle and Peru, Ill., the gases from the blende-roasting furnaces and at Clifton, Ariz., those from roasting copper pyrites, are thus employed, while a small plant for the manufacture of sulphuric acid from the fumes of furnaces roasting pyritous ore was installed last year by the Consolidated Kansas City Smelting and Refining Co. at its Argentine works.

The price of pyrites in New York and vicinity in 1896 averaged 14½ cents per unit. The increased consumption of pyrites in 1896 is attributed to the high price of brimstone that was maintained during the latter part of the year. Estimating the yield of all the pyrites used at 45% sulphur, the increase in consumption over 1895 was equivalent to about 9000 tons of brimstone.

The production of sulphuric acid in the United States in 1896 was upwards of 1,000,000 tons (of 2000 lbs.), the figure being reduced to the basis of acid of 66° B. The production of the blende-roasting works of Illinois was 23,300 tons. With this exception and the small amount made at Clifton, Ariz., the entire product was derived from brimstone and pyrites. Of the total production of acid about 75% was consumed in the manufacture of fertilizers.

For further details as to the sulphuric acid industry reference should be made to the articles on "Sulphur" and "The Chemical Industry."

Spain.—The Sociedad des Agŭas Teñidas produced 200,000 tons of iron pyrites in 1896, which was 40,000 tons less than in 1895. The production of cupriferous pyrites in this kingdom is referred to in the review of the copper industry in the first part of this volume.

QUICKSILVER.

THE production of quicksilver in the United States in 1896 was derived wholly from California as in previous years. The new discoveries in Texas, which were reported in THE MINERAL INDUSTRY, Vol. IV., have not yet become productive. In California there were no changes of importance in the industry, though some new prospecting has been done in Siskiyou County.

The New Idria mine in San Benito County was acquired by the Napa and Ætna companies, which organized a company with $500,000 capital for its exploitation, and erected new reduction works. Exploration work was also carried on in the Keystone mine, Cleelum district, Kittitas County, Wash., where ore assaying 2 to 10% mercury is said to have been opened.

Literature.—The metallurgy of mercury was fully treated in Vol. IV. of this series. The estimation of mercury in ores, etc., has been discussed by M. G. Deniges, *Bull. Soc. Chim.* 1896, 15–16, pp. 862–871, and *Journal Soc. Chem. Ind.* 1896, Sept., 678.

QUICKSILVER STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.			Exports.		
	Flasks.	Metric Tons.	Value.	Flasks.	Metric Tons.	Value.	Flasks.	Metric Tons.	Value.
1891	22,926	777	$1,037,381	*nil*	....	....	3,714	129	$145,502
1892	27,993	974	1,119,720	*nil*	....	....	3,518	122	133,626
1893	30,164	1,046	1,108,527	395	13.7	$12,507	16,631	575	539,385
1894	30,440	1,056	1,095,840	*	....	6	14,408	500	397,528
1895	33,978	1,179	1,313,589	*nil*	....	....	15,542	539	482,085
1896	33,180	1,151	1,227,660	*nil*	....	....	19,944	692	618,437

* Seven pounds.

Prices.—The New York market opened at $40 per flask in January, with £7 7s. 6d. in London. These prices held until the middle of February, when the New York quotation fell to $37.50, and London to £7 2s. 6d. Another sharp drop occurred in March, quotations going to $35 and £6 17s. 6d. respectively, but the New York market quickly recovered to $37.50. In May and June the New York quotation shaded off to $37, following the successive drops to £6 15s. and £6 10s. in London. Again in July and August prices fell to $35.50, the London rate touching £6 7s. 6d.; but in the early part of September there was a reaction to $36 and £6 10s., and toward the end of the month to $36.75 and £6 12s. 6d., which quotations were maintained to the close of the year.

PRICE OF QUICKSILVER PER FLASK IN NEW YORK.(*a*)

Month.	Highest.	Lowest	Month.	Highest.	Lowest	Month.	Highest.	Lowest	Year.	Highest.	Lowest
January.	$40.00	$40.00	May ...	$37.50	$37.00	Sept....	$36.75	$35.50	1893....	$43.50	$30.00
February	40.00	37.50	June...	37.00	37.00	Oct.....	36.75	36.75	1894....	37.00	27.50
March....	37.50	36.00	July....	37.00	35.50	Nov....	36.75	36.75	1895...	41.00	35.90
April	37.50	37.50	August	35.50	35.50	Dec. ...	36.75	36.75	1896....	40.00	35.50

(*a*) The San Francisco quotations are usually $0.25 less than the New York.

THE QUICKSILVER MINES OF CALIFORNIA.

The following article is an abstract of the chapter on Quicksilver in the 13th report of the State Mineralogist of California, published September 15, 1896. It is interesting as showing the condition of every quicksilver mine in the State.

Colusa County: *Central*, Sulphur Creek district; cinnabar occurs in seams in altered serpentine; idle. *Elgin*, 3 miles north of Sulphur Creek; idle. *Empire*, south side of Sulphur Creek; has a 168-ft. adit, showing a little cinnabar. *Ferris* (*Oriental*), *Manganite*, and *Rathburn*; idle; the last has a shaft 130 ft. deep, and drifts 145 ft. long. *Buckeye*, southwest of the village of Sulphur Creek; has a shaft 200 ft. deep, and is said to have produced good ore in the past. The Sulphur Creek district lies about 20 miles southwest of Williams, the nearest railway station.

Lake County: *Abbott*, near head of Grizzly Creek, and 28 miles southwest of Williams; cinnabar occurs in seams in shale near contact with serpentine; opened by a cross-cut adit 500 ft. long, and a drift 700 ft. along the ore belt; the ore-shoot, which is 2 to 18 ft. thick, is followed from this level 600 ft. upwards on a slope of 30 to 40°, connecting with an upper tunnel; the reduction works, ¼ mile north of main adit, comprise a 10-ton Knox & Osborne coarse-ore furnace, six small iron water-jacketed condensers, a blower run by a 20-ft. overshot wheel, ore-sheds, etc.; the fine ore is made into "adobes" and treated as coarse ore. *American*, 7 miles west of Middletown; idle. *Baker*, 5 miles east of Lower Lake; idle. *Bullion*, 4 miles southwest of Middletown; undeveloped. *Chicago*, 6 miles west of Middletown; the vein is a hard opaline serpentine, interspersed with cinnabar, between walls of green and black serpentine; opened by two adits, 100 ft. and 400 ft. long; at the end of 1895 there were 5000 tons of sorted ore awaiting treatment. *Destinell*, an eastern extension of the Abbott; idle. *Great Western*, 4 miles southwest of Middletown; the vein-matter, 10 to 150 ft. wide, consists of a black, green, or yellow translucent rock, occurring in thin layers, which are separated by thin seams of clay or calcite; cinnabar, associated with pyrites, occurs through this vein; the hanging wall is black "alta,"* and the foot a greenish-gray, highly metamorphosed sandstone; the mine is opened by a shaft 700 ft. deep, and an adit nearly a mile long connecting with the 500-ft. level; the reduction works comprise two furnaces, one for coarse and one for fine ore, with capacities of 25 and 15 tons per day respectively. *Maypole*, 7 miles west of Middletown; has a vein 8 ft. thick, carrying cinnabar in silicified serpentine. *Mirabel*, 4 miles southwest of Middletown; has a vein of hard dark green or black opaline serpentine, carrying cinnabar in seams and incrustations; both walls are

* "Alta" is a layer of attrition products, so-called clays, full of slickensides, and of fragments of rocks more or less rounded by attrition; it is followed by the miners in their search for ore.—*Becker.*

light green serpentine; the ore bodies are 20 to 60 ft. wide; the mine is opened by a main shaft, 425 ft. deep, and levels at 200 ft., 250 ft., 300 ft., and 400 ft. deep; a new shaft 500 ft. southwest from the main shaft is being sunk; the reduction works comprise two large new furnaces of latest type for coarse and fine ore, but only fine ore is treated. *Sulphur Banks*, 10 miles north of Lower Lake. *Uncle Sam*, 10 miles northwest of Lower Lake; idle. *Utopia*, 8 miles east of Lakeport; has a vein of greenish-gray sandstone with seams of serpentine, about 25 ft. thick; hanging wall "alta," and foot-wall serpentine; opened by a cross-cut adit 120 ft. long with a 30-ft. drift in vein, and a cross-cut 150 ft. long with a 50-ft. drift in vein, at the end of which a 60-ft. winze shows good ore. *Wall Street* (*Jewess*), 6 miles west of Middletown; idle.

Napa County: *Ætna*, situated at the head of Pope Valley, 11 miles south of Middletown; opened very extensively, a main tunnel being 4000 ft. long and 350 ft. deep, while two other tunnels are respectively 200 and 138 ft. deeper than the main tunnel; the chief ore supply comes from east of the main tunnel at contact of sandstone and argillite, the ore impregnating the sandstone; the ore-chute is very irregular, but averages 200 ft. in length and 20 ft. in width; associated with the cinnabar are marcasite, pyrite, native mercury, and napalite; the reduction works comprise two furnaces, one of 20 tons capacity for coarse ore, and one of 15 tons for fine ore. *Corona*, at head of Pope Valley, 8 miles south of Middletown; opened by three cross-cut adits, the second 60 ft. below the first, and the third 194 ft. below the second, each of which shows a hard, silicious rock rich in cinnabar; the vein, striking east and west and dipping 55° south, consists of alternate thin strata of sandstone and "alta"; the country rock south of the vein is serpentine, and north is a soft gray sandstone; the reduction plant consists of one 20-ton fine-ore furnace. *Harrison*, 16 miles southeast of Lower Lake; idle. *Manhattan* (*Lake*), 17 miles southeast of Lower Lake; the older workings were closed in 1895, but shortly afterwards a new discovery was made, which has since then been exploited; the reduction plant comprises two 20-ton furnaces. *Oat Hill*, 9 miles south of Middletown; owned by Napa Consolidated Quicksilver Mining Company; the country rock is soft gray sandstone, interstratified with numerous seams of soft "alta"; three veins (the Humboldt, Osceola, and Escape) or zones impregnated with cinnabar are developed; generally the ore occurs in sandstone, but sometimes it is found also in "alta"; the ore bodies are very irregular, and prospecting for them is difficult; opened by tunnels 500 ft., 500 ft., and 1200 ft. long; reduction works consist of two fine-ore furnaces each of 50 tons per day capacity. *Oakville*, 1½ miles south of Rutherford; idle. *Redington* (*Knoxville*), 18 miles northeast of Lower Lake; opened by shaft to depth of 300 ft.; country rock serpentine; vein 15 ft. wide, black chalcedony, with cinnabar and pyrite; the ore, which is very hard, is crushed with Blake crushers, and distilled in a 30-ton fine-ore furnace. *Summit*, 3 miles southwest of Rutherford; idle.

San Benito County: *Gypsy* (*San Benito*), 16 miles northeast of Hollister; opened by adit 900 ft. long; has a 10-ton furnace; idle since February, 1895. *New Idria*, situated at the head of San Carlos Creek, in a mountainous region, 58 miles southeast of Tres Pinos; developed to a depth of 1200 ft. by adits, which altogether have a length of 3000 ft.; the country rock is shale and sandstone; there was in 1896 a good deal of fine ore assaying 1 to 15% cinnabar on the dump,

and some pay ore exposed in the mine; reduction works comprise a 10-ton coarse-ore furnace and a 30-ton furnace for fine ore, the latter erected in 1896.

San Luis Obispo County: The quicksilver claims located several years ago in the Santa Lucia Mountains, northwestern part of this county, remain generally idle. The *Black Hawk* and *Santa Cruz* (*Sunderland*) are the only ones which have been worked during the last two years.

Santa Clara County: *Bradford* and *Treadwell*, 11 miles east of San José; idle. *Guadalupe*, 6 miles east of Los Gatos; formerly a large producer; now idle. *Hillsdale*, ½ mile west of Hillsdale; idle. *New Almaden*, 15 miles south of San José; the occurrence of the ore bodies in this famous mine is characterized by great irregularity in size and grade, which has caused a great variation in output; the ore occurs in shoots and bunches in a brecciated zone along a contact of serpentine and shale, the vein-matter being chiefly serpentine, generally richer near the hanging wall, and becoming poorer with depth as the vein-matter becomes harder; the mines are now developed to a depth of 2500 ft. below the summit of the mountain, and the underground workings are said to be more than 70 miles in length; there are six shafts which are equipped with hoisting works, more than that number of abandoned shafts, and numerous adit levels; the reduction works comprise eight furnaces, of which five are now in operation. *North Almaden*, 9 miles southeast of San José; idle. *Wright*, 20 miles south of San José; idle.

Siskiyou County: *Barton* and *Lange*, situated on Klamath River, 4 miles below Oak Bar; cinnabar occurs in a bed of dark gray-green sandstone, lying between strata of black shale. *Siskiyou*, 15 miles west of Cole's Station; opened by several hundred feet of workings and equipped with a 10-ton furnace.

Sonoma County: *Black Bear*, on Little Sulphur Creek, 2 miles south of the Geysers; located on a zone of serpentine carrying cinnabar and native mercury; formerly a producer, but property now in state of decay, and mine workings caved. *Cinnabar King*, on Pine Mountain, 12 miles east of the Geysers; a vein of sandstone, with seams of "alta" and yellow clay, lying between soft gray sandstone (hanging) and green-gray serpentine (foot); opened by adit levels; an important ore body was discovered in 1895, and 2000 tons of selected ore produced from same, a furnace to treat which was built in 1896. *Great Eastern*, 4 miles north of Guerneville; a vein of hard silicified serpentine, 20 to 80 ft. thick, separated by 6 to 8 ft. of "alta" from the hanging wall (serpentine); the foot-wall is sandstone; opened by shaft 360 ft. deep, and levels at 140 ft., 220 ft., and 360 ft.; reduction works, one fine one coarse ore furnace, each of 15 tons capacity. *Rattlesnake*, 4 miles south of the Geysers; idle; the ore of this mine contains a large percentage of native mercury. *Red Cloud*, 5 miles southeast of the Geysers; idle. *Sonoma*, adjoining the Red Cloud; formerly a large producer; now idle, and workings caved.

Stanislaus County: *Summit*, situated on the divide at the south end of Red Mountain; cinnabar occurs in seams in crushed serpentine, sandstone, and shale; has a 5-ton furnace.

Trinity County: *Altoona*, 16 miles west of Castella and 22 miles northeast of Trinity Center; the ore, occurring in a porphyritic matrix, adjacent to serpentine, averages 4% Hg; a reduction plant was erected in 1895, consisting of a 30-ton

Knox furnace, 32½ by 30 by 68 ft. and 12 Knox & Osborne iron double condensers. *Integral*, adjoining the Altoona; idle.

Yolo County: *Reed*, situated on Davis Creek near the western boundary of the county; this mine was formerly worked extensively, and has recently been reopened after idleness for several years.

The quicksilver production of California by counties in 1894 and 1895 was as follows: (*a*)

Year.	Lake.		Napa.		San Benito.		Santa Clara.		Colusa.	
	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.
1894	12,471	$382,954	9,705	$298,016	1,005	$30,861	7,235	$222,169	.....	.....
1895	12,856	465,074	9,318	372,500	1,100	36,000	7,050	253,800	1	$40

Year.	San Luis Obispo.		Sonoma.		Stanislaus.		Trinity.		Total.	
	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.	Flasks.	Value.
1894	.....	.....	.....	.....	.....	.....	.....	.....	30,416	$934,000
1895	20	$800	1,813	$70,707	20	$800	3,926	$137,410	36,104	1,337,130

(*a*) Flask = 76.5 lbs.

QUICKSILVER MINING IN OTHER COUNTRIES.

The foreign countries important as producers of quicksilver are Spain, Austria, Italy, Russia, and Mexico. The production of Italy fell off in 1896, while that of Austria remained approximately the same as in 1895. No statistics are yet to hand for Russia and Mexico.

QUICKSILVER PRODUCTION OF THE WORLD, IN METRIC TONS.

Year.	Austria.	Italy.	Mexico.	Russia.	Spain.	United States	Totals.
1891	570	330	.....	324	1,665	794	3,683
1892	542	325	.....	343	1,682	971	3,863
1893	511	273	.....	201	1,672	1,047	3,704
1894	521	258	268	196	1,653	1,056	3,952
1895	535	199	160	152	1,484	1,179	3,709
1896	520	132	.....	.....	1,513	1,151	.....

GENERAL QUICKSILVER STATISTICS.(*a*)

	1887	1888	1889	1890	1891	1892	1893	1894	1895	1896
	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)	Flasks. (*b*)
Shipments from Spain to London.	51,011	52,243	49,778	50,202	47,993	47,321	44,670	42,414	40,409	40,999
Shipments from Italy to London.	7,400	10,460	10,498	12,470	10,440	6,765	6,680	8,700	5,775	3,800
Total.	58,411	63,703	60,276	62,672	58,433	54,086	51,350	51,114	46,184	44,799
Shipments from London.	64,049	45,806	55,366	57,702	63,143	46,055	42,265	41,046	37,937	31,273
Austrian production	14,000	14,000	14,000	14,000	15,000	15,000	15,000	15,000	15,000	15,000
Californian "	33,760	33,250	25,650	22,925	22,904	25,554	30,164	30,183	36,104	31,000
Maximum price of Spanish.	$56.25	$52.50	$48.75	$51.88	$45.00	$38 75	$34.375	$33.75	$36.875	$36.25
Minimum " " "	32.50	33.75	36.87	44 37	36.25	30.25	30.62	27.50	31.87	32.12

(*a*) From W. T. Sargant & Co.'s Annual Metal Circular.
(*b*) American, Russian, and Italian flask, 76 5 lbs. (34.7 kgs.); Mexican flask, 75 lbs. (34.03 kgs.); Spanish flask, 76 lbs. (34.5 kgs.).

CONSUMPTION OF QUICKSILVER IN THE UNITED KINGDOM.

Year.	Imports.	Exports.	Consumption	Year.	Imports.	Exports.	Consumption
	Flasks.	Flasks.	Flasks.		Flasks.	Flasks.	Flasks.
1891.....	61,884	55,884	6,000	1894.....	51,250	43,598	7,652
1892..	56,203	49,939	6,264	1895...	49,658	36,826	12,832
1893.....	51,797	37,478	14,319	1896.... ...	44,799	31,273	13,526

Brazil.—Large deposits of ore assaying 0.1 to 2.0% are known to exist near Nazareth, and some of them are now being developed by Americans, who have already erected a reduction furnace.

British Columbia.—Quicksilver mines have been opened in the Kamloops District, Yale Division. The most important undertaking is the Cinnabar Mining Company, Ltd., which acquired a group of claims on the north side of Kamloops Lake in 1895. Two retorts were erected and 100 flasks of quicksilver were produced, which exhausted the high-grade ore that was on hand. The process was not adapted to the treatment of low-grade ore. A modern coarse-ore furnace, of 25 tons per day capacity, is now in course of construction. The Rose Bud and Yellow Jacket mines have been explored rather extensively. The country rock is picrite, a dark green, highly acidic rock of Tertiary volcanic origin. It is intersected by fissures of varying width, the smaller filled with a dolomitic gangue and containing the high-grade ore, and the larger with a rock graduating from tufa to rhyolite, much of which contains disseminated, minute crystals of cinnabar.

Mexico.—The Huitzuco mines are situated 5 hours' ride east of Iguala in the State of Guerrero. The country rock is limestone. The ore deposits are found at the summits of small hills or mounds as a soft, sandy, clay-like filling in crevices often radiating irregularly like a star from a central fissure or pipe. The average grade of the ore is about 1.7% Hg. The mines have produced as much as 5000 flasks per annum, but recently have fallen off. The metallurgical methods are crude, and according to Mactear no more than 50% of the mercury in the ore is recovered.

The Guadalcazar mines 70 miles northeast of San Luis Potosi, in the same State, are an occurrence of veins and pockets of cinnabar in gypsum. The ore averages 2 to 3.5% Hg. These mines are operated by an English company, which has installed good reduction works. In distilling 2¼% ore a yield of about 85% of the mercury is realized.

Spain.—In Asturias the Sociedad minera "El Porvenir" produced 7065 tons of mineral, the Union Asturiana 4471, and La Soterraña, in the mines of Muñon Cimero, 670. In Granada 800 tons were produced.

In the Almaden district 16,076 tons of ore were beneficiated, which produced 41,330 flasks (1,426,188 kilos) of mercury, or 8.87%. The Porvenir Company distilled 3800 tons of ore and 442 tons of arsenical residues (calcined), which produced respectively 1343 flasks (46,342.90 kilos), and 358 flasks (12,353.50 kilos), a total of 1701 flasks, which was 441 more than in 1895. The average yield of the ore was 1.21% and of the residues 2.73%. The Union Asturiana produced 738 flasks, and the Soterraña 105. The total production of mercury in Spain in 1896 was 43,884 flasks, or about the same amount as in 1895.*

* Roman Oriol, Revista Minera, Metalurgica, y de Ingenieria, Feb. 8, 1897.

THE RARE ELEMENTS.

By L. M. Dennis.

The more important contributions to our knowledge of the chemistry of the rare elements that have appeared during the year 1896 are summarized below, and are arranged under the same plan as was followed in The Mineral Industry, Vol. IV. Of chief interest, from the standpoints of both pure and applied chemistry, are the researches of Henri Moissan on the metallic carbides, and the advances that are being made in the chemistry of the rare earths and in their application to incandescent gas lighting. These two subjects, therefore, have been described in considerable detail.

Boron.—H. N. Warren describes (*Chem. News*, Vol. LXXIV., p. 64) an improved method for the preparation of boron. The process is used by the Boron Carbon Battery Company, and their entire output is consumed in the manufacture of the boronized graphite used in the construction of their primary batteries.

Boric acid is calcined to free it from water and convert it into the trioxide, and is then ground to a fine power and intimately mixed with a suitable proportion of sodium magnesium chloride. This mixture is then rapidly heated to fusion, and metallic sodium, in large pieces, is added from time to time, the mass being well stirred after each addition. The reaction is not violent, but a large amount of heat is evolved. Metallic magnesium is first set free ($MgCl_2+2\ Na=2\ NaCl+Mg$), and this at once attacks the boron trioxide, forming magnesium oxide and free boron ($3\ Mg+B_2O_3=3\ MgO+2\ B$). The pasty mass thus formed is detached from the crucible and thrown into an excess of hydrochloric acid. Everything dissolves except the boron, which is collected on a filter, washed and dried. After further heating in special gases, the boron becomes peculiarly active, its surface action, after it is impregnated with graphite, resembling, to a certain extent, that of platinum black. Carbons thus prepared, about 2 by 5 in., when arranged as an outer or positive element of a battery, and charged with chromic acid solution, have been found sufficiently effective to evolve over 5 ampères for each carbon. Small cells thus arranged, equal in size to a quart Bunsen cell, give over 240 ampère-hours, or 25 ampères for 12 hours constant. The best depolarizers for these carbons are a saturated solution of ferric chloride with oxide of manganese, chromic acid, or acidulated potassium permanganate.

It is stated that there is great demand for these carbons, and that the company consumes several pounds of boron per week in their manufacture.

Warren also gives a method (*Chem. News*, Vol. LXXIII., p. 262) of preparing boron bronze, which he says is denser and more durable than ordinary aluminum bronze. It can be easily melted and cast, a merit not possessed by the aluminum bronze.

Glucinum.—The *Electrical Review*, of London, states that the price of metallic glucinum is about $18 per lb., and that, if the supply is sufficient and the methods of production are improved, the metal may find useful application in the electrical industries.

Lithium Carbide.—H. Moissan (*Annales Chim. Phys.*, 7, Vol. IX., p. 302; *Comptes rendus*, Vol. CXXII., p. 362). A carbide of the formula Li_2C_2 was obtained by heating in the electric furnace a mixture of lithium carbonate and carbon in the proportions indicated by the equation $Li_2CO_3+4\ C=Li_2C_2+3\ CO$.

The mixture is placed in a carbon tube closed at one end and subjected for 10 to 12 minutes to the arc obtained by a current of 350 ampères and 50 volts. If a current of 950 ampères and 50 volts be used the action should be allowed to continue only 4 minutes. The end of the operation is determined by the appearance of metallic vapors. Prolonged action or too high a temperature seems either to decompose or volatilize the carbide.

Lithium carbide thus prepared is a colorless, transparent, crystalline mass, which under the microscope is seen to consist of brilliant crystals, readily decomposing in the air. It has a sp. gr. of 1.65 at 18° C., is easily powdered, will not scratch glass, and is a most powerful reducing agent. Placed in a current of fluorine or chlorine it takes fire at once, at ordinary temperature, burning with a brilliant light. In bromine or iodine vapor the carbide must be gently heated to produce the same phenomena. When heated to a dull red heat the carbide behaves similarly in oxygen, and in the vapors of sulphur and of selenium.

Fragments of the carbide thrown into cold water decompose at once with a brisk evolution of gas. This reaction becomes violent when hot water is used. The gas given off was found to be pure acetylene. One kilogram of the carbide yielded 587 liters of acetylene gas.

Lithium carbide of the same composition can also be prepared by heating metallic lithium in a current of acetylene gas.

Molybdenum.—H. Moissan (*Annales Chim. Phys.*, 7, Vol. IX., p. 238). Pure molybdenum dioxide (MoO_2), obtained by the ignition of pure ammonium molybdate, is mixed with carbon in the proportion of 300 gms. of the oxide to 30 gms. of carbon. This mixture, in which the oxide is in great excess, is subjected for 6 minutes to the arc produced by a current of 800 ampères and 60 volts, precaution being taken to avoid complete fusion of the mass, as otherwise the melted molybdenum will unite at once with the carbon of the crucible in which the operation is conducted. This reaction is sure to take place if the heating is continued beyond 6 minutes. Properly prepared the molybdenum obtained is absolutely free from carbon. The pure metal has a sp. gr. of 9.01, and is as malleable as iron, and, like the latter, can be welded, filed, and polished with ease, and takes up carbon by cementation. It is, however, less acted on by water and carbon dioxide, but is readily oxidized when heated. It takes fire in a cur-

rent of oxygen between 500° and 600° C., burning with great brilliancy even when the source of heat is removed. The powdered metal is acted on by fluorine in the cold, by chlorine at a low red, and by bromine at a cherry-red heat. Gently heated with phosphorus pentachloride a volatile chloride is formed, turning a beautiful blue in air. This reaction applies equally well to the majority of molybdenum compounds, and can be used as a rapid test for determining the presence of this element in various substances.

Nitrogen fails to form any nitride at 1200° C. Molybdenum containing carbon can be refined or lowered in carbon by heating in contact with the dioxide at a temperature far below that of its point of fusion.

J. Férée (*Comptes rendus de l'Académie des Sciences*, Vol. CXXII., p. 733) describes the electrolytic preparation of an amalgam of molybdenum. From this amalgam he obtains molybdenum by distilling off the mercury in a vacuum at a low temperature. The properties of the metallic molybdenum thus prepared are described as follows:

The metal is pyrophoric, taking fire in the air and forming oxides of molybdenum which are partially volatilized by the heat liberated. It loses this property when it has been heated above 400° C.

In a current of sulphur dioxide this molybdenum becomes incandescent, absorbing the gas and forming molybdenum sulphide and oxides.

Nitrogen, carbon dioxide, and hydrogen sulphide seem not to act upon the metal, either at ordinary temperature or at a low heat. Carbon monoxide, however, is rapidly decomposed when the molybdenum is gently heated: the metal becomes red hot and absorbs the oxygen of the gas, carbon being deposited.

If the metal has not been previously heated it becomes incandescent in nitrogen tetroxide, the nitrogen and oxygen being absorbed simultaneously; there are formed oxides of molybdenum and a molybdenum nitride.

Molybdenum Carbide.—Heated with an excess of carbon in the electric furnace (800 ampères and 50 volts), MoO_2 unites to form a definite carbide, Mo_2C. This compound is obtained as a fused ingot with brilliant white fracture, whose density is 8.9. It can be easily crushed on an anvil and shows the presence of distinct crystals. The compound made as above always contains graphite.

Titanium.—H. Moissan (*Annales Chim. Phys.*, 7, Vol. IX., p. 229). By means of the electric furnace there are formed four different bodies from a mixture of titanic acid and carbon, according to the conditions and strength of current used.

1. Blue protoxide of titanium.
2. Fused titanium nitride.
3. Metallic titanium.
4. Titanium carbide.

1. The protoxide is obtained in the form of a dark indigo-blue mass of crystals, when an arc of but little intensity is used.

2. Heating with a current of 300 ampères and 70 volts there results a bronze-colored, fused mass, hard enough to scratch the ruby with ease and to slowly polish diamonds. It is crushed with difficulty, and has a sp. gr. of 5.18; its formula is Ti_2N_2.

3. To obtain titanium free from nitrogen it is necessary to employ a current of

such a strength that the temperature shall be above that of the decomposition of the nitride. This is accomplished by means of a dynamo of 100 horse power. Even with a current of 2200 ampères and 60 volts it was found impossible to completely fuse the mass. Titanium is by far the most infusible substance yet met with by Moissan. The purest sample of the element still contained about 2% carbon, and all attempts to diminish this percentage have thus far failed. In this state it is exceedingly hard, but very brittle and easily powdered, and has a density of 4.87.

4. With a current of 1000 ampères and 70 volts, and an action of 10 minutes, TiC is obtained in well-fused ingots with crystalline fracture, or as a mass of crystals whose density is 4.25. This carbide remains undecomposed by steam even at 700°.

Joseph Barnes, in an article in the *Journal of the Society of Chemical Industry*, Vol. XV., p. 420, states that in 1885 he discovered that titanium oxide could be applied as a mordant to cotton, in the same manner as alumina, and in 1887 he exhibited samples of cotton yarn dyed by means of titanic oxide mordant. He has now found that titanic oxide is also an excellent mordant for animal fiber; and he calls attention to the fact that the widespread occurrence of titanium in nature renders feasible its commercial use on a large scale. For the mordanting of wool Barnes uses titanium chloride, and he prevents the precipitation of meta-titanic acid by adding a soluble tartrate to the solution. He prepared this "tartrate" mordant as follows: Twenty gms. of anhydrous titanium chloride was mixed with 80 gms. of cream of tartar and 50 c.c. of water. The mixture was then evaporated on the water bath to a clear, viscid residue which, upon cooling, weighed 113 gms. From 20 to 25 gms. of this mordant was used for each 100 gms. of wool, and the mordanting bath was kept at the boiling point for about two hours. The dyeings were generally done with an addition of 3 c.c. of acetic acid per liter of dye bath.

The following is a list of colors thus obtained on wool:

Alizarin gives a deep maroon.

Alizarin orange gives a bright scarlet.

Cœruleïn gives a dark green, yellower than with a chromium mordant.

Alizarin blue gives a blue, redder than with a chromium mordant.

Logwood gives a deep black.

Tannic acid gives a deep yellow.

Salicylic acid gives a sulphur yellow.

Each of these colors except the last withstands the action of dilute mineral acids and soaping. Even the logwood black and tannic acid yellow, after being soaked in dilute hydrochloric acid at 3° Tw. for one hour, and then rinsed and soaped, suffered to a scarcely perceptible degree.

"The behavior of the mordanted cloth in the alizarin dye bath is very remarkable; before the final maroon shade is developed the cloth assumes a brilliant red color, and if it is taken out at this stage and dried it will be found to be nearly the same color as that produced by alizarin orange. It is in this condition sensitive to acids, which turn it to a dull brown color; washing and heating with water turns the brown color to a maroon. The red color withstands the action of the soap bath, and after this treatment is much less sensitive to acids. When

the red color, before soaping, is heated with distilled water it gradually assumes a maroon shade. At first it occurred to me that this difference in color was owing to a difference in ratio between the coloring matter and the mordant; but dye tests made with varying amounts of coloring matter and the same amount of mordant gave gradations in depth with little variation in tone.

"Oxide of zirconium also has an attraction for coloring matters, and when wool is boiled in a solution of zirconium sulphate it becomes mordanted with the oxide, and will yield with alizarin a color very like the one produced on a chromium mordant.

"Cerium does not appear capable of yielding a mordant to wool; when wool is boiled in a solution of a cerium salt it will not take up any color in the dye bath."

Tungsten.—H. Moissan (*Comptes rendus,* Vol. CXXIII., p. 13) has obtained pure tungsten by reducing tungstic acid by carbon in the electric furnace. A mixture of 800 gms. tungstic acid and 80 gms. of carbon (from sugar) is heated in the furnace with a current of 900 ampères and 50 volts. An ingot is thus obtained of which the upper part is well fused and the lower very porous and touching the sides of the carbon crucible only at a few points. The upper part of this ingot yields pure tungsten. A sample thus prepared was submitted to M. Deslandres for spectroscopic examination and was pronounced by him to be of great purity.

Metallic tungsten can be welded like iron when heated, even if far below its melting point. It is also easily filed, and when free from carbon will not scratch glass. Like iron, tungsten takes up carbon by cementation and then becomes hard. The purest sample prepared had a sp. gr. of 18.7, and was without action on the magnetic needle; it contained 99.87% tungsten, and was free from carbon; its melting point is above that of chromium and molybdenum.

Tungsten carbide, W_2C, is formed when tungstic acid is heated with an excess of carbon in the electric furnace. This carbide has an iron-gray color, and is so hard as to scratch corundum. It is more easily attacked by reagents than the pure metal. Its sp. gr. at 18° is 16.06. In oxygen it burns at 500° C., forming tungstic acid and carbon dioxide. Nitric acid dissolves the carbide readily, but other acids act upon it slowly.

Uranium.—H. Moissan (*Comptes rendus,* Vol. CXXII., p. 1088). Three methods were used in preparing this metal.

1. Reduction of the double chloride of sodium and uranium by metallic sodium.

2. Electrolysis of the double chloride.

3. Reduction of U_3O_8 by carbon in the electric furnace.

1. The double chloride is easily prepared by passing vapors of uranium chloride over sodium chloride in a combustion tube heated to low redness. The most satisfactory method of accomplishing this is to place uranium carbide at one end of the tube and sodium chloride at the other; a current of chlorine is passed over the heated uranium carbide, forming uranium chloride, which, being volatile, is carried along to the sodium salt and unites with it, forming, when cold, a crystalline mass of apple-green color. This melts at 390° and is soluble in cold water but is dissociated by alcohol.

Reduction of this double salt was accomplished by means of a heavy iron cylinder, closed by screwing in a plug. The cylinder was charged by placing alternate layers of the double salt and freshly cut sodium. When charged the cylinder was heated for 25 minutes in a very hot wood fire, the heat of the reaction being such that the crucible was raised to a cherry-red in a few seconds. After cooling, the metallic sodium was removed by 96% alcohol and the residue washed rapidly with water, alcohol, and finally ether. The best sample obtained contained sodium and gave for uranium (weighed as U_3O_8) 99.40%.

2. The double chloride of sodium and uranium can be easily electrolyzed. Spongy uranium, inclosing small crystals of the metal, is deposited on the negative electrode. In order that the reaction shall be regular, a difference of potential of 8 to 10 volts is sufficient. Moissan used a current of 50 ampères. The double chloride is usually kept fluid by its own heat of reaction. It is necessary that the vessel in which the reaction takes place be closed and that a steady current of hydrogen be passed through to exclude all air, since nitrogen unites easily with uranium. The metal obtained in this way is crystalline, the crystals being often 1 mm. on a side. If iron electrodes are used there is formed a silver-white alloy of uranium and iron which has a very close and fine grain and can be easily filed. Electrolytic uranium gave on analysis 99.27% U.

3. U_3O_8 (prepared as below) is mixed with carbon in the proportion of 500 gms. to 40 gms. carbon. This mixture is placed in a carbon crucible and heated 7 to 10 minutes by 800 ampères and 405 volts. A fused ingot weighing about 350 gms. is obtained. If the heating has been properly done the resulting uranium will contain but little carbon; under long-continued heating the metal takes up more and more carbon, and finally becomes converted into the carbide U_2C_3. In order to prevent the formation of a nitride it is advisable to carry on the reaction in a closed tube. This is accomplished by placing the crucible within a larger one, lined with a titaniferous mixture, and luting on the cover.

To obtain U_3O_8, uranium oxide of commerce is dissolved in nitric acid, evaporated and recrystallized. The crystals are extracted with ether, which dissolves the uranium and leaves foreign salts. The ether extract is mixed with an equal volume of water, evaporated on the water bath, and the residue dissolved in water and again evaporated. This residue is then ignited, yielding a yellow oxide. This oxide, when re-ignited in a Perrot furnace for two hours, becomes green in color and corresponds to the formula U_3O_8.

The metal obtained by the electric furnace contained: Uranium, 99.52%; carbon, 0.005%; slag, 0.421%.

Pure uranium is perfectly white, with less of a bluish tint than iron. It can be easily polished, does not scratch glass, and takes up carbon by cementation, it being then possible to temper it. If free from iron it is not magnetic. A yellowish color which the metal may possess is due to the presence of nitrogen.

Finely powdered uranium takes fire in fluorine, burning with brilliancy, and forming a volatile, green fluoride. Chlorine unites with it at 180° C., bromine at 210°, and iodine at 260°; in each case the reaction is complete.

Hydrochloric acid gas unites with it at a low red, forming a green chloride, stable in solution. The finely powdered metal burns in oxygen at 170° C., and is acted on by sulphur at 500° C. Nitrogen converts it into a nitride at 1000° C.

If ammonia gas be passed over the metal at a low red heat, hydrogen is given off and a nitride is formed. Water is slowly decomposed at ordinary temperatures and more rapidly at 100° C. Like iron, the metal is readily oxidized by water, particularly if carbon dioxide is present.

Uranium Carbide.—H. Moissan (*Comptes rendus*, Vol. CXXII., p. 274; *Annales Chim. Phys.*, 6, Vol. IX., p. 323). U_3O_8, obtained as above, is mixed with carbon in the proportion of 500 gms. oxide to 600 gms. carbon. With a current of 900 ampères and 50 volts the mixture required 8 to 10 minutes' action. About 5 minutes after the current is turned on brilliant sparks escape from the furnace; these soon disappear and there remains in the crucible a liquid carbide of uranium, which is allowed to cool in the furnace.

This carbide consists of dense metallic fragments with crystalline fracture, and closely resembles metallic bismuth, both in color and in form. It has a formula of U_2C_3 and a sp. gr. at 18° of 11.28. Its hardness is such that it scratches glass and quartz, but not corundum. When struck with a hard body sparks are readily thrown off, as is also the case with metallic uranium. When it is powdered in a mortar it is liable to take fire, the combustion continuing through the whole mass.

Fluorine fails to act in the cold, but at a gentle heat a brilliant reaction takes place.

Chlorine unites at 350° C., raising the carbide to incandescence.

Nitrogen unites at 1100° C., but the reaction is not complete.

Concentrated acids (except nitric acid) attack this body but slowly unless warmed.

Placed in water, decomposition takes place slowly, the reaction increasing in rapidity as the temperature is raised. The gas given off by two different samples had the following composition:

	(I.) Per Cent.	(II.) Per Cent.
Acetylene	0.17	0.72
Ethylene	6.77	5.16
Methane	78.05	80.60
Hydrogen	15.01	13.52

It is a remarkable fact that these gases account for only one-third of the carbon contained in the carbide. The other two-thirds, which is found in the water used for the decomposition, and which can be extracted by means of ether, consists of liquid and solid hydrocarbons. Examination of these shows that they distil between 70° and 100° C., leaving a tarry residue in the flask. They contain unsaturated compounds, for alkaline silver nitrate is easily reduced, with the formation of a metallic mirror. This reaction cannot be due to an aldehyde, since the color is not restored to a solution of fuchsine containing sulphur dioxide. The presence of hydrogen in the gases may arise from a secondary action, due to a hydrated oxide of uranium, as it is known that there exists a suboxide of uranium having the power to decompose water.

Vanadium.—H. Moissan (*Comptes rendus*, Vol. CXXII., p. 1297). For the preparation of metallic vanadium and of vanadium carbide the author started with vanadic acid, which he obtained from a vanadium-bearing coal, the ashes of which contained 38% vanadic acid. The vanadium was extracted by treating

the ashes repeatedly with nitric acid to which small amounts of hydrochloric acid were added from time to time. The residue was treated with water. All the liquids thus obtained were united and evaporated to dryness and the residue extracted with water containing ammonia. The solution of ammonia vanadate thus obtained was precipitated by nitric acid and the crude vanadic acid separated and converted into vanadyl chloride by the method of L'Hôte (*Annales Chim. Phys.*, 6, Vol. XXII., p. 407), and this in turn fractionally distilled. The purest fractions obtained were decomposed by water, yielding a pure vanadic acid, which was then dried and converted into the pentoxide.

All attempts to obtain pure vanadium free from carbon failed on account of the fusibility of vanadium pentoxide and because of its great affinity for nitrogen.

The best results were obtained by heating a mixture of vanadium pentoxide and carbon, the former in excess, for only 2 minutes with a current of 1000 ampères and 60 volts, the precaution being taken to keep a constant current of hydrogen passing through the tube. Samples thus made contained from 4.4% carbon to 5.3%. If the heating was continued for 3 minutes the carbon rose to 7.42%. With a current of 900 ampères and 50 volts the carbon content averaged 9.6%. When heated in theoretical proportions for 5 minutes the carbon was found to increase to from 10 to 16%.

Vanadium containing 5% of carbon is of a white color, with brilliant metallic fracture and a sp. gr. of 5.8 at 20° C. It is permanent in air, but burns in oxygen at a red heat. Nitrogen unites with ease, and it is more easily acted on by alkalies and acids than is the definite carbide described below.

Vanadium Carbide.—If the heating of the mixture described above be continued for 9 or 10 minutes (900 ampères and 50 volts) an ingot is obtained consisting of a definite carbide of vanadium of the formula VC. Its melting point is a little above that of molybdenum; its sp. gr. is 5.36, and its hardness such that it scratches quartz with ease. If care be used it is possible to obtain it in beautiful, well-formed crystals. Chlorine decomposes this compound at 500° C., forming a liquid chloride easily volatilized. Vanadium carbide burns brightly in oxygen at a low red heat, but sulphur has no action at the temperature of softening glass. Nitrogen and ammonia unite at a red heat, forming a nitride. Hydrochloric and sulphuric acids have no effect, but nitric acid acts in the cold.

Alloys of Vanadium.—Notwithstanding its high melting point vanadium readily forms alloys, a few of which were made. *With Iron.*—A mixture of oxide of iron, vanadium pentoxide and carbon, heated in the electric furnace, furnished an ingot of uniform texture, bluish-white color, and crystalline fracture. The alloy can be easily filed, and was found to contain: Iron, 72.96%; vanadium, 18.16%; carbon, 8.35%. *With Copper.*—In the same manner an alloy of copper and vanadium was prepared, using copper oxide. The resulting bronze-colored, malleable ingot contained 96.52% copper and 3.38% vanadium. It is easily filed, but is harder than copper. *With Aluminum.*—In order to obtain an alloy it was found necessary to throw a mixture of vanadium pentoxide and finely powdered aluminum into a crucible containing molten aluminum. The liquid metal was well stirred and the cooled mass proved to be a soft, malleable alloy contain-

ing 2.5% vanadium. *With Silver.*—All attempts to form an alloy with silver proved futile. The ingots obtained were formed of two layers, the upper one of metallic vanadium free from silver and the lower one of silver containing no trace of vanadium.

M. K. Hélouis, in the *Bull. de la Soc. d'Encouragement,* for June, 1896, gives some interesting details concerning the alloys of vanadium and their applications. The following abstract of his article appeared in the *Journal of the Society of Chemical Industry,* Vol. XV., p. 657.

"At a height of about 4800 m., on the elevated plains of the great chain of the Andes, an anthracite mine is worked, comprising two parallel, inclined beds of vanadium-bearing coal, from 2 to 3 m. thick, and 1400 m. long. The beds are separated by calcareous deposits. This coal burns readily, leaving 2% of ash, or 20 kgms. per ton, having an umber color, and yielding 14 to 25% of vanadium as tetroxide, a little zirconia, notable traces of platinum, besides about 16 kgms. per ton of silver. The author has devised a process whereby vanadic acid and vanadates may be obtained at small cost from this ash, and anticipates from this source the probability of technical applications, not only in relation to dyeing, but to metallurgy.

"Vanadic acid is reduced by aluminum at a red heat, by successive stages, to V_2O_4; but reduction to metal requires a temperature of about 1700° C., when the evolution of heat from the formation of the alloy is liable, without certain precautions, to give rise to violent explosions, the light given out being comparable to that of the electric arc. Alloys may be obtained containing from 1 to 40% of vanadium, out of which the aluminum may be dissolved, leaving the vanadium in powder or in brilliant scales, which latter are not attacked by hydrochloric acid or alkalies, and scarcely by concentrated sulphuric acid, but dissolve in nitric acid with production of vanadic acid, and when thrown on a red-hot plate they inflame with projection of brilliant sparks.

"Moissan has observed that the Swedish iron from the magnetic oxide of Mount Taberg, so remarkable for its ductility, contains vanadium; and also that certain slags from Staffordshire iron furnaces, producing very ductile iron, contain a considerable proportion of vanadic acid. For application in the metallurgy of iron, alloys of vanadium are obtained by reduction of the oxides, by aluminum in powder, the alloys named being aluminum-vanadium, iron-aluminum-vanadium, iron-nickel-vanadium, and iron-chromium-vanadium. Alloys of copper-aluminum-vanadium have also been obtained for experiments with bronze. There is a special alloy of aluminum-vanadium, remarkable for its sonorous quality, which is applicable in making bells and certain musical instruments.

"A specimen of dephosphorized steel, tested after being melted in a plumbago crucible, gave breaking charge, 96 kgms. per sq. mm.; elongation, 2.3%. The same, similarly melted with 1% of vanadium, gave breaking charge, 109 kgms.; elongation, 7.53%; limit of elasticity, 78.7. A second experiment was made, a magnesia-lined crucible being used, and 0.5% of vanadium added, which gave on trial, breaking charge, 66 kgms.; elongation, 16%. A third experiment, using a magnesia-lined crucible, but adding 1% of vanadium, gave an alloy having breaking charge, 97 kgms.; elongation, 14%. The same forging, annealed, gave breaking charge, 71 kgms.; elongation, 20%. This last specimen, though very

soft, took a remarkably high temper. A specimen of soft iron, with resistance 38 to 39 kgms., and elongation 19%, after fusion with iron-vanadium equal to 0.5% of vanadium, gave a bar, forged without annealing, having breaking charge 61.25 kgms., and elongation 12%, which, after annealing, became respectively 53 kgms. and 32% elongation. This last specimen showed very great malleability and ductility. An aluminum-bronze was also tested, containing 8% and 1%, respectively, of aluminum and vanadium, which gave breaking charge, 71 kgms., elongation, 12.5%.

"Nickel oxide is reduced by aluminum powder, and nickel-aluminum alloys containing from 25% to 90% of nickel have been obtained. Aluminum added to fused commercial nickel instantly reduces the oxide present with a strong reaction. Nickel, chromium, tungsten, and molybdenum are easily extracted from their alloys with aluminum, and nickel thus obtained becomes pyrophoric at over 100° C. Great care is necessary in preparing molybdenum-aluminum, since a mixture of molybdenum oxide and aluminum powder, when strongly heated, explodes with a report like a gun-shot and projection of the metal. Iron-molybdenum may be prepared without danger. An alloy containing 21% of molybdenum is described as being hard enough to scratch glass. Steel, melted with 0.5% of molybdenum, acquires extraordinary hardness."

Zirconium Carbide.—H. Moissan and Lengfeld (*Comptes rendus,* Vol. CXXII., p. 651). The zirconium used in this work was prepared from native zircon as follows: The finely powdered mineral was intimately mixed with carbon, placed in a carbon crucible and heated for 10 minutes in the arc produced by a current of 1000 ampères and 40 volts. Silicon, being much more volatile than zirconium, escapes, and there remains a mass of metallic appearance, consisting of zirconium carbide and a very little silicon. The impure carbide thus obtained was submitted to a current of chlorine at a low red heat, and the resulting chloride was dissolved in boiling concentrated hydrochloric acid, from which solution $ZrCl_4$ separates in an almost pure condition. The precipitated chloride was washed with concentrated hydrochloric acid, dissolved in water, evaporated to dryness, dissolved again in water, and the clear solution precipitated with ammonia. The precipitated hydroxide was ignited in a muffle and an oxide of zirconium thus obtained which proved to be free from iron and silicon.

The oxide prepared as above was mixed with carbon, the mixture moistened with oil, and the mass strongly compressed into cylinders, which were then gently ignited. These small cylinders were then placed in a carbon tube and heated in the electric furnace for 10 minutes. A current of 1000 ampères and 50 volts was used. The zirconium oxide fuses and part of it is volatilized. It is therefore important that a very high temperature be used at once, and it is only in the hottest part of the tube that a small ingot of zirconium carbide is obtained. It frequently happens that the carbon tube, being unable to withstand the enormous heat, becomes fissured or perforated, and in such cases the zirconium carbide will be found to contain calcium carbide, the latter coming from the walls of the furnace.

The ingot has a metallic gray color and has the composition ZrC. It is very hard, scratching glass and quartz with ease, but not affecting the ruby. This compound is permanent in moist or dry air even at 100° C. Fluorine attacks it

in the cold, chlorine at 250°, bromine at 300°, and iodine at about 400° C. In oxygen it burns at a low red heat. Hydrochloric acid, dilute or concentrated, has no effect, even at the boiling temperature. Dilute nitric acid has but little effect, but the concentrated acid attacks it with violence, which is true of nearly all oxidizing agents.

Below will be found a brief summary of the carbides thus far described by Henri Moissan, and of the action of metals toward carbon at high temperatures:

Metal.	Formula.	Sp. Gr.	Color and Appearance.	With Water.	With Chlorine, Temperature of Attack.
Platinum group *a*..					
Gold, bismuth, tin *b*					
Silver, copper *c*.....					
Iron *d*.....					
Lithium.....	Li_2C_2....	1.65	Colorless or white transparent crystalline mass.	Decomposed by water gives pure acetylene....	Cl unites in cold.
Calcium.....	CaC_2....	2.22	Opaque, brilliant crystals, monoclinic.....	Pure acetylene.....	Cl at 245°; Br, 350°; I, 305°.
Barium.....	BaC_2....	3.75	Black crystalline mass....	Pure acetylene.....	
Strontium.....	SrC_2....	3.19	Black crystalline mass....	Pure acetylene.....	
Aluminum	Al_4C_3 ...	2.36	Yellow transparent hexagonal plates	Pure methane.....	Cl at low red; Br, 700°.
Cerium.....	CeC_2	5.23	Homogeneous reddish-yellow crystalline mass....	Acetylene, 76%; methane, 22%; ethylene, 4%; and a small amount of solid and liquid hydrocarbons	Cl at 230°.
Lanthanum	LaC_2....	5.02	Homogeneous crystalline mass, lighter than cerium compound.....	Acetylene, 72%; ethylene, 2%; methane, 28%; also small amounts of solid and liquid hydrocarbons	Cl at 250°; Br, 255°.
Yttrium.....	YC_2.....	4.13	Yellow transparent crystalline mass.....	Acetylene, 72%; ethylene, 4.6%; methane, 18.9%; hydrogen, 4.7%	Cl at low red.
Thorium.....	ThC_2....	8.96	Yellow transparent crystalline mass.....	Acetylene, 47.7%; ethylene, 5.7%; methane, 21.7%; hydrogen, 17.1%; also a small amount of solid and liquid hydrocarbons.....	
Manganese.	Mn_3C....	6.89	Black submetallic mass...	Methane, 51%; hydrogen, 49%: no liquid or solid hydrocarbons.....	Cl at gentle heat.
Chromium.....	Cr_3C_2...	6.47	Opaque crystalline mass, with metallic luster....	Not acted on by water....	
	Cr_4C....	6.75			
Uranium.....	U_2C_3....	11.28	Resembles metallic bismuth.....	⅛ carbon = acetylene, 0.5%; ethylene, 5.5%; methane, 71%; hydrogen, 14.3%. ⅔ carbon = solid and liquid hydrocarbons	Cl at 350°; Br, 390°.
Molybdenum.....	MoC_2....	8.9	Brilliant white metallic ingot.....	Not acted on by water....	
Tungsten.....	W_2C.....	16.06	Iron-gray metallic crystalline mass.....	Not acted on by water....	Cl at red heat.
Vanadium.....	VaC.....	5.36	Metallic crystalline mass..	Not acted on by water....	Cl at 500°.
Zirconium.....	ZrC.....	Undetermined..	Gray metallic mass.....	Not acted on by water....	Cl at 250°; Br, 300°.
Titanium.....	TiC.....	4.25	Dark-colored crystalline mass.....	Not acted on by water....	
Silicon.....	SiC.....	3.12	Colorless or dark hexagonal crystals.....	Not acted on by water....	Cl at 1,200°.
Boron	B_6C.....	2.15	Brilliant black crystals...	Not acted on by water....	Cl at a little below 1,000°.

(*a*) Dissolve carbon at a high temperature, but on cooling the carbon separates as graphite. (*b*) Will not dissolve carbon in electric arc. (*c*) Take up but very little carbon; drop most of it on cooling as graphite; properties of metals changed a little. (*d*) Has not yet yielded any carbides of definite composition.

In a resumé of the various carbides made by the means of the electric furnace H. Moissan (*Annales*, Vol. IX., p. 334) concludes his article with the following interesting deductions:

"From all my experiments thus far made with the electric furnace it seems to

be a general law that the carbon compounds formed are always of a very simple formula, and, with but few exceptions, there is but one compound formed.

"The most interesting reaction, it seems to me, is the ease with which hydrocarbons, either gaseous, liquid, or solid, can be produced by the action of cold water on certain metallic carbides. These reactions are of considerable interest from a geological standpoint.

"The discharge of methane, more or less pure, in certain localities and in amounts which have lasted for centuries, may be due to the action of water on aluminum carbide. A similar reaction may explain the formation of liquid hydrocarbons.

"The theories for the formation of petroleum are as follows:

"1. Production by the decomposition of organic matter, either vegetable or animal.

"2. Formation by purely chemical reactions—a theory first proposed by Berthelot and elaborated by Mendeleef.

"3. Formation of petroleum as a result of volcanic phenomena, as suggested by Humboldt in 1804.

"Starting with 4 kgms. of uranium carbide, we obtained in a single experiment more than 100 gms. of liquid hydrocarbons. The mixture thus obtained consisted mostly of hydrocarbons allied to ethylene, together with a small amount of acetylene and a small amount of saturated hydrocarbons.

"These hydrocarbons are formed in the presence of a large proportion of methane and hydrogen at ordinary temperature and pressure, a fact which leads us to think that, were the decomposition to take place at a high temperature, there would be produced saturated hydrocarbons analogous to petroleum. Berthelot has proved that direct union of hydrogen with an unsaturated hydrocarbon can be caused by the action of heat alone.

"The existence of these new metallic carbides, destructible by water, may possibly modify the theories which hitherto have been advanced to explain the formation of petroleum. We should, however, be careful to avoid making too hasty generalizations.

"It is probable also that there exist petroleums of different origins. At Autun, for example, the bituminous schists appear to have been formed by the decomposition of organic matter. On the contrary, in Limagne, asphalt impregnates all the fissures of a limestone poor in fossils. This asphalt is in direct contact with peperite (basaltic tufa), and therefore bears some relation to the volcanic eruptions in Limagne.

"A well, recently sunk at Riom to a depth of 1200 m., yielded a few liters of petroleum. The formation of this liquid hydrocarbon could be attributed in this region to the action of water on metallic carbides.

"We have already shown under what conditions calcium carbide burns and forms carbon dioxide.

"It is possible that in the early geological periods almost the whole of the carbon existed in the form of metallic carbides, and that, upon coming in contact with water, these carbides gave rise to the formation of hydrocarbons, which in turn, by oxidation, yielded carbon dioxide.

"An example of such a reaction can be found in the neighborhood of St.

Nectaire. The granites, which here form the edges of a Tertiary basin, have yielded a constant and large supply of carbon dioxide.

"We can also readily conjecture that certain volcanic phenomena may be attributed to the action of water on easily decomposed metallic carbides.

"All geologists are aware that the final product of volcanic activity consists of a variety of hydrocarbons, passing from asphalt and petroleum to the final product of oxidation, carbon dioxide.

"An earthquake causing contact between water and metallic carbides could produce the disengagement of immense quantities of gases, and at the same time, as the temperature rose, the phenomena of polymerization of hydrocarbons might give rise to a long series of complex compounds.

"Compounds of hydrogen and carbon might first be formed, and then the phenomena of oxidation follow and complicate the reactions.

"In certain localities a volcanic fissure can act as a powerful flue. It is well known that the gases escaping through fumerolles are of different composition when the volcanic source is submerged in the ocean than when the gases escape directly into the air.

"At Santouir, for example, Mr. Fouqué collected free hydrogen from submerged volcanic orifices, but found only water vapor in the aerial fissures.

"The existence of these metallic carbides, which are so easily prepared at high temperatures and which probably are to be found in the depths of the earth, enables us to explain, in some cases, the formation of gaseous, liquid, or solid hydrocarbons, and might be the cause of certain volcanic eruptions."

THE RARE EARTHS.

Cerium Carbide.—H. Moissan (*Annales Chim. Phys.*, 6, Vol. IX., p. 304; *Comptes rendus*, Vol. CXXII., p. 357). Pure CeO_2, prepared from cerite by fusion of the nitrates of the rare earths with a mixture of sodium and potassium nitrates, is mixed with carbon (from sugar) in the proportion of 198 gms. of CeO_2 to 48 gms. C, in accordance with the equation $CeO_2+4\ C=CeC_2+2\ CO$.

This mixture is heated in a carbon tube in the electric furnace. The oxide first melts and then begins to boil, carbon monoxide being given off. The reaction is continued until the mass is in quiet fusion. With a current of 300 ampères and 50 volts, 100 gms. of cerium oxide can be completely converted into the carbide in from 8 to 10 minutes. A current of 900 ampères and 50 volts reduced 600 gms. of CeO_2 in 3 minutes. Cerium carbide, CeC_2, thus made is a homogeneous mass with crystalline fracture, and has a sp. gr. of 5.23. Under the microscope the powdered carbide is found to be composed of reddish-yellow, perfectly transparent hexagons, which are easily altered in air, rapidly becoming covered with a yellow powder of the dioxide. Fluorine does not act until the temperature is raised, when union takes place with a great rise in temperature. Chlorine decomposes the carbide at 230° C., cerium chloride and graphite being formed. Bromine and iodine unite at correspondingly higher temperatures. Oxygen acts at a low red heat to form CeO_2 and CO_2; the reaction is quantitative. Hydrochloric acid reacts at 650° C., a mixture of cerium chloride and bulky carbon resulting, and hydrogen gas being given off.

Water decomposes the carbide with the production of hydrocarbons, gaseous,

liquid, and solid, and the formation of a cerium hydroxide, white at first but changing on exposure to air to a purplish color. The gas given off has the following composition:

	(I.) Per Cent.	(II.) Per Cent.
Acetylene	75.00	75.50
Ethylene	3.52	4.23
Methane	21.48	20.27

This formation of methane and acetylene led Moissan to the idea that cerium might possibly be composed of two elements, one whose carbide on decomposition would yield acetylene and the other methane. Attempts were therefore made to obtain a separation by fractional solution of the carbide by various solvents and under various conditions. All attempts failed. No better results were obtained by fractional distillation in the electric furnace.

It is of interest to note, however, that the percentage of acetylene in the gases evolved by the action of water is greater in the case of dilute nitric acid and less in the case of dilute hydrochloric acid than when pure water alone is used.

Lanthanum Carbide.—H. Moissan (*Annales Chim. Phys.*, 6, Vol. IX., p. 310). Lanthanum oxide is easily reduced by carbon in the electric furnace, but requires a higher heat than does ceria. If a mixture of 100 gms. of lanthana be heated with 80 gms. carbon for 12 minutes, by 350 ampères and 50 volts, there is obtained a fused ingot homogeneous throughout, with crystalline fracture, and a sp. gr. of 5.02 at 20° C. Its composition corresponds to the formula LaC_2. It is lighter in color than the corresponding cerium carbide, and under the microscope the crystalline fragments appear of a transparent yellow color. Fluorine does not act upon this compound in the cold. Chlorine reacts at 250° C. with incandescence, and bromine at 255° C. It burns in oxygen to La_2O_3 and CO_2, but with more difficulty than does cerium carbide.

Lanthanum carbide is easily decomposed by dilute acids, but fuming monohydrated nitric acid has no action. On the other hand concentrated sulphuric acid, when warm, reacts with the formation of sulphur dioxide. Lanthanum carbide is decomposed by water, gases being given off as below, a small amount of solid and liquid hydrocarbons remaining in solution in the water:

	(I.)	(II.)	(III.)	(IV.)
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Acetylene	71.75	70.18	71.17	70.71
Ethylene	1.93	1.15	0.95	2.04
Methane	27.22	28.67	27.88	27.98

In the case of Sample IV., the gases account for 11.66% of the carbon in the carbide; theory calls for 13.7%. Ether extracted from the water gives sufficient hydrocarbons to account for the difference.

Lucium.—Prosper Barrière, of Paris, has patented a process for extracting from monazite a new element which he calls "lucium" (*Chemical News*, Vol. LXXIV., p. 212). His patent covers the use of this element in incandescent gas lighting. In the *Chemical News*, of November 27, 1896, p. 259, William Crookes describes his examination of a sample of "lucium" sent to him by Barrière and states that it is nothing but impure yttrium.

Thorium.—G. Urbain (*Bull. Soc. Chim.*, Paris, Vol. XV., p. 347) points out objections to the usual method of preparing pure thoria from thorite, and states that when the anhydrous sulphates of the rare earths are to be dissolved in ice water the operation can be greatly hastened by the addition of ammonium acetate. He has found that a mixture of one volume of a solution of ammonium acetate saturated in the cold with two volumes of water dissolves thorium sulphate instantly. Urbain has prepared pure thorium, in which no trace of cerium could be detected by the hydrogen peroxide test, by the following procedure:

The crude oxalates were treated with ammonium oxalate and the solution thus obtained was precipitated by ammonia. The hydroxide was then dissolved in hydrochloric acid and again precipitated by sodium thiosulphate. The thorium hydroxide was then suspended in dilute alcohol and treated with acetylacetone, this forming thorium acetyl acetonate, $Th(C_5H_7O_2)_4$. Upon evaporating this solution nearly to dryness on the water bath, dissolving the residue in chloroform, and allowing the solution to slowly crystallize, the thorium salt is obtained in the form of fine crystals. Under the same conditions the hydroxide of cerium changes to an amorphous gray substance which is almost insoluble in either chloroform or alcohol.

L. M. Dennis (*Jour. Amer. Chem. Soc.*, Vol. XVIII., p. 947) gives further details (THE MINERAL INDUSTRY, Vol. IV., p. 547) concerning the precipitation of thorium by potassium trinitride, KN_3. He finds that when a dilute solution of potassium trinitride is added to a neutral solution of a thorium salt, and the mixture is heated to boiling point for one minute, all of the thorium is precipitated in the form of thorium hydroxide. None of the other earths of the cerium group are precipitated by this reagent, so that it is thus possible to separate thorium from cerium, lanthanum, and didymium in a single simple operation.

Thorium Carbide.—H. Moissan and Étard (*Comtes rendus*, Vol. CXXII., p. 573). Thoria was obtained from orangite in the usual way by decomposition in hydrochloric acid, precipitation by means of oxalic acid, and extraction of the thorium oxalate by repeated treatments with a boiling saturated solution of ammonium oxalate.

Seventy-two gms. of thorium oxide and 6 gms. of carbon were treated with turpentine exactly as in the case of yttrium. The resulting carbide has a sp. gr. of 8.96 at 18°. Heated in oxygen it burns with a blinding light. The same phenomenon is observed with sulphur vapor, in this case forming a dark-colored sulphide, soluble in acids. Selenium acts in a similar manner. When heated in a current of ammonia gas, hydrogen is given off and a nitride is formed. Concentrated acids have but little action, but dilute acids attack it with readiness. In cold water decomposition takes place, giving rise to various gases.

	(I.) Per Cent.	(II.) Per Cent.
Acetylene	47.05	48.44
Methane	31.06	27.69
Ethylene	5.88	5.64
Hydrogen	16.01	18.23

There is also a small amount of solid and liquid hydrocarbons formed, which can be extracted from the water by means of ether.

Yttrium Carbide.—H. Moissan and Etard (*Comptes rendus*, Vol. CXXII., p. 573). The yttria used was obtained by the following method, which the authors claim to be the quickest yet proposed, and which yields most excellent results: The yttrium-bearing minerals are decomposed by concentrated sulphuric acid, thrown into ice water, precipitated with oxalic acid, and the resulting oxalates ignited. The oxides are then dissolved in sulphuric acid and the cerium and didymium groups removed by means of potassium sulphate. The clear solution is precipitated with oxalic acid, the oxalates ignited, dissolved in sulphuric acid, and the resulting solution fractionally precipitated by means of normal potassium chromate (K_2CrO_4). Those earths are first precipitated which tend to form basic chromates—*e. g.*, erbium, thulium, holmium, etc. The last portions contain most of the yttrium. These are treated with a very little ammonia, and then more potassium chromate is added. A flocculent precipitate is first produced, which rapidly becomes crystalline. A concentrated solution of the last fraction showed no absorption bands and had an equivalent weight of 89. The various fractions can, of course, each be treated in turn and a large amount of yttria thus obtained.

Yttria obtained as above was intimately mixed with finely powdered carbon and a little spirits of turpentine. The mass was then compressed in cylinders, dried, and strongly ignited. The resulting mass was submitted for 5 minutes to the action of an arc of 900 ampères and 500 volts. Metallic vapors were given off, which burned with a white flame tinged with purple.

Yttrium carbide, YC_2, is thus obtained in well-fused ingots with distinct crystalline fracture. Under the microscope this brittle mass is seen to be composed of transparent yellow crystals, mixed with graphite. These crystals have a sp. gr. of 4.13 at 18° (taken in benzine). Fluorine attacks them in the cold, and in a current of chlorine they burn at a temperature below a red heat. Oxygen and the vapors of sulphur and selenium react similarly. Yttrium carbide is decomposed by cold water, the gases evolved from two different samples having the following composition:

	(I.) Per Cent.	(II.) Per Cent.
Acetylene	71.7	71.80
Methane	19.0	18.80
Ethylene	4.8	4.45
Hydrogen	4.5	4.95

INCANDESCENT GAS LIGHTING.

The *Journal of the Society of Chemical Industry*, Vol. XV., p. 701, gives the following abstract of an article by Söhren upon the Welsbach light, which appeared in the *Jour. für Gasbeleuchtung*, Vol. XXXIX.:

"When the Auer von Welsbach incandescent light first came before the public, about the middle of the year 1886, it gave an illuminating power of about 14 candles, with an hourly consumption of 2.8 cu. ft. of gas, or 5 candle power per cu. ft. per hour. The mantles were composed of various mixtures of the oxides of zirconium, lanthanum, yttrium, magnesium, thorium, cerium, etc. They were so fragile, and the illuminating power was so little better than that given by the ordinary burner, that it was not until 1891, when Welsbach had further perfected his invention, that the new light really attracted public attention.

The illuminating power had then increased to about 50 candles, with an hourly consumption of 3.5 cu. ft., or 14 candles per cu. ft. per hour, and the mantles lasted for about 800 hours. A more recent development has been the alcohol incandescent lamp, suitable for districts where no gas supply is available.

"The author discusses the various forms of burner used for incandescent gas lighting. The most noteworthy recent improvements consist in arrangements for regulating the air supply to the burner, and for effecting a very thorough mixture of the air and gas before the latter is burned.

"In making the mantle, a tubular-formed fabric of the purest cotton, free from ash, is used, and is first thoroughly purified by means of hydrochloric or hydrofluoric acid. It is then impregnated with 'lighting fluid,' consisting of a solution of the nitrates of the rare earths; and one end, which is to constitute the upper end of the mantle, is treated to another impregnation, either with the fluid or with solution of aluminum and magnesium nitrates, etc., in order to strengthen it. After drying, the upper end of the fabric is drawn together somewhat, by means of asbestos thread, the whole is suspended from a holder, and the cotton burned away, leaving a skeleton of the oxides. The average weight of an impregnated mantle is 5.123 gms.; after 'burning off,' 0.680 gms.; and the available superficies is 54 sq. cm., of which 45 sq. cm. act as glowing surface.

"According to Welsbach's first patent, mantles for emitting white light consist of various mixtures of lanthana, yttria, zirconia, and magnesia, neodymia being added to produce yellow light, and erbia to produce greenish light. His second patent claims the use of thoria in such mixtures as the above, and the use of ceria to produce an intense yellow light; it also claims the use of niobates, tantalates, silicates, titanates, and phosphates of thorium, zirconium, and magnesium for strengthening the contracted part of the mantle, and the use of ammonium nitrate in the fluid to expedite the 'burning off.' A later patent claims the use of thoria mixed with uranium oxide in making mantles. According to Mackean, the individual pure oxides give, with a consumption of 3 cu. ft. of gas per hour, at a pressure of 1 in., the following results:

	Candle Power.	Light.
Lanthana	22.5	White.
Zirconia	12.2	White.
Thoria	25.0	Bluish white.
Yttria	19.8	Yellowish white.
Ceria	4.0	Nearly red.

"It is therefore evident that the extraordinary illuminating power of the Welsbach light is not due to one of the oxides alone, but to certain mixtures of them.

"The chief sources of the rare earths are orthite and allanite, containing 10 to 20% of cerous oxide, 3% of yttria, 10% of lanthana, some erbia, etc.; gadolinite, with yttria, lanthana, cerous oxide, erbia, etc.; xenotime, with 62% of yttria, 11% of cerous oxide, 27% of phosphoric acid; euxenite, with 31% of niobic acid, 18% of yttria, 2% of cerous oxide, etc. (according to other analyses, containing still

more Nb_2O_5 and Y_2O_3); yttrotantalite, with 19% of yttria, etc.; fergusonite and tyrite, with 48% of niobic acid, 46% of yttria, and 4% of cerous oxide; cerite, with 13% of cerous oxide, and 8% of lanthana and didymia; zircon, with 57% of zirconia, tantalic acid, etc.; and monazite, with cerous oxide, lanthana, etc. These minerals contain but little thoria, are very scarce, and accordingly expensive. Thoria was obtained at first from thorite and orangite, which occur as small crystals scattered through Norwegian igneous rocks, and contain on an average 73% of ThO_2, 17% of SiO_2, and 10% of water. These minerals are, however, so extremely rare, as is also æschynite (which contains 15% of ThO_2), that some other source had to be found to meet the demand for thoria. Monazite has supplied the want. It is a mineral occurring in large quantities in the gneiss and granites of the Brazilian coast, and in the river sands of North and South Carolina and Canada. Monazite contains from 3 to 4% of ThO_2, and the sand at present shipped to Europe consists of 70% of monazite, mixed with zircon, xenotime. ilmenite, magnetite, gravel, and organic matter. Drossbach found in Brazilian monazite 60 to 70% of cerous oxide, 0.5 to 1.5% of thoria and erbia, with other oxides of the yttrium group. Fresh sources of monazite are continually being discovered, and the price of sand containing 70% of monazite is about £30 per ton f. o. r. in North Carolina, £37 per ton f. o. b. New York, or £48 per ton c. i. f. Hamburg.

"Analyses of mantles made by the German Incandescent Gas Light Company having shown that the prescriptions of Welsbach's patents were not being exactly followed, but that the mantles consisted chiefly of thoria, with about 1 to 2% of ceria, a great number of competing companies have been manufacturing mantles of similar composition, on the supposition that such prescriptions were not protected by Welsbach's patent, so that there are now about 74 manufacturers of incandescent burners in Germany. The prices have accordingly been much reduced, and in the endeavor to obtain a still cheaper form of mantle, Haitinger has patented a combination of alumina and oxide of chromium. Much costly litigation has been entered into by the competing companies in endeavoring to procure the revocation of Welsbach's patents, and the decision of the Supreme Court has turned out much to Welsbach's disadvantage."

R. Fresenius and E. Hintz (*Zeit. für anal. Chem.*, Vol. XXXV., p. 525) describe experiments which they made to ascertain whether the cerium present in the Welsbach mantles should be regarded as an impurity in the original thorium nitrate employed or whether it had been added to the thorium solution before the mantle was impregnated with the earths. They conclude—

(*a*) That the mantles lowest in ceria still contain much more ceria than the most impure samples of commercial thorium nitrate which are highest in ceria.

(*b*) That the amounts of ceria found in the mantles cannot be regarded as impurities arising from the use of commercial thorium nitrate.

(*c*) That the ceria present has been added, at least in part, or has been introduced into the mantle through the use, intentional or unintentional, of impure thorium nitrate high in ceria. The article contains the following table, giving the percentage composition of various mantles manufactured in Germany:

Constituents.	Firm Supplying the Mantle.										
	F. H. Aschner & Co., Berlin.	C. H. Stobwasser & Co., Berlin.	F. Butzke & Co., Berlin.	Emil Billeit, Berlin.	Horrwitz & Saalfeld, Berlin.	C. Kramme, Berlin.	Friedr. Siemens & Co., Berlin.	Meteor, Berlin.	S. Salzmann, Berlin.	Neue Gasglühlicht-Act.-Ges., Berlin.	Fritz Trendel
Thoria.....	98.03	97.87	96.42	98.20	98.91	98.23	99.06	98.31	98.20	98.98	98.22
Ceria.....	0.61	1.35	2.02	0.84	0.38	0.78	0.71	1.21	1.34	0.82	0.93
Yttria.....	0.24	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Neodidymia..	.....	.....	1.00	0.36	Trace.	.....	Trace.	.....	.....	.....	0.08
Zirconia.....	.....	Trace.	.....	.....	.....	0.10	Trace.	.....	.....	Trace.	.....
Lime.....	1.05	0.65	0.44	0.45	0.62	0.77	0.20	0.40	0.29	0.15	0.64
Magnesia....	0.07	0.13	0.12	0.15	0.09	0.12	0.03	0.08	0.17	0.05	0.13

The *United States Consular Reports* for June, 1896, p. 242, give the following interesting review of the Welsbach patents:

"The long-pending litigation over the German patents covering what is known as the Auer von Welsbach incandescent gas-burner terminated on the 7th of November last in a decision by the German Patent Office which had the outward appearance of a substantial victory for the patentee, but which has led to a situation in this country so anomalous as to claim the interest of not only makers of incandescent gas-burners in other countries, but of producers of monazite, thorite, and other minerals from which the thoria and other rare oxides used in such manufacture are derived.

"The German patents of Dr. Carl Auer von Welsbach, the Viennese inventor of the incandescent gas-burner which bears his name, are four in number, viz.: The original patent, No. 39,162, of September 23, 1885, and three additional or subsidiary patents, No. 41,945, of April 29, 1886; No. 44,016, of January 20, 1887; and No. 74,745, of August 15, 1891, all of which will, of course, expire with the lapse of the original patent on the 22d of September, 1900.

"The enormous profits reaped by the Deutsche Gasglühlicht Gesellschaft, at Berlin, which controls the Auer patents in Germany, have started up a swarm of competitors, who have put on the market burners of similar construction at prices which forced the Berlin company several months ago to reduce its retail price from 15s. to 10s. per burner. The Berlin company thereupon began suit against these rival makers for infringement of the Auer patents, and, pending the decision, published warnings against the makers and users of the infringing gas-burners.

"The decision is to the effect that the scope of claim 3 in the original patent (which describes the construction of the incandescent mantle, which forms the essential feature of the invention) is restricted so as to cover only the combinations of rare earths which are definitely specified in claims 1 and 3 of the same instrument; and, further, claim 3 of the additional patent No. 41,945 is cancelled altogether. Now, claim 1 of the original patent specifies certain combinations of oxides for the impregnation of mantles, which render a gas flame incandescent and produce therefrom a white, yellow, or greenish light. The substances named in this clause are magnesia and the oxides of lanthanum, yttrium, and zirconium for white light, to which is added "neodymzircon" to produce a yellow light, and erbium to give the flame a greenish tinge. Neither in clause 1 nor in clause 3 of the original patent is there any mention of thorium oxide (thoria), which, as is well known, forms the principal material in all modern

incandescent mantles, the proportion of thoria ranging in some formulæ as high as 98 or even 99%, the remaining ingredient being an oxide of cerium, yttrium, erbium, or zirconium, to modify the color of the light produced.

"This importance of thoria seems to have been realized by Dr. Auer only after long experiment, and its use for the purpose named was covered by claim 3 of his supplemental patent (No. 41,945) of April 29, 1886. But in the recent trials it has been shown that the use of thoria for this purpose was not original with Dr. Auer, and that clause of his patent is therefore annulled, leaving the use of pure thoria open and free to every one in Germany. But, as is generally known, pure thoria produces under such conditions a greenish, ghastly light of no industrial value. In order to give out the white and yellow shades of incandescent light required for ordinary use, it must be confined, as above stated, with other oxides of the cerium-yttrium group. Experts declare that the exact formula by which the Welsbach mantles of the Berlin Gasglühlicht Gesellschaft are actually made is not described or even mentioned in any of the Auer patents, and, as the use of thoria for this purpose is free, they rely upon the fact that thoria, when derived from monazite, usually contains small percentages of cerium and erbium oxides, and may therefore be used directly, in its natural condition, for the manufacture of incandescent mantles. As a natural combination of chemical elements cannot be patented, it would be very difficult to prove whether the combination of thoria and other oxides used by a rival manufacturer were natural or artificially composed. So confident are many experts of their ability to evade the existing patents that there are now in the field not less than 12 manufacturers of gas-burners more or less similar in type to the Auer. Of these rival makers, 7, including the Meteor Gaslight Continental Company, are located at Berlin, and there are others at Dresden, Munich, Carlsruhe, and other German cities.

"These makers, undismayed by the decision of November 7, have pushed the competition vigorously, and to the great temporary advantage of the public. The Auer Company, at Berlin, was forced to again reduce its price to 5s. per burner, while the Meteor Company furnishes burners of apparently identical brilliancy and permanence for 5s. each, and at these rates the use of incandescent gas-burners is becoming all but universal in Germany. Further litigation is certain to follow, but the power of the original monopoly is apparently broken, and its former exorbitant prices can probably never be re-established."

Concerning the market for monazite the report goes on to say: "From the American standpoint, one of the most interesting results of this controversy is its influence upon the current demand for monazite. According to Professor Nitze, Assistant State Geologist of North Carolina, the product of monazite sand in the Carolinas was 65 tons in 1893, 300 tons in 1894, and about 600 tons in 1896, and as such sand, containing from 60 to 75% of monazite, is worth in Germany from £25 to £27 per ton, the subject is one of definite commercial importance. Thus far, the European market has been largely dominated by the supply of monazite sand from Brazil, where it is found in the form of a fine, washed seashore sand, containing usually only from 2 to 3% of thoria, whereas the Carolina monazite abounds in crystals of all sizes up to that of a grain of wheat, and contains generally from 3 to 6% of thoria. But for the reason that the Brazilian sand is uni-

formly fine, free from particles of titaniferous iron, and can be put directly into the extracting process, without being artificially pulverized, it is preferred by the Auer-Welsbach companies at Vienna and Berlin. The North American monazite, on the other hand, must first be ground to a fine powder, which, in consequence of the hardness of the crystals, is a somewhat tedious process, unless the best machinery is employed. For these reasons, and owing to the fact that large consignments of Brazilian monazite sand have been recently received at Hamburg, the market is now somewhat congested, and the prices offered no longer tempt exporters in the United States. A ton of Brazilian monazite sand, costing at present in Hamburg £24 15s., yields, when well worked out, from 20 to 25 kgms. of pure thoria, which is worth from £500 to £625, according to the degree of purity.

"The whole situation confirms the suggestion already made in these reports, that the ultimate and effective plan of organizing the monazite industry in our country would be to establish there, at a convenient point in the mining district, a laboratory, where, by employing the most improved and economical methods, the monazite—including the poorer sands which have been concentrated by a process recently perfected—may be worked up, the thoria extracted, and made available as a finished product in all countries where incandescent gas-burners are manufactured.

"Thorium oxide is now worth in Germany from £25 to £31 per kgm., according to purity, and the demand is thus far greater than the supply. Most manufacturers of incandescent burners would prefer to buy their supply of thoria ready for use than to conduct a separate laboratory for extracting and preparing it.

"The one pertinent question, from an economic standpoint, would be whether the chemicals which are used in extracting thoria can be transported from New York or Baltimore to the Carolina monazite district more cheaply than the crude sand can be brought from there to Europe, where from 96 to 98% of its bulk becomes at the first operation absolute waste material. The chemicals required for this process are neither rare nor costly, and with a well-chosen site, at which water power may be utilized for pulverizing the raw material, the success of such an enterprise would, in the opinion of competent authorities, be practically assured."

In the *Engineering and Mining Journal*, Vol. LXI., p. 207, there is a description of the patent of Rudolph Langhaus, who uses fine platinum wire (0.035 mm. diameter) for the mantle, and electrolytically coats the wire with either the nitrates or sulphates of thorium and cerium, these compounds being then converted into oxides by heat. It is stated that mantles thus prepared are unbreakable by ordinary usage and that they yield higher candle-power because of the greater amount and consequently greater surface of oxides in the mantle. The cost of this mantle is only 10c. more than the usual fragile kind.

SALT.

THE salt produced in the United States is obtained partly by solution and partly by dry mining, the former method being by far the more important. A small amount of salt is obtained by the evaporation of sea-water, but this is only to meet local requirements and the total is insignificant. Rock salt is mined in the Warsaw district of New York, in the Hutchinson district of Kansas, at the Avery Island mine of Louisiana, in Utah, and in California. In 1896 there were 19 mines reporting a production of rock salt. The full details of the salt industry of the United States are given in the subjoined tables.

PRODUCTION OF SALT IN THE UNITED STATES.

(In barrels of 280 lbs.)

Year.	California.	Illinois.	Kansas.	Kentucky.	Louisiana.	Michigan.	Nevada.	New York.	Ohio and W. Va.	Utah.	Other States.	Total Barrels.	Total Short Tons.
1892	235,000	60,000	1,232,850	.....	192,850	3,812,054	20,000	4,400,000	738,000	900,000	225,000	11,784,954	1,649,894
1893	312,850	65,000	1,607,000	.....	172,500	(a)3,514,485	7,988	4,413,181	(b) 925,620	438,002	182,435	11,639,081	1,629,471
1894	324,623	65,000	907,000	34,400	185,000	3,485,428	5,100	6,529,694	1,342,400	821,565	439,421	14,140,581	1,979,680
1895	365,280	68,000	1,168,951	20,024	159,775	3,929,342	8,428	5,919,155	1,704,000	411,784	521,115	14,275,854	1,998,507
1896	408,000	55,600	1,347,793	2,400	173,117	3,163,542	7,818	5,597,852	1,778,342	349,056	461,040	13,354,560	1,869,534

(a) Amount inspected. (b) Includes the production of Pennsylvania.

SALT PRODUCTION IN THE UNITED STATES.

(In barrels of 280 lbs.)

States.	1895.					1896.				
	Evaporated.	Rock Salt.	Total Barrels.	Value.		Evaporated.	Rock Salt.	Total Barrels.	Value.	
				Total.	Per Bbl.				Total.	Per Bbl.
California	329,566	35,714	365,280	$114,150	$0.35	236,000	172,000	408,000	$127,500	$0.31
Illinois	68,000	.....	68,000	47,600	0.70	55,600	.....	55,600	33,396	0.60
Kansas	581,451	587,500	1,168,951	481,970	0.41	757,007	590,786	1,347,793	519,475	0.31
Kentucky	20,024	.....	20,024	12,800	0.64	2,400	.....	2,400	1,632	0.68
Louisiana	.....	159,775	159,775	100,209	0.62	.....	173,117	173,117	94,016	0.54
Massachusetts	1,115	.....	1,115	1,595	1.43	1,240	.....	1,240	1,488	1.20
Michigan	3,929,342	.....	3,929,342	2,357,605	0.60	3,163,542	.....	3,163,542	1,645,062	0.51
Nevada	8,428	.....	8,428	7,495	0.89	7,818	.....	7,818	6,188	0.79
New York:										
Onondago district	2,562,825	.....	2,562,825	896,888	0 35	2,514,123	.....	2,514,123	879,943	0.35
Warsaw district	1,959,530	.....	1,959,530	646,644	0.33	1,837,329	.....	1,837,329	687,627	0.37
Rock Salt	.....	1,396,800	1,396,800	377,136	0.27	.....	1,256,400	1,256,400	326,664	0.27
Ohio	1,479,000	.....	1,479,000	798,660	0.54	1,564,752	.....	1,564,752	672,843	0.43
Pennsylvania	145,000	.....	145,000	87,000	0.60	120,000	.....	120,000	72,000	0.60
Texas	255,000	.....	255,000	211,560	0.83	197,000	.....	197,000	157,600	0.80
Utah	406,981	4,800	411,781	144,125	0.35	343,056	6,000	349,056	125,752	0.36
Virginia	120,000	.....	120,000	56,400	0.47	142,800	.....	142,800	67,116	0.47
West Virginia	225,000	.....	225,000	130,500	0.58	213,590	.....	213,590	121,796	0.57
Total barrels	12,091,265	2,184,589	14,275,854	$6,472,337	$0.45	11,156,270	2,198,303	13,354,560	$5,540,098	$0.41
Total metric tons	1,735,591	277,443	1,813,034	.....	a 3.57	1,416,846	279,184	1,696,030	.....	a 3.26
Total short tons	1,692,682	305,825	1,998,507	.....	b 3.24	1,561,789	307,745	1,869,534	.....	b 2.96

(a) Per metric ton. (b) Per short ton.

The Salt Deposits of Louisiana.

These were described by A. F. Lucas in the *Engineering and Mining Journal* of Nov. 14, 1896. According to him, the Avery Island mine was discovered in 1862 at a depth of 20 ft., but no systematic work was done until it was leased by the American Salt Co. in 1879. This company was succeeded in 1883 by the New Iberia Salt Co., and the latter in 1893 by the firm of Myles & Co., of New Orleans. The discovery of the Kansas deposits affected seriously the demand for the product of this mine, and only its exceptional purity has retained for it more than a local sale. In recent years its production has averaged about 30,000 tons per annum, one fourth of which goes to packers in the Northwest, one fourth to St. Louis and Chicago for refrigerating purposes and salting hides, and the remaining half to supply local demand.

The operations of the American Salt Co. and its successors were not carried on in a scientific manner, and various difficulties have arisen as a consequence. In 1883 a cave occurred, and since then there have been others of increasing size and frequency. Attempts to stop them have been fruitless, and eventually the mine will be lost on account of them.

The Avery Island salt formation is of Tertiary origin. It is supposed to rest on the Cretaceous, but this has not yet been proved. The overlying sand (of the Quaternary period) rests directly on the salt, with only occasional evidences of hard-pan between the two formations.

The mine was unfortunately located in a sink, which receives the drainage of a considerable area, and from this there has been continual trouble. The American Salt Company sunk a fairly good double-compartment shaft only 90 ft. deep, however, and carried on heavy operations from that depth. It was soon found that cracks existed in the salt, through which water percolated. In a short time a cave of considerable area occurred, which necessitated the abandonment of the east end of the mine. For some time the west end was then worked, when some heavy caving occurred there also, carrying immense amounts of water and sand into the mine. It was then decided to sink the shaft 75 ft. deeper, and a second level was opened which gave a new lease of life to the mine. However, the continuous flooding of the upper level with fresh water and pumping out saturated brine caused the opening of new communications between the two levels, so that in July, 1895, the lower had to be abandoned. Operations are now confined to a small area of the old 90-ft. level, which can last but a limited time, even if no further cavings occur to stop operations altogether. The New Iberia Salt Company made repeated attempts to sink additional shafts in 1886 and 1887, but after spending considerable money on them they were abandoned as utter failures.

The system of mining employed is an undercut method, which is very economical. Chambers of about 80 ft. width are opened and an undercut is made of, say, 7 ft. height and about 200 ft. length. This undercut is the most expensive part of the operation. As fast as it proceeds the salt is cleared out, and when a fair distance is reached the roof is attacked by hand-drills and the salt is shot down to a height of about 20 ft. This salt is cleared out by tram-cars and hoisted

to the mill, leaving a chamber 80 ft. wide, 20 ft. high, and, say, 200 ft. long. The roof is then again attacked and more salt is shot down. Standing on this pile of salt the men work up to a height of 60 ft. in the middle and about 50 ft. on each side. The chambers are opened 60 ft. apart and parallel with each other and then crosscuts are run, leaving pillars 60 ft. by 60 ft. to hold the roof. None of the arches of the roof have ever given out except where fresh water found a way through; this water, increasing in quantities in proportion to the area opened, caused the caves above mentioned.

Owing to the great purity of this deposit—from 97% to 99% sodium chloride—the salt is marketed without any preparatory treatment whatever. It is hoisted in ordinary mining-cars, dumped on top of the head-frame, passed through a crusher, sized through a series of screens for coarse grades, and the fine comminuted by ordinary milling-stones.

On Orange Island, or, as it is better known, Jefferson Island, which is the property of Mr. Joseph Jefferson, salt was discovered accidentally in 1895 by a borehole at a depth of 235 ft. The ground was then tested systematically. In all 8 holes were put down, from 600 ft. to half a mile apart, and from 150 to 2100 ft. in depth. These proved a salt deposit of immense extent. The salt was encountered at depths varying from 90 to 400 ft. One hole which struck salt at 235 ft. was sunk to a depth of 2100 ft. and finished still in salt. The salt is of very pure quality.

SALT PRODUCTION OF THE CHIEF COUNTRIES OF THE WORLD. (*a*)

(In metric tons and dollars.)

Year.	Algeria.		Austria.		Canada.		France.		Germany.	
1891	34,665	$145,217	301,422	$10,866,780	40,854	$161,179	810,675	$2,827,777	1,170,179	$4,300,340
1892	24,784	104,086	288,424	9,982,094	41,265	162,041	974,000	3,353,390	1,167,264	4,168,915
1893	19,008	93,416	306,856	.....	56,539	195,926	1,114,000	3,317,471	1,215,015	4,140,279
1894	17,830	74,763	311,570	.....	51,890	170,687	890,607	2,287,624	1,253,026	4,333,707
1895	.....	.....	278,875	.....	47,515	160,455	.....	.....	1,212,336	4,340,321

Year.	Greece.		Hungary.		India.		Italy.	
1891	19,772	$316,352	162,891	$6,801,222	1,032,268	(*b*)	40,543	$129,280
1892	21,660	345,600	176,368	7,538,891	913,292	$1,794,280	23,721	88,074
1893	18,329	293,264	167,209	7,010,400	853,180	1,594,430	25,392	96,720
1894	21,310	332,436	169,282	7,153,440	1,318,198	(*b*)	30,793	114,835
1895	22,238	335,800	169,395	6,456,480	.....	.....	29,315	110,190

Year.	Russia.		Spain.		United Kingdom.		United States.	
1891	1,351,186	$2,586,280	582,833	$1,748,498	2,077,072	$4,884,120	1,300,107	$5,639,083
1892	1,458,508	2,404,000	682,634	2,596,741	1,988,024	4,307,005	1,542,133	5,900,000
1893	1,351,056	2,224,234	151,464	85,052	2,015,027	3,676,110	1,478,230	5,623,647
1894	1,354,218	1,723,200	206,565	88,898	2,271,686	3,818,145	1,498,193	6,370,339
1895	1,519,672	(*b*)	326,320	952,098	2,218,025	3,548,755	1,813,034	6,472,337

(*a*) From the official reports of the respective countries. For Austria, Hungary, Russia, and Spain the production of all kinds of salt is given; Germany, rock salt and common salt; Greece, sea salt; France, rock and sea salt; Algeria, sea and rock salt; Italy, rock and salt from brine; United Kingdom, rock and brine salt; India, salt which is liable to British salt tax only and does not include salt made in certain native States.
(*b*) Not reported in the official statistics.

Salt in Austria.

By R. Helmhacker.

The salt deposits of Austria are found in two geological formations, the Triassic and the Tertiary. The Triassic are confined to the northern limestone Alps; the Tertiary extend on both sides of the Carpathian Mountains. In each the mining of salt is conducted in different ways. While the Tertiary deposits supply chiefly rock salt, in the Triassic the manufacture of brine salt is carried on. The principal salt deposits being geographically distributed in the northern part of the empire, the southwestern portion derives its supply from the Adriatic Sea.

The Triassic Salt Deposits.—In Austria the Triassic rocks are subdivided into the Noric (comprising the red or Werfner sandstone, the Alpine shell-limestone and other strata), the Carnic, and the Rhaetic, the last two composed of limestones and dolomites. The geologists of the Austrian Geological Survey believed that the greater part of the saltbeds is included either in the lower part of the Carnic division or the top of the Noric; but we are also indebted to many eminent salt-miners for their conclusions relating to this question. The mining engineers who have had opportunity to study the relative position assign the beds below the top of the Werfner sandstone, and this is the true horizon in which the salt in the Austrian Alps is deposited. The Werfner sandstone, forming the lowest stratum of the Triassic, does not carry salt throughout its whole area, but only on the northern declivity of the Alpine range, and especially in the northern flanks of the Austrian Alps, along a line that can be traced from the east at Altenmarkt (San Gallen) on the Ens River, Hall north of Admont and Aussee, all in Upper Styria, through Hallstadt and Ischl in Upper Austria, Hallein in the Salzburg, Berchtesgaden and Reichenhall to Traunstein in Southeastern Bavaria, a distance of 135 kilometers, and ending, after a stretch of 110 kilometers barren of the saliferous deposits, at Hall northeast from Innsbruck in the Tyrol. The saliferous series appear in this zone suddenly, and end in the same manner. In the eastern part, in the neighborhood of Admont, they evidently correspond in age to the upper part of the Werfner sandstone group, and we may assume the conclusion above mentioned to be correct. But between Aussee and Hallein many modifying agencies have been at work to disturb the saliferous deposits, and the determination of their age was until lately a subject of much discussion.

As to the saline deposit at Hall in the Tyrol, all agree that it belongs to a higher horizon in the Triassic series. A section presenting the geological features in the disturbed part between Aussee, Hallstadt, and Ischl will show the saliferous deposits, though tilted, covered conformably with Upper Triassic limestones and dolomites, *Lebergebirge*, hydraulic marl, and Werfner sandstone. At St. Gallen and Altenmarkt, the salt-bearing deposit rests on Werfner sandstones, and is covered by limestones of the *Muschelkalk* division (Guttenstein or Reichenhall limestone); above them is a marl (Zlambach marl) covered by heavy limestones (Hallstadt marble) with a rich Triassic fauna. The higher Triassic series are wanting, and the Hallstadt limestone is capped with Liassic limestones. At Aussee and Hallstadt the beds next below the saliferous series are not accessible to observation; overlying are weathered clayey beds (liver-clay, *Lebergebirge*), limestone

(the Alpine shell-limestone or Recoaro limestone), Zlambach marl, and Hallstadt limestone.

At Ischl the order of succession is exceptional. The beds beneath the salt-bearing are not easy to prove, but overlying are: Lustrous slates, gypsum-clay, Werfner sandstones covered with pyritic beds, the limestone of Hallstadt, and uppermost dolomitic limestones of the Upper Triassic.

The saliferous deposits (*Haselgebirge*) contain clays and salt in immense thickness, but they thin off on their strike, and sometimes also in the dip. The saliferous clay of the *Haselgebirge* of gray or, as it always contains much iron oxides, of brown color, averages: SiO_2, 40%; Al_2O_3, 16.5%; Fe_2O_3, 7%; CaO, 1%; MgO, 4.5%; K_2O, 3%; Na_2O, 0.5%; SO_3, 0.5%; CO_2, traces; H_2O, 20%; NaCl and other salts soluble in water, 6.75%.

The gray clay passes into a clayish marl interstratified with foliated rock salt, but on the borders of the deposit becomes ferruginous and contains much gypsum.

Besides the saliferous clay the following minerals constituting the immense salt-bearing deposit are remarked:

1. Rock salt, massive and granular, also columnar; gray, green, reddish, white; containing 100% to 96% NaCl. It occurs in strings, veins, or beds.

2. Polyhalite, usually in compact fibrous stringers 2½ cm. thick, of flesh or brick-red color. It is commonly mixed with other substances and contains: $CaSO_4$, 52.9%; $Al_2O_3 + SiO_2$, 10%; $MgSO_4$, 9.6%; K_2SO_4, 12.4%; Na_2SO_4, 0.5%; NaCl, 5.4%; H_2O lost at 100° C., 4.2%; H_2O lost at red heat, 4.2%.

3. Small particles and efflorescences of soluble and moisture-attracting salts: blödite, löweite, and rarely kieserite.

4. Anhydrite, massive and in granular crystalline masses, is common.

5. Gypsum is rare within the beds of the *Haselgebirge*, but penetrates in extensive patches and strings in the hanging wall.

6. Fragments of the liver-rock (*Lebergebirge*).

7. Ferruginous marly clay.

8. Fragments of the Zlambach marls, lying over the *Lebergebirge* above the saliferous beds, and containing many petrifactions of the Triassic system.

9. Fragments of red sandstones (Werfner sandstone) whose normal position is either below or above the saliferous layer, and lustrous schists of the same age.

10. Different limestone fragments (Hallstadt limestone) derived from the overlying.

11. Some rarer minerals, as sylvinite, carnallite, symonite, galenite, and pyrite.

The coarse fragments of foreign mineral or rock imbedded in the saline clay are thrown together confusedly. The *Haselgebirge* displays in its stratification many sharp and repeated folds, whence we imagine that it was very pliant when bent into its present position.

The upper part of the *Haselgebirge* at the junction with the overlying marls was subjected to the action of moisture and air, and, where it was not capped by superincumbent strata, also to the action of rain. The saliferous clay being thus permeated by water, salt was dissolved and the grayish clays were oxidized and discolored. Such effects are traced far into the interior of the saliferous clays; and they turned, after the salt had been removed, into liver-brown, more or less

tenacious clays, named *Lebergebirge* (contemporaneous with the Werfner Alpine sandstone), completely preserving the *Haselgebirge* from further lixiviation. It is admitted that nothing strictly analogous can be seen on the earth's surface.

Six great beds of saliferous clay are now worked; other deposits, in the eastern part, were abandoned several centuries ago. The outcropping of a part of these is due to the upheaval on fault lines. They are not equally rich in salt. The yield of salt from the brine of Aussee is about 80%; in Hallstadt it is 50 to 70%; at Ischl it is 57 to 60%; in Hallein and Berchtesgaden, 50 to 60%, and at Hall in the Tyrol—the poorest of all—it is 30 to 33%.

The richest salt-clay deposit, at Aussee in Upper Styria, is entered by 13 adits; the salt is covered by hydraulic marls and Hallstadt limestones, which are again capped by Jurassic limestones. The length of the deposit is 1610 meters, the width 870 meters, and the extension in depth is known to be 545 meters. The total length of drifts is 36 kilometers.

The deposit at Hallstadt in Upper Austria is opened by 15 adits. The saline clay is covered by barren clay. In the salt clay are included huge masses of barren clay and Jurassic limestones; melaphyr has also penetrated it. The series is partly covered by gravel and earth. The beds have a length of 2600 meters, a width of 800 meters, and they are known to the depth of 450 meters.

Though the salt-clay at Ischl in Upper Austria lies on the line of extensive dislocation of strata, trending from Hallstadt, and bringing the *Haselgebirge* by a vertical displacement in contact with the Lower Cretaceous (Neomiocene) limestones, the natural order of the succession of the beds is still regular. The *Haselgebirge* is covered by barren clays or lixiviated saline clay, penetrated with gypsum; then liver-rock, red sandstone of the Werfner series, and lustrous shales; both the liver-rock and the lustrous shales should be classed as equivalent to the Werfner shale or sandstone and contemporaneous with it. Further upward follow beds impregnated with pyrite; then ore-bearing (galena) dolomite which is covered by masses of Upper Triassic limestones, capped in some places by limestones of the Jurassic period. The salt mine at Ischl is entered by 14 adits, of which the deepest has been driven to the length of 1600 meters. The length of the saliferous deposit is 900 meters, the breadth 300 meters, and the dip is ascertained by a bore-hole to extend below the depth of 723 meters. The mines above enumerated are especially known as the Kammergut salines.

The salt mine at Hallein in Salzburg forms a part of the deposit extending from Bavaria (Reichenhall) to Salzburg, but the depth to which it can be traced is still unknown. The saline clay is covered by gypsum and hydraulic marls, lying over the younger Hallstadt bed (marble); at its outcrop it is covered by barren clay. The length is 2400 meters, the breadth probably 1500 meters, and the depth 280 meters. The *Haselgebirge* is contaminated with many by-salts containing chloride of magnesium, as impurities of the common salt.

The salt mine at Berchtesgaden in Bavaria forms the continuation of the vast beds of Hallein to the west. Besides the rock salt many by-salts have been deposited in the course of the formation of the *Haselgebirge*. It is ascertained that the geological horizon of these deposits is at the junction of the German *Muschelkalk* (represented in the Alps by the Recoaro or Guttenstein limestone) and the new red or Werfner sandstone.

But Hall in the Tyrol, whose saline clay yields also magnesium salts, makes the comparison with all the other Alpine salines unsatisfactory, for the salt-bearing rocks have been produced at a time contemporaneous with the deposition of the German *Keuper* (mottled clays). The following series can be traced from bottom upward: Werfner beds, Alpine *Muschelkalk* (Partnach shale, Partnach dolomite, and Reichenhall limestone), on which the *Haselgebirge* is deposited; it is overlaid by Alpine *Keuper* and limestones of the Carnic series (Wetterstein and Seefeld limestone), but the Hallstadt limestones are wanting. The length of the deposit is 1900 meters, the breadth 800 meters, and the explored depth 300 meters.

The Extraction of the Salt.—The salt mines situated along the northern Alpine range are of great antiquity, having been mined by a Gallic tribe, the Halores (which means salt-makers). The name *hal* or *hall*, signifying salt, derived from the Gallic tongue, is still preserved in many words referring to salines. Many traces of that extinct tribe remain in the burying-places where their tools and implements, made of bronze, were interred, it is said, in the first century of our era. The mining was carried on by wells, from the sumps of which the brine was drawn by small tubs; ruins of these old mining-works exist to a depth of 120 meters. Since the twelfth century shafts with adits and draining-works have been in use, and since the fourteenth century chambers (*Wehre*, *Werker*) in which the salt from the ceiling and the side walls was dissolved by water. The rock salt, which is too much mixed with earthy and stony matter, cannot be won by the ordinary methods, hence the method of extraction by leaching, which is peculiar to these Alpine deposits. A chamber is opened between two levels, from the upper of which a winze is sunk for hoisting, and an incline for pipe and ladder way, while an outlet drift is driven from the lower. The latter is closed by a clay dam through which the outlet-pipe passes. As the salt is dissolved the clay falls to the bottom, so that the wooden box through which the outlet-pipe takes off the brine has to be raised from time to time.

The residual clay absorbs about 30% of brine, swells up in volume, and the whole chamber fills to the roof, whence it is hoisted in buckets through the shaft and delivered to the refuse-dumps. At least enough must be cleaned out to make room for a new portion of leaching-water.

The roof rises from 0.1 meter to 1 meter at each leaching, and consequently the chamber gradually assumes the form of a truncated cone with sides inclined at 45°, till at length it reaches the higher level. When this occurs the room is pronounced "dead."

At the end of each leaching, which gives about 30,000 hectoliters of brine, the chamber is partly cleaned of waste clay and rock, surveyed and mapped. Hollows are filled with tamped clay; protrusions must be stripped off; some parts require props or pillars made of wooden blocks or walls of clay; if the roof has partly collapsed, the *débris* is leveled, and the top secured; in short, all is done to maintain the regular shape of the chamber. By this process of intermittent leaching about 700,000 hectoliters of brine can be obtained from one chamber before it is pronounced "dead."

To prevent the collapse of the roof the plan of the chamber must be in proportion to the breaking strain of the roof. Long experience allows the unsupported roof-plane in the chambers to be extended, at Aussee, up to 14,000 square meters:

at Hallein to 9700 meters; at Hallstadt to 7900 meters; at Ischl to 7600 meters; at Hall to from 4750 to 5000 meters. A chamber whose roof covers 4750 sq. meters yields 1,500,000 cu. meters of salt clay. The chambers at Aussee are the largest, the rock being the stronger. But notwithstanding this, many have totally caved in.

During the second half of the present century new methods of extraction have been introduced. Thus, the water to dissolve the salt is continuously conveyed by the feeding-pipes in the incline; and the corresponding volume of saturated brine is drawn off on the lower level. The leaching is also assisted by pressure, the head of the water being from 7 to 80 cm. higher than the roof of the chamber which is to be leached. Instead of beginning with the extraction of a single chamber, and after its exhaustion proceeding higher, a method of leaching two chambers simultaneously was tried. This method is called "shaft leaching."

As all the Alpine salt mines command a sufficient supply of water at a proper height, and have an easy outlet for the turbid stream at a lower level, a jet of water under pressure was tried for driving drifts, but this method has lately been replaced by blasting and picking. With a hand-drilling machine handled by a skilled laborer and a helper one cubic meter of rock extracted from the breast of a drift costs 1.80 fl., including blasting material and light, the skilled laborer earning 1.17 fl. and the ordinary laborer 96 kr. as an average day's wages.

The present average cost of one hectoliter of brine drawn from the mine is 4.8 kr. In the saline at Kossov in Galicia the salt clay of high salt percentage is worked on the dry method and afterward treated by lixiviation above ground; the brine obtained at the Alpine salines by the application of leaching by the Kossov method would cost 8.6 kr. per hectoliter. In the German salines the mining costs 1.18 fl. per cu. meter of rock which is leached in the works. One hectoliter of Alpine brine obtained by the German method would cost 12.2 kr.

Hallstadt, the largest producing mine, has 30 chambers at work, 12 in preparation, and 18 in reserve. Ischl possesses 30 chambers, besides 4 reserved chambers. At Aussee 18 active chambers, 1 reserved chamber, 8 chambers probably caved, and 41 chambers pronounced dead may be counted. Hallein produces brine in 11 chambers and has yet 5 in reserve. At Hall there are 18 working chambers, and 8 chambers used as reservoirs for brine.

In the year 1894 the saline at Aussee produced 603,590 hectoliters of brine of 27% strength; Hallstadt 2,500,000 hectoliters, Ischl 906,050 hectoliters; but Hallstadt and Ischl evaporated only 2,325,610 hectoliters of brine in the salt works; Hallein drew off 716,250 hectoliters and evaporated 735,210 hectoliters; Hall 459,130 hectoliters of 32½% strength. Berchtesgaden in Bavaria is able to produce yearly 1,200,000 hectoliters of brine.

The number of laborers, all men, employed in the Alpine salines are: In Aussee, 88; in Hallstadt and Ischl, 353; in Hallein, 193; in Hall, 131.

The chambers are laid out in plan, one above the other, so that in vertical section two or more eventually unite, the pillars left between each series supporting the roof. The pillars amount to 51–60% of the ground and involve a loss of 32.6% of the salt; 22.5% is lost in the residual clay, and 44.9% is extracted. Consequently the yield of the saliferous clay is only 18.9% salt, and may be as low as 5%. The brines contain:

	Hallstadt.	Hall.		Hallstadt.	Hall.
	Per Cent.	Per Cent.		Per Cent.	Per Cent.
NaCl	22.36	25.28	NH_4Cl	... ...	0.01
$CaSO_4$	0.19	0.23	$MgBr$	Trace.	0.005
$MgSO_4$	0.91	0.01	$NaSO_4$	.....	0.23
$MgCl_2$	0.95	0.22	KCl	0.93	0.01
$CaCl_2$	.....	0.22	H_2O	74.66	73.775

Hallstadt and Ischl have 16 kilometers of wooden tracks and 400 meters of iron; Hallein 0.8 km. wooden; Aussee 6⅓ km. wooden and 1.1 km. iron; Hall 18½ km. wooden and 300 meters iron.

The brine is piped from the mines to the boiling-houses. The length of the pipe-lines in kilometers is at Aussee 9¾, Hallein 4.1, Hall 13, Hallstadt and Ischl, from where the brine is partly conveyed to the boiling-works at Ebensee, 58⅙ kilometers. At Hallein there are 10 brine reservoirs holding 1350 cu. meters, and at Ebensee 43 with a capacity of 3540 cu. meters.

Weak brines are frequently conveyed into elevated reservoirs, from which they are suffered to trickle over bundles of shrubs or thorns built up into very long walls of considerable height, exposed to the rays of the sun or to the wind. The great amount of evaporating surface thus obtained causes the concentration to go on very rapidly. Sometimes the process must be repeated to bring the brine up to a suitable strength (32%) for boiling. After this the brine is evaporated in vats or pans, by which either coarse or fine-grained salt may be obtained. In conducting the evaporating process slowly coarse salt results, but only small quantities of this are produced. It is the very fine-grained salt which is desired, for making which the brine is constantly stirred and rapidly evaporated. The fine-grained salt is formed into small bricks weighing one kilogram, or keg-shaped blocks of 18 kilograms weight. Finally, the salt is dried in heated chambers. The boiling establishments have the following plants:

Establishments at	Evaporating Vats or Pans.	Area of all Vats in Square Meters.	Drying Chambers.	Drying Area of the Chambers in Square Meters.	Volume of Drying Chambers in Cubic Meters.
Hallstadt, Ischl, Ebensee	13	2330	49	2042	.....
Hallein	4	800	4	892	.....
Aussee	5	675	26	368	1600
Hall	7	.....	12	.....	.....

The bittern remaining after the precipitation of common salt in the boiling-pans shows by analysis the following constitution at Hallstadt: NaCl, 17.5%; KCl, 4.0%; $MgCl_2$, 4.6%; $MgSO_4$, 2.8%; $CaSO_4$, 0.1%; NaBr, 0.1%; total salts, 27.1%; H_2O, 72.9%.

The following table shows the other conditions of the boiling-works in 1894:

Establishments and Temperature of the Vat-brine at which the Salt Settles.	Salt Settled in 24 hours on 1 Square Meter of the Vat's Bottom.	Average Settling Bottom of the Vats.	Fuel Used.	Salt Produced	
				With 100 kg. Wood Burned, in Kilos.	With 100 kg. Brown Coal Burned, in Kilos.
Hallein ..... 100° C.	71–105 kg.	119 sq. m.	Brown coal.	.....	119–176
Ebensee, ordinary pans	94	215	Brown coal.	.....	117
Ebensee, pans heated with refuse steam	27	6	Brown coal.	.....	166
Ischl	92	182	Brown coal and wood.	138	127
Hallstadt ..... 100° C.	90	278⅓	Brown coal and wood.	135	119½
Aussee ..... 96° C.	85⅝	136	Brown coal and peat.	.....	114

Establishments.	Production of the Mine. Rock Salt in Tons.	Employees.				Production of the Boiling Works in Tons.		Value of the Salt Production in Florins.
		Males.	Females.	Boys.	Total.	Common Salt.	Salt for Manufacturing Purposes.	
Ischl, Ebensee.....	214	828	25	2	855	71,931	5,473	7,008,420
Hallein.....	.....	173	2	....	175	21,687	672	2,072,854
Aussee.....	207	290	4	8	302	18,741	1,792	1,681,608
Hall.....	11	116	.....	....	116	14,958	1,092	1,231,042

The Tertiary Salt Deposits.—The Lower Tertiary formation in that part of Austria where salt deposits are known is allied with the antecedent Cretaceous formation, and petrographically the formations are not distinguishable. The Cretaceous and Lower Tertiary or Eocene strata are mostly arenaceous deposits successively accumulated under similar circumstances. In former times, before the age of the Eocene sandstones passing imperceptibly into the subjacent Cretacean sediments was ascertained, the name of Carpathian sandstones or Carpathian strata was proposed for that series, deposited chiefly in the Carpathian Mountains and their lateral ranges. Now the full correlation of the Cretaceous and Eocene group is established. At the close of the Eocene period a great upheaval took place, and the Carpathian range received partly its present features. After this a different Upper Tertiary sea—the Tertiary Mediterranean—deposited its strata unconformably to the Eocene (Carpathian) sandstones on the northern, northeastern, and southern slopes of the Carpathian range, namely, in Galicia (Bukovina), Moldavia, Hungary, and Transylvania. All the other Upper Tertiary strata in Austria are barren of salt. The Upper Tertiary or Miocene series are classed into the first and second Mediterranean division, and an upper division, the Sarmatic. The immediately succeeding Pliocene strata are mostly fresh-water deposits.

The northern deposits are marine; the southern, which occupy also large areas in Transylvania, present a mixture of marine with tufaceous or volcanic strata. Along the ancient coast of the Lower Miocene sea, salt was deposited in lagoons. In the first (and partly second) Mediterranean series a group of more laminated deposits containing some gypseous clays and sands is to be observed, and is called *Shleer*. This *Shleer* is more strongly developed in the Carpathian section, and bears the saline deposits which may be divided into those of the northern and southern declivities of the range.

North Carpathian Miocene Salt Deposits.—Along a trend parallel to the Carpathian summit at a distance of about 50 to 60 kilometers, beginning near Cracow and ending in Moldavia in a length of 700 to 800 kilometers, of which 550 is in Austria, a zone can be traced where great lenticular masses or layers of salt, mostly inclined at small or high angles to the north, occur. In Galicia (and Bukovina) about 200 brine springs indicate the salt deposits, which, however, are worked only in 12 mines. The two principal mines are in Western Galicia at Wieliczka and Bochnia, both east of Cracow at distances of about 15 and 40 kilometers. The other mines in Eastern Galicia are situated in the districts of Drohobicz and Stanislav.

The salt deposits of Wieliczka extend over an area of about 3½ kilometers in length from west to east by nearly 800 meters in breadth from south to north,

lying directly beneath the town. Towards the west the salt in the Miocene strata thins out suddenly, and no salt indications west of Wieliczka can be recognized. The region exploited by mining extends over an area of about 80 to 90 hectares.

The superincumbent beds are saliferous sandstones and clays with thin seams of very fine granular or cryptocrystalline anhydrite. Sometimes these beds are accompanied with barite, gypsum, celestite, and native sulphur, the last mostly in earthy varieties. Anhydrite and marl are also imbedded in the salt strata.

In these sandstones and clays are two salt-bearing series. The lower, the Szibik, is composed of a more or less transparent white or grayish-white pure coarse crystalline or granular salt, varying in thickness from 2 to 8 meters. The impurities of this rock salt are admixtures of anhydrite or clay usually less than 1% and seldom 1.5%. Another layer above the Szibik salt is the Spiza, attaining a thickness of 20 meters, but its salt is contaminated with more impurities than the lower bed. In the Spiza salt beds are found many organic remains which permit a comparison of these strata with the other marine deposits of the first Mediterranean division in the Miocene formation. Hence no ambiguity remains respecting the precise chronological relations with the Lower Miocene series.

A pure variety of the Spiza salt is that which decrepitates when dissolved in water, owing to its tenor of gases derived from organic matter. One kilogram disengages 60 c.c. of gas containing C_2H_4, 20. 9%; H,10.2%, and CO, 8.0%. The decrepitating salt is found in some places in banks of 0.33 to 4 meters thickness, and when cut by drifts inflammable gas is set free. Between the higher Spiza salt layer and the lower Szibik salt intervene beds of marls and clays with contorted bands of anhydrite. The Spiza salt layer is immediately overlain by indefinite strata of sandy clay 90 meters thick, but thinning out in a westward direction to 30 meters; it is saliferous clay of gray or in some places of brown color, analogous to the *Haselgebirge*. In this are scattered irregularly shaped masses of salt, sometimes 30 to 40 meters thick and more than that high.

The *Haselgebirge* with its irregular-shaped masses of green salt is overlain by barren clays with a layer of gypsum, to which succeed marine, exceedingly fine sandy clays (*Tegel*), sands and marls barren of salt.

The other salt deposits on the northern or northeastern side of the Carpathian range display geological features similar to those of Wieliczka.

Bochnia.—The saliferous region near Wieliczka extends farther to the east, where at a distance of about 20 kilometers another saline, also remarkable, is worked to a depth of 400 meters on a lenticular mass of a length of 3700 meters, in breadth varying from 100 to 200 meters on its eastern extension to 150 to 200 meters at the western limits.

The deposit is composed of saliferous clay with anhydrite. In the upper part of it the clays alternate with interstratified laminæ of massive anhydrite of bluish-white color. In the lower part similar laminæ of gypsum prevail. The salt is interstratified in the brownish-red clay near the outcrop in five lenticular layers which can be traced in horizontal and vertical extension to the limits of about 140 and 120 meters. But the saltbeds are curved, thin out or swell up, and vary much in their size; also forking into two or more layers or uniting with one another. With the saltbeds alternate either brown-red clays or dark gray saline clayey marls with interstratified flat beds of whitish-blue anhydrite.

The saltbeds attain a thickness of 3 meters, the average being $1\frac{1}{3}$ meters. The salt is coarse, granular, white to gray, ribboned with pale green and gray bands, and one variety of it resembles the Szibik salt of Wieliczka. The salt deposits extend beneath the town of Bochnia, covering 33 hectares.

In Eastern Galicia there were formerly 92 large and 10 small salines, but to concentrate and control the production they were reduced to 9; the others have been left unproductive. A tenth saline is in the Bukovina near the Galician boundary.

At No. 1, Latzeko, the formation is *Haselgebirge* with pure or impure saltbeds of wave shape, deposited on bluish-gray clays (*Tegel*) with gypsum and sandstones, and covered by bituminous clayey slates. The average yield of salt in the deposit is 58.5 to 68%; of the rest 41.5 to 32% is clay and sand and 0.5 to 1.4% water.

At No. 2, Stebnik, the salt deposit is basin-shaped. The thickness of the salt layers is not known exactly, but is very great; the thickness as known is over 125 meters. The *Haselgebirge* as it is worked is a dirty, impure salt mixed with 34.5 to 48% of its bulk of sand and clay and 1.75 to 2% moisture.

At No. 7, Kaloosh, the saliferous clay of 120 meters thickness contains from 43.5 to 62.25% of its bulk rock salt. The rock-salt beds themselves contain 76 to 99.8% pure salt, 22.5 to 0.1% clay, and 0.4 to 1.2% moisture. Some upper beds contain 1.5% chloride of potassium. In the upper part of the saliferous deposit, in a band of 14 meters, occur large nodules or short lenticular slabs parallel to the stratification of coarse white sylvinite mixed with blue halite. In the strata superimposed on the salt layer appears a layer of kainite partly mixed with clay and partly composed of pure granular segregations. This layer, of 8 to 16 meters thickness, is accompanied by intercalated lenticular beds of sylvinite from 2 to 30 meters in length and 2 meters thick, mixed with coarse crystalline blue halite and a small quantity of anhydrite. The upper kainite-bearing layer, if pure and free of clay, is composed of about 60% granular kainite, 10% sylvinite, and 20% rock salt. The pure kainite found sometimes in crystals in cavities has the composition SO_3, 39.8%; MgO, 9.7%; KCl, 23.5%; H_2O, 26.8%. The kainite varies in color from grayish to yellowish, and is sometimes, when in beds, reddish from a slight admixture of carnallite. The kainite is also accompanied, but more seldom, with glass-like, coarse grains of picromerite or schönite, and syngenite sometimes occurs. The sylvinite, which is thought to be a product of decomposition of the kainite, becomes more rare in Kalusz, but the kainite deposit is promising, though its content of potassium oxide varies only from 10 to 12%.

But at No. 5, Dolina, or better at Turza, at a distance of 17 kilometers, a borehole found at the depth of 507 meters about 200 meters saliferous clays yielding 69 to 75% pure salt, and in its upper part kainite with other potassium salts, yielding 7 to 12%, and in some beds even 17 to 18% K_2O.

The saline No. 9, Kossow, is the last of the Galician mines in a southeastern direction; the saliferous deposit, known to a depth of 155 meters, dips west-southwest at an angle of 60°, with a thickness of about 60 meters. The rock salt resembles the Spiza variety and contains 94 to 98% pure salt, 5.33 to 1.75% clayey matter, and 0.25 to 0.33% moisture.

The other salines at work, as No. 3, Drohobicz, No. 4, Bolechow, No. 6, Delatyn, No. 8, Lanczin, are like the others, and nothing particular can be said

of them. The last mine in the Bukovina, No. 10, at Kaczyka, close to the Galician boundary, works on a deposit of 230 meters thickness, whose salt yields 80 to 97% chloride of sodium and other soluble salts. But the rich beds contain salt 93.5 to 98.8%, clay 4 to 1%, and moisture 0.3 to 0.1%. The salt in its texture is like the Szibik variety. Beneath the salt deposit is a group of sandstones, grits, and conglomerates; the incumbent beds are formed of clay with gypsum, covered by sandstone, shales with earthy and massive sulphur, and sometimes with layers of lignite.

The Extraction of the Salt.—The stone implements found in the salt mine Ootorop, now idle, lead us to believe that the first traces of salt extraction in Galicia are of greater antiquity than in the Alpine mines, and that the tribes which used stone tools were acquainted with salt mining.

Though the mines of Wieliczka are old, it is not known when they were discovered, but they were first mentioned in the eleventh century. In the fourteenth century the Polish kings established regulations. The kings of Poland depended upon the salt mines for the dowries of their queens and the endowment of convents; the Polish noblemen also stipulated at each royal election that the rock salt should be supplied to them at the cost of extraction. In the year 1772, when the dismemberment of Poland took place, Austria obtained the Galician salines.

Before mentioning the methods of salt extraction, it may be stated that an impurity of 3% in the rock salt renders it unfit for domestic purposes, especially if the admixtures consist, not of clay or sand, but of chlorides of calcium and magnesium, particularly the former. These by-salts render it liable to deliquesce. The rock salt is worked in the mines when the amount of impurities, mostly clay, attains the rate of 4%; a higher rate of admixture begins to destroy its transparency. The percentage of impurities in the rock salt determines finally the method of extraction. Deposits yielding less than 96% pure salt are worked by extraction with water.

The mines at Wieliczka are entered by 4 hoisting, 2 pumping, and 2 ladder shafts, besides 13 vertical downcasts below ground, and are opened by 7 levels. The total length of drifts and galleries is about 80 kilometers. The deepest point reaches 286 meters below ground or 51 meters below the level of the sea. The dimensions of the drifts are at least $2\frac{1}{3}$ meters high and 2 meters in width, as also in Bochnia, but they are, when driven in salt, still more spacious, and timbered only exceptionally.

In former times the great green salt masses were worked out fully. The impure massive rock salt enveloping the green salt masses is thick enough to secure the chamber caused by the removal of the green salt from caving. One of these immense excavations was in active operation from 1717 to 1861 before its salt was exhausted, leaving an open chamber about 95 meters high.

Now mining is carried on by the pillar and stall method. The thickness of the salt deposits being so great, this method of extraction is executed in several stories separated, if possible, by a horizontal layer of impure salt, the pillars being arranged one above the other. The salt in the Galician mines is broken out by making deep cuts, horizontal and vertical, in the working face, whereby huge blocks of salt are easily wedged off. The large blocks are afterwards cut into smaller blocks for market.

The salt mass being exceedingly tough, all the rooms stand without filling; only trickling water could injure the safety of the mine. It is seldom that a log is used as prop or otherwise. If there is need to support parts of the roof, particularly in old excavated chambers, props are built up of cribbing or of rectangular blocks of salt. Sometimes walls of large salt-blocks are filled inside with refuse salt, and soon the mass becomes cemented together. The timber used in salt mines needs no renewing, as the wood never rots.

In the mines of Bochnia, which have been in operation for upwards of eight centuries, there are 4 hoisting-shafts, 1 climbing-shaft, and 3 vertical downcasts below ground. The layers are extracted by stoping. But if a layer in the salt-beds alternating with saliferous clay thins out to 67 cm. in thickness, it is said not to be workable. A miner extracts on an average in a shift of 8 hours 5.6 metric cwts. of salt blocks, and refuse salt corresponding to 0.26 cu. meter.

The other salt deposits in Western and Eastern Galicia, with the exception of a few, are composed of layers of rock salt contaminated with 3 to 4% of clayey and sandy impurities; they are therefore unfit to be worked as rock salt, but being very rich and large, the salt can be extracted by the leaching method.

The salt mined in the dry way is either hoisted, as in Kossov, and lixiviated above ground, or the mines opened by a shaft or a bore-well are filled with water, which is allowed to stay there until it has dissolved enough salt to make a concentrated brine. Or else chambers like those in use in the Alpine salt region are opened. The brines are composed as follows:

	NaCl.	Other Salts.
Lacko	24.5%	0.25 to 2.0%
Stebnik	24.1 to 25.2	.6 to .7
Kossow	25.7	1.5 to 1.7
Kalusz	25.8	1.5 to 2.0
Kaczyka	25	1.5 to 1.7

The soluble salts in the brine reduced to 100 show the composition as follows:

	Kossow.	Stebnik.	Kalusz.
	Per Cent.	Per Cent.	Per Cent.
NaCl	97.9	97.2 to 97.4	89.1 to 91.5
$MgCl_2$	0.2	.6 to 1.1	0.6 to 1.0
$CaCl_2$	0.4	0 to 0.2	0.01 to 0.02
K_2SO_4	.....	0 to 0.1	.....
$MgSO_4$	.....	0 to 0.6	.....
$CaSO_4$	1.5	1.4 to 1.5	9.8 " 7.2
$CaSO_3$	....	0 to 0.1	....
Total by-salts	2.1	2.3 to 2.8	11.0 to 7.9

Salt is mined and extracted in brines at Kalusz and Kaczyka. At Kalusz the potassium salts in the hanging wall of the salt-bearing deposit are mined in the common way with galleries. Only the kainite deposit is a promising one; the attempt to mine the accompanying sylvinite did not result successfully.

The brine lifted from the mines of Wieliczka and Bochnia in the year 1894, at the rate of 4870 and 344 hectoliters, respectively, was formed by the trickling of water in the mines. The salines in Eastern Galicia in the district Drohobycz (Lacko, Stebnik, Drohobycz, Bolechow, Dolina) drew 1,135,700 hectoliters brine; those in the district of Stanislaw (Delatyn, Kalusz, Lanczin, Kossow) pumped 658,520 hectoliters and Kaczyka in Bukovina 653,180 hectoliters. The number of

laborers (males only) occupied in salt mining is in Western Galicia (Wieliczka and Bochnia) 1253; in Eastern Galicia in 5 mines of the Drohobycz district 97; in 4 mines of the Stanislaw district 64; in the one saline in Bukovina 63.

The brine is evaporated in boiling-houses; in all there are 19 pans and 4 reserve vats in Galicia and 2 pans in Bukovina, besides 63 and 6 drying chambers, respectively. The bottom area of all vats in square meters is 1890; of all drying chambers 510 sq. meters. To convey the brine there are 7 kilometers of pipe-lines in Galicia and 1⅓ kilometers in Bukovina, and there are 30 brine reservoirs with a capacity of 6100 cu. meters.

Salt evaporating is conducted to produce coarse salt and fine-grained salt. The boiling salt at Stebnik dried at 160° C. has the composition: NaCl, 96.8 to 98.5%; $MgCl_2$, 1.2 to 0%; $CaCl_2$, 0.02 to 0%; Na_2SO_4, 0.1 to 0%; $CaSO_4$, 1.4 to 1.1%; K_2SO_4, trace to none; H_2O, 0.8 to 0.5%. The total of the by-salts is 2.7 to 1.1%.

The bittern or residual liquor after the precipitation of coarse salt or fine granulated salt has the following composition:

	Stebnik.	Kalusz.
NaCl	94.1%	26.1%
KCl	0.6	....
$MgBr_2$	0.5	0.3
$MgCl_2$	22.2	62.4
K_2SO_4	2.1	4.9
$CaSO_4$	0.5	....
$MgSO_4$	....	6.2

All the 10 evaporating establishments in Eastern Galicia and Bukovina settle the salt in the boiling-pans at the temperature of 105° C.; one square meter of the evaporating vats precipitates in 24 hours 142 kgms. coarse or fine stirred salt. The average settling bottom of one of the vats is equal to 59.7 sq. meters. The fuel used is wood, and with 100 kgms. of it 136 kgms. of salt are produced.

The laborers engaged in salt boiling-houses are all males; in the Drohobycz district with 5 establishments the number is 259, in the Stanislaw district with 4 establishments 196, in Bukovina (Kaczyka) in one establishment 18.

The production of salt was as follows in the year 1894:

	Rock Salt, Tons.	Common Salt, Tons.	Salt for Manufacturing Purposes (Refuse Rock Salt), Tons.	Value of Production in Florins.
Western Galicia (Wieliczka, Bochnia)	37,135	....	62,497	3,287,310
Eastern Galicia, district of Drohobycz, 5 establishments	....	33,183	....	4,129,290
District of Stanislaw, 4 establishments	....	15,799	....	2,986,220
Bukovina, 1 establishment	2,210	2,151	80	385,460

The mine of Kalusz produced besides this 350 tons in lumps and 743 tons ground kainite, sold as fertilizing salt for 10,930 fl.; but the price per ton is since the beginning of 1897 fixed at 7 fl., instead of 10 fl. as before.

The salt which is destined to serve for manufacturing purposes is either nearly pure refuse salt, or salt contaminated with earthy substances. The salt for cattle has 91 to 96% NaCl; 0.6 to 0.9% moisture; 3.3 to 7.7% insoluble matter. At the present time the latter salt is mixed—or, as it is called, denaturalized—purposely with peroxide of iron and the bitter herb artemisia, to make it unfit for culinary purposes.

The South Carpathian Miocene Salt Deposits.—These deposits are also in the Lower Miocene series, or the first Mediterranean division, and trend along the southern slopes of the Carpathian range through the northeastern part of Hungary to Transylvania, where they are extensively developed. They form also separate basins in other parts of the country, distant from the Carpathian Mountains. From Transylvania and Bukovina the salt-bearing Miocene strata can be traced to Moldavia, where the deposits are also rich. But it may be observed that in the sub-Carpathian salt series in Moldavia the older strata of the Cretacean and Eocene period, consisting of sandstones and ash-colored clayey marl shales in highly disturbed positions, are connected in a confused way with salt and naphtha-bearing strata, probably Miocene. Consequently the true geological horizon of the salt deposits, which is conjecturally also Lower Miocene, cannot be exactly distinguished. The character of the salt-bearing strata in the valley of the Trotush, a tributary of the Sereth, at Sanik, Tirgu-Okna, at the old mines of Grozesti, and other places, as Beznuc, Breteela, Negoesti with its sandstones, gypsums, and salt clays, is similar to those known in Austria, though they do not clearly show their geological age.

In Hungary in the Miocene basin of the Hernad valley in the district of Sharosh a thick saltbed is known, and is worked at Shovar (meaning salt-town) southeasterly from Preshov.

Farther to the east, from Hust to Borsho in the upper Tisa valley, there is a Miocene basin in deep Eocene basin-shaped depressions, which is separated to the southward from the great Hungarian plain by the trachytic range of Vihorlat-Gutin. This occurrence is known as the Marmaros basin in the district of the same name, the town of Marmaros-Szigeth being the seat of the mining administration. As far as geological observations have been carried, the strata in the hanging wall of the salt deposit are soft marls, sandstones, clays, with volcanic (trachytic) tuffs and semi-tufaceous sands, with gypsum, anhydrite, and fragments of carbonized wood. Underneath these strata, but by no means deep below the surface, horizontal beds, or rather immense masses of salt in basin-shaped form, are deposited in several places. The thick salt deposits in the Marmaros Miocene basin cover an area of about 400 hectares. The beds are composed of crystalline, white or transparent rock salt in which layers of sandy or clayey salt form parallel horizontal partings and pass insensibly into pure rock salt. In some places volcanic tuffs form partings, blending also with the salt. The thickness is not exactly known, no foot-wall of the rock-salt layers having been met with. The thickness of the salt layer at Ronaszek is about 160 meters, though the salt is mined only from a depth of 120 meters. At Slatina the bed is more than 140 meters thick; at Shugatag more than 100 meters. Other mines not now worked are at Senesh, Baranya, Sanderfaln, and Königsthal, where there are layers of salt of great thickness. The salt in the mine at Ronaszek has no trachytic (tuff) partings, and all through the vast grayish mass of rock-salt beds of transparent crystalline or coarse granular white rock salt, measuring some meters in thickness, are met with. Some saltbeds discharge inflammable gases freely, and this gas was used in the Slatina mine to light it. In some salines, but mostly in Shugatag in different saltbeds, fragments of carbonized wood and lignite occur.

In the three salt mines actually operated, salt of all grades of purity is found.

The highest grade beds giving salt used for domestic purposes have the following composition: NaCl, 98.9 to 99.9%; $CaCl_2$, 0.1 to 0%; $CaSO_4$, 0.5 to 0%; $NaSO_4$, 0.03 to 0%; insoluble, 0.3 to 0%; water disengaged above 160° C., 0.2 to 0.1%.

Transylvania.—Of all Austrian provinces this is best provided with salt. The beds with indurated clays and sandy tuffs, in which, as in Hungary, the thick salt masses are deposited, form either basins in the neighborhood of trachytic mountains, on the southwestern Carpathian slopes drained by the Samosh River (Deshakna), or the little Kokel River (Parajd), or are situated in or near the Marosh valley (Torda, Marosh-Ujvar, Vizakna) in central or southern Transylvania. The strata on which the salt deposits are superimposed are not exactly known, because the thickness of the salt layers is very great and the mines have nowhere penetrated to the foot-wall. The salt layers crop out in hills in 40 places as naked rocks on the surface; and there are known numerous springs—192 strong salt springs and 593 weaker salt springs—rising from the Miocene strata, besides many salt ponds and salt marshes. Salt is won only in eight mines at the five above-named places. Many of the older mines, as Sek, Kolosh, and others, have been closed to concentrate the working in a few mines. From this fact one may judge of the great extent of the salt deposit in Transylvania. The salt deposits as they are worked in the mines are pure though interstratified with bands contaminated with impurities. It is said, or rather conjectured, that some of the salt layers become in the lowest portion more impure and mixed with clay and sand. All the salt fit for culinary purposes from four mines (except Vizakna) yields: NaCl, 98.5 to 99.9%; $CaCl_2$, 0.1 to 0%; $CaSO_4$, 0.9 to 0%; $NaSO_4$, 0.4 to 0%; insoluble in water, 0.6 to 0.2%; water above 160° to 170° C., 0.4 to 0.1%.

The rock salt from Vizakna is the least pure. This, when fit for table use, averages: NaCl, 97.4 to 99.7%; $CaCl_2$, 0.1 to 0%; $CaSO_4$, 0.6 to 0%; $NaSO_4$, 0.3 to 0%; insoluble in water, 2.6 to 0.3%; H_2O at 160° to 170° C., 0.2 to 0.1%.

The Deshakna mine (*akna* or *okna*, in the Wallachian or Roumanian tongue, means saline) in the district of Solnok-Doboka, to the north of Samosh-Ujvar, and in the Samosh valley, where 10 kilometers to the east croppings of salt cliffs are exposed on the river, has a regular layer of salt which is thought to be 110 to 120 meters thick. It is overlain by saliferous clays, and marls alternating with trachytic tuffs, while underneath the salt deposit the strata are not known, but it is thought they may be gypsum, marls, and tuffs. The deposit has on the surface a rhomboidal shape; it is 1700 meters long and 1150 meters broad. Near the main deposit is situated another, 380 meters long and 215 meters broad.

Parajd in the district of Udvarhely east from Marosh Vasharhely, on the Little Kokel River, has a deposit explored to 170 meters depth, all in salt. On the surface it covers an area whose greatest dimensions in length and width are 2170 and 1150 meters. On the Kokel River rock salt forms high naked cliffs 70 meters above the river level at Sovata. This salt deposit, which is not worked, is known to a length of 1800 meters, a width of 592 meters, and a depth of 70 meters. The salt deposit is connected with overlying sandstones and trachytic conglomerates.

Torda, also Thorda Akna (in German, Thorenburg), is in the district of Torda-Aranyosh southeast from Koloshvar. The two salt deposits worked here have on the surface a lenticular shape; the greater one is known to a depth of 217 meters,

the smaller only to 4 meters; the length and width at the largest part of the larger are 3330 and 570 meters respectively, and of the smaller 760 and 610 meters. The fossils found in the saline clay show the series to be contemporaneous with those of Wieliczka, Galicia.

The salt deposit at Torda is a peculiar one; the bands or the bedding being not as nearly horizontal as in the other saltbeds, but disposed in zigzag lines with repeated sharp bends. In the salt marls bordering the beds a copious discharge of marsh gas is observed.

Marosh-Ujvar in the Marosh Valley, in the southwestern portion of Transylvania in the district of Bros (Sasvarosh) (in Magyar, Alsho-Teher), has a salt deposit of oval shape 900 meters in the longer and 815 meters in the shorter diameter. The mining is conducted to a depth of 125 meters.

The mines at Vizakna (in Roumanian, Akna; German, Salzburg), northeast from and near Hermannstadt or Nagy Seben in the district of Alsho-Teher, are a deposit 1430 meters long, with an extreme breadth of 690 meters and a depth known to 190 meters. The saltbeds are in some spots mingled with fragments of lignite or brown coal.

Since about 1850 the mining of salt in Transylvania has been conducted by excavating large oblong chambers, of which two are always worked out side by side, being separated by pillars of salt. The chambers are connected by gangways with winding and draining shafts, sunk in the salt. By this system of mining the chambers are safe in case water should be tapped.

The chamber work begins with a large drift, from which inclines are driven. The salt is worked out from the side faces and then the bottom is stoped downward in much the same manner as at Wieliczka. The small salt is used in chemical manufacture.

Mining can be conducted at a minimum of cost when the bottom of the chambers is entirely dry. The productive capacity of the mines worked by the chamber method depends on the area of the chambers in which the mining is performed. The following tabulated statistics of the Hungarian and Transylvania rock-salt mines show besides the plant the bottom area or extension:

Rock-salt Mines.	Chambers.	Working Floor Area in Square Meters.	Adits.	Shafts.	Steam-engines for Hoisting and Pumping.		Horse-whims.	Horse Power.	Salt Extraction.		Workmen Engaged.	Officials.
					Number.	Horse Power.			Rock Salt Salable, Tons.	For Manufacturing Purposes, Tons.		
Hungary: Ronasek	6	56,670	16	53	3	46	7	51	17,581	.....	374	13
Slatima	4	21,335	13	29	10	165	2	10	35,157	4,668	461	11
Shugatag	2	35,000	12	9	.....	.....	6	24	16,810	336	336	8
Transylvania: Deesakna	2	4,840	4	7	.....	.....	2	6	9,823	.....	180	6
Parajd	5	8,670	6	3	.....	.....	1	8	3,655	.....	99	4
Torda	3	10,710	5	3	.....	.....	2	14	2,051	2	46	3
Marosh-Ujvar	6	44,900	26	23	13	297	.....	.....	54,467	135	580	13
Vizakna	3	5,240	1	3	.....	.....	1	6	3,173	.....	75	4

Besides the rock-salt mines there is also one brine-extraction saline in Shovar. The brine is extracted at the depth of about 140 meters out of an old conical excavation that crushed in at its apex more than a century ago, and became flooded, the means of draining at that time being inadequate to cope with the water. The brine is boiled in two establishments (employing 86 laborers and 6

officials), in 3 evaporating vats, in which 6891 tons coarse salt is settled. One square meter of the boiling surface precipitates 34 kgms. salt, the average area of each vat being 83 sq. meters; 100 kgms. of the oak wood used as fuel produces 164 kgms. salt.

All the salines in Austria (and Hungary) which evaporate brine by artificial heat used fuel as follows in the year 1894: Pine wood, 109,457 cu. meters; oak wood, 610 cu. meters; brown coal, 47,309 tons; peat, 1232 tons.

SEA-SALT OR BAY-SALT.

Of the sea-salt industry there is not much to say, though it is of considerable importance for the southern provinces, the output reaching a large amount per annum and giving employment to thousands of men during the summer months. In Istria there are two sea-salt works at Pirano and Capo d'Istria, on the seashore of Dalmatia, one at Stagno, and two on the islands Arbe and Pago. The latter belong to private parties, but the salt extraction is inspected by soldiers appointed by the government, to which the salt must be delivered at the cost of extraction.

The process of sea-salt-making is simple, as the evaporation takes place by the heat of the sun in shallow reservoirs. The methods pursued here are all similar. Originally the salt ponds or reservoirs were simply lagoons on the marshy coast, but with time they were enlarged into artificial ponds with clay bottoms, sometimes with a dam built across the entrance to a small cove on the seashore. In the artificial sea-salt works embankments or dikes were constructed, furnished with gates which could be opened or closed at pleasure. The interior portion is subdivided into several shallow basins in which the salt is obtained by solar evaporation. The large number of hands employed by the sea-salt works is due to the shortness of the season, for the evaporation of water can be conducted only in the driest and hottest months of the summer and the work must be done quickly.

The refined sea-salt of the Istrian establishments shows the following chemical analysis: NaCl, 98.1%; $MgSO_4$, 1.3%; $CaSO_4$, 0.7%; $MgCl_2$, 0.2%; H_2O, 0.6%; insoluble matter as clay, limestone, and oxide of iron, 0.1%.

The following table shows the statistics of sea-salt extraction for the year 1894:

Salines.	Salt-works.	Hands Engaged.				Salt Production.		Worth of Production, in Florins.
		Men.	Women.	Boys.	Children.	For Culinary use, in Tons.	For Industrial Purposes, in Tons.	
Istria	2	1,192	525	1,138	739	42,000	3,211	3,287,310
Dalmatia	3	1,522	446	.....	140	9,286	.....	455,350

SALT IN BOSNIA.

In addition to this description, a few words on the salt of Bosnia, now under Austrian rule, may be given. In the northeastern part of this country, in the section drained by the Sprecha River, a tributary to the Bosna River, at Dolni

Toozla salt springs were known from an early date, and the weak brine was used. The geological features are like the Carpathian salt deposits.

The natural salt springs at Dolni Toozla, lying on the Tala rivulet, flowing towards the Sprecha, rise in wells of from 10 to 13 meters in depth. The strength is 5.5 to 6% salt, the flow varying from 1.71 to 1.77 cu. meters per hour. The brine is boiled in copper vats with wood fuel, and coarse-grained salt saved, of which on an average 140 tons yearly were produced, paying a slight royalty to the Turks. The salt industry had fallen into a low condition under the Turkish rule. But in the years 1883–84 deeper wells were bored to 380 meters, to the salt-yielding Mediterranean group, which furnished brines at Gorni Toozla with 24%, at Dolni Toozla with 14.1%, 18%, 20.7%, and 25%, and at Simin-han, not far distant from Dolni Toozla to the east, even with 26.7% salt. At the depth of 221 meters a salt layer was perforated. At the same time the two evaporating works at Dolni Toozla and Simin-han were erected.

The brine of Dolni Toozla from bore-holes shows the constitution: NaCl, 21.2%; Na_2SO_4, 3.8%; K_2SO_4, 0.06%; $MgSO_4$, 0.01%; CaSO , 0.003%.

The salt-evaporating works at Dolni Toozla and Simin-han are constantly enlarging their plant. Their production from 1885 to 1894 has been, in tons: 1857; 2362; 3180; 4944; 5357; 4991; 5970; 8007; 8551, and 10,250, respectively. The number of hands employed is 150. The residual liquor from the coarse salt manufacture at Simin-han shows the constitution: NaCl, 23.8%; Na_2SO_4, 4.1%; K_2SO_4, 0.1%; $MgSO_4$, 0.2%; $CaSO_4$, 0.04%.

In Herzegovina salt springs exist at Koinica, but the precise conditions of the occurrence are not well known.

THE ADMINISTRATION OF THE SALINES.

As salt is a necessary of life, both for man and beast, it has in many countries been a favorite subject of taxation or monopoly, and among these countries is Austria. To support this monopoly a high tax is levied on all the salt produced. Hence table salt is costlier in Austria than in almost any other country, and the naturally impure salt and the salt purposely mixed with impurities for cattle, though somewhat cheaper, is sold with restrictions and impositions on the part of the state agents. Since salt is a state monopoly, no one can ask a concession for mining salt, which is conducted by state employees exclusively. Where there are salt springs they are either fenced and covered, or their use is forbidden. Salt ponds, as in Transylvania, may be used only as bathing-places, and all these places, particularly the dumps of refuse salt or saliferous country rock, are surrounded by walls and watched by sentinels. In the year 1894 the proceeds from the salt monopoly amounted to 22,128,780 fl. In Western Austria the mining expenses amounting to 3,080,460 fl., and the administration absorbed 237,920 fl., leaving a net profit of 18,810,400 fl., which is really a tax on salt. All expenses of salt extraction per metric cwt. (100 kgms.) were 0.987 fl. The retail price of 1 cwt. salt for table use is 13 fl., and the wholesale price for cattle salt 5 fl. (the latter since 1897); the wholesale price of table salt at the pit's mouth or the storehouses at the boiling establishments varies from $7\frac{1}{2}$ to 9 fl. per cwt. This high

price and the restrictions practiced in the delivery of cattle salt for the trade explain the fact that the proportional consumption of salt for each person in Austria is low, being only 6.5 kgms.; lower than it is in the United States, Great Britain, Germany, or Russia. The salt used for chemical or manufacturing purposes, and for agriculture, shows an annual consumption of 2.76 kgms. per head. In Eastern Austria the figures are very similar.

Many pamphlets and other valuable contributions referring to the Austrian salt mines have been issued. These contributions include valuable observations by Aigner, Herdina, Paul, Nedwiedzki, Posepny, and others. Aigner first published a most complete description of the Alpine salines; the three following increased the knowledge of the north Carpathian salines, and Posepny presented a noteworthy description of those of Transylvania.

SALT IN RUSSIA.

Important deposits of salt were discovered at Bardymkul, in the region of Ferghana, Turkestan, in the fall of 1895. Belgian engineers sent to make an examination estimated that 2 square versts (1 verst = 0.663 mile) contained 36,112,000,000 lbs. of salt. The deposits were leased from the government by a Belgian company and operations were begun in January, 1896. The product, which is of very superior quality, is shipped by camels to Samarcand, and thence by Transcaspian Railway to various points in the empire, being delivered at St. Petersburg for 2.40 rubles per pood, which enables competition with English salt. It is thought that India can also be supplied to a large extent with this salt, especially after the completion of the Samarcand-Kokand-Marghelan Railway, with a line to Krasnovodsk, on the Caspian Sea, the salt beds being only 15 miles from Potar, on the new line, and 1100 miles from Usun-Ada.

SLATE.

THE production of slate, which increased heavily from 1894 to 1895, fell off again in 1896. The reason for this was the less activity in the erection of new buildings and the tendency to defer repairs of old ones, much the same effect having been observed in the paint trade and other allied industries. There was a large number of buildings constructed in 1896 in the large cities, but these require far less slate and paint proportionately than the structures which are put up in smaller cities, towns, and the country. The use for slate by far the most important is for roofing, and in the large cities metal is generally used. Prof. Seamon in the *Engineering and Mining Journal*, Nov. 14, 1896, discussed the question of roofs, and demonstrated that of the material commonly used in this country slate is the cheapest; but in this respect it is considerably surpassed by zinc, which is extensively employed in Europe. Prof. Seamon's data on this subject will be found under the caption "Zinc and Cadmium" in a subsequent part of this book.

PRODUCTION OF SLATE IN THE UNITED STATES.

States.	1895.				1896.			
	Squares.	Value.	Mfrs. Value.	Total Value.	Squares.	Value.	Mfrs. Value.	Total Value.
California	1,350	$9,450	.....	$9,450	500	$3,500	.....	$3,500
Georgia	6,500	27,505	.....	27,505	5,697	22,788	.....	22,788
Maine	23,572	117,388	$20,980	138,368	22,987	99,303	$23,760	123,063
Maryland	13,180	58,914	1,150	60,064	15,498	69,896	1,740	71,636
New York and Vermont	233,523	721,970	124,225	846,195	173,891	525,699	95,670	621,369
Pennsylvania	423,467	1,388,409	227,522	1,615,931	426,172	1,358,297	328,528	1,686,825
Virginia	50,814	169,877	19,000	188,877	53,795	179,339	16,680	196,019
Other States	200	700	.....	700	560	2,040	1,200	3,240
Total	752,606	$2,494,213	$392,877	$2,887,090	699,100	$2,260,862	$467,578	$2,728,440

The falling off in the production of slate would have been much greater if the export trade had not risen suddenly to an unprecedented figure, slate being shipped abroad from both Vermont and Pennsylvania. This unexpected business was due chiefly to the strikes in Wales, and the consequent inability of the quarrymen there to fill their contracts. American producers took advantage of the situation by sending agents abroad to solicit orders, while not a few buyers' agents came to this side. The American slate has apparently made a good impression in Europe, and there is no doubt that much of the new business will

be retained. A good deal of the slate shipped to England was reshipped to Australia and South America, it being cheaper to ship that way than directly from the United States.

During 1896 new slate quarries were opened in Blount County, Tenn., and in Carlton County, Minn. Each of these places shows a fair quality of stone, and it is probable that regular quarrying will be undertaken.

The increase in the production of manufactured slate in 1896 is to be attributed to the increased exportation of blackboards.

The American consul at Ghent writing under date March 10, 1897, states that to introduce American slate in that country it will be necessary to furnish a product of equal quality, and at the price of the French article. Slate is produced in Belgium at Herbemont and Vieille Salm in the province of Luxembourg. In quality the Belgian slate is superior to the French, which is its only considerable competitor. The French is, however, the better in form and appearance. Foreign slate cannot be employed on public buildings in Germany, since the government provides in its contracts that native slate must be used.

STONE.

THE stone-quarrying industry is one of the most important branches of mineral production in the United States. Its magnitude is indicated by the classification of the uses of stone. Most important is the stone used for building, comprising sandstone, granite, marble, and limestone. Next in importance is undoubtedly the stone used for paving, comprising cut granite and trap blocks for city streets and curbings, flagstones, and various kinds of rock, chiefly limestone and trap, which are broken for the construction of macadam and telford roads, for concrete, etc. With the general movement in favor of improved roads in the United States the amount of stone consumed for the last purpose is increasing immensely. Next in importance to the stone for building and for paving is the limestone used for the production of lime, an industry which exists in nearly every State of the Union. The amount of limestone used as flux in iron, lead, and copper smelting is very large, as is also the amount of granite and marble used for monumental purposes.

Owing to the manner in which the stone-quarrying industry is carried out it is difficult to compile statistics which fully cover the production. The more part, it is true, is represented by the figures of large concerns and combinations which are able to make reports of statistical value, but there is a large part supplied by contractors who keep no systematic accounts, and by farmers and others not regularly engaged in the industry, who keep no accounts whatever, so that at the best a large element of uncertainty is involved in any statement of the total production. The production of slate, for which there are full returns, this industry being carried on entirely by well-organized firms and companies, is given under a separate caption.

STATISTICS OF THE STONE INDUSTRY IN THE UNITED STATES—PRODUCTION AND IMPORTS.

Year.	Bluestone.	Granite.	Limestone.	Marble.	Onyx Marble.	Sandstone.	Totals. (b)	Imports.
1891.....	a$1,500,000	$13,867,000	$15,792,000	$3,615,000	(c)	$8,700,000	$43,474,000	$1,312,856
1892.....	a1,600,000	12,627,000	18,392,000	3,705,000	$40,000	8,265,500	44,629,500	1,525,271
1893.....	a1,000,000	8,808,934	13,947,223	2,414,592	28,750	5,195,151	31,394,650	1,637,165
1894.....	a900,000	10,029,156	16,190,118	3,201,585	29,000	3,945,847	34,295,706	1,652,390
1895.....	a750,000	8,894,328	15,308,755	2,888,114	12,000	4,211,314	32,064,511	1,276,365
1896.....	a700,000	8,000,000	16,000,000	2,600,429	24,000	5,000,000	32,324,429	1,196,009

(a) Estimated. (b) This does not include the vast amount of trap-rock, etc., used for macadam, and probably omits a large part of the stone of various kinds used for rubble masonry, etc. (c) No statistics.

PRODUCTION OF MARBLE IN THE UNITED STATES.

Year.	California.	Georgia.	Maryland.	New York.	Tennessee.	Vermont.	Other States	Totals.
1891.....	$100,000	$275,000	$100,000	$390,000	$400,000	$2,200,000	$150,000	$3,615,000
1892.....	115,000	280,000	105,000	380,000	350,000	2,275,000	200,000	3,705,000
1893.....	10,000	261,666	130,000	206,926	150,000	1,621,000	35,000	2,414,592
1894.....	13,420	724,385	175,000	501,585	231,796	1,500,399	55,000	3,201,585
1895.....	56,566	547,312	135,000	376,683	328,711	1,363,842	80,000	2,888,114
1896.....	32,415	490,144	97,000	400,952	321,757	1,173,161	85,000	2,600,429

The production of oölitic limestone in Indiana in 1895 was 4,790,422 cu. ft. valued at $1,005,192, or 21c. per cu. ft.; in 1896 it was 4,040,999 cu. ft., valued at $755,891, or 18c. per cu. ft.

PRODUCTION OF LIMESTONE FOR IRON-FLUX IN THE UNITED STATES.

Year.	Long Tons.	Value.	Year.	Long Tons.	Value.	Year.	Long Tons.	Value.
1891.....	5,000,000	$2,300,000	1893.....	3,750,000	$1,687,500	1895.....	4,723,154	$1,889,262
1892.....	5,016,227	2,307,464	1894.....	3,544,393	1,524,089	1896.....	4,311,563	1,724,625

The States producing the most granite are Massachusetts, Maine, Vermont, and Rhode Island, which rank in the order named. The four of them furnish nearly 60% of the total production of the country. The most important quarries in Massachusetts are at Rockport and Quincy; in Rhode Island at Westerly. In sandstone production Ohio holds first place, Pennsylvania second, New York third, Connecticut fourth, and Massachusetts fifth. The five produce nearly 75% of the total for the country. The value of the limestone quarried is greatest in Pennsylvania, which is followed by Illinois, Indiana, Ohio, and New York; in each of these States the product exceeds $1,000,000 per annum. The most important marble quarries are in Vermont, which supply about 50% of the product of the country.

One of the uses for stone which is increasing most rapidly and is very difficult to estimate statistically is paving, especially in suburban and rural districts, where the general movement toward road improvement is leading to the laying of vast amounts of macadam. *Municipal Engineering* gives valuable data as to the relative value of stone for this purpose.

Taking as a standard 1 cu. meter = 1.3 cu. yards of basalt with a crushing strength of 1500 kgms. per sq. centimeter (21,335 lbs. per sq. in.) the equivalent necessary quantities of other stones are as follows:

Material.	Crushing Strength.		Proportionate Quantity Required, Cu. M., or Cu. Yds.
	Kgms. Per Sq. Cm.	Lbs. Per Sq. In.	
Basalt	1,500	21,335	1.00
Basalt	1,400	19,912	1.08
Diorite, aphanitic greenstone, melaphyr or augite porphyry, gabbro	1,300	18,490	1.20
Granite, syenite, compact quartz porphyry	1,200	17,068	1.32
Porphyry of the granite group	1,100	15,645	1.59
Quartzy graywack, millstone grit	1,000	14,223	1.84
Graywack	900	12,801	2.15
Quartzy limestone, compact new red sandstone	800	11,378	2.57
Jura limestone	600	8,534	4.00
Jura sandstone	500	7,112	7.00
Chalky limestone	400	5,689	12.00

Certain of the States make statistical reports of stone production within their borders. A recent report of the Indiana Bureau of Statistics says, concerning the limestone industry, that returns were received from 112 quarries, covering 4944 acres of ground. The estimated value of grounds and machinery in 1895 was $3,056,320. There were quarried and shipped during the year 61,217 carloads (say 1,225,000 tons) of stone of the value of $1,958,376. The total of $940,648 was paid in wages to 3519 men and 175 boys. The average wages of men were $1.65 per day; of boys, 67c. The average number of days worked was 225.

In California, according to the State Mining Bureau, the production of various kinds of stone in 1895 was as follows: Marble, 14,864 cu. ft. ($56,566); onyx, 1200 cu. ft. ($12,000); paving blocks, 2,332,000 ($73,338); rubble, 414,038 tons, ($394,952); sandstone, 55,242 cu. ft. ($35,373); serpentine, 4000 superficial feet ($4000). In 1896, marble, 7889 cu. ft. ($32,500); sandstone, 58,000 cu. ft. ($28,300); granite, 185,000 cu. ft. ($278,000); lime. 20,275 bbls. ($152,000); onyx, 3000 cu. ft. ($24,000).

THE MARBLE QUARRIES OF CARRARA, ITALY.

The British vice-consul at Spezia, in a report on the Carrara marble industry, says that in 1895 the production of the quarries was 108,951 tons of ordinary and statuary marble, and 52,360 tons of sawed and worked marble.

The different kinds of marble in the market from the Massa Carrara quarries are statuary or Carrara, properly so called, Sicilian, veined, dove, and peacock. There are a few colored quarries, but their product is insignificant. Massa produces some colored marble. There is a quality of marble, perhaps the most rare, and for some purposes the most beautiful, known as "*pavanazzo*," or peacock. It has a creamy ground with blood violet or purple markings or veins. Of the Sicilian (*bianco chiaro*) blocks of almost any size can be obtained. It is only a question of transport. Blocks weighing as much as 40 tons have been seen at Carrara. A quarry of red marble has lately been worked near Garfagnana.

The main valleys in which the quarries lie are the Ravaccione and Fantiscritti. To reach the Ravaccione a long valley of quarries has to be passed, at one end of which, named Crestola, the finest statuary marble is excavated, while at the other end the commonest "Sicilian" is found. Two explanations are given for naming the ordinary *bianco chiaro* marble "Sicilian." One is that during the French occupation of Italy it was sent to Sicily and thence to England; the other that the vessels loading marble afterward went to Sicily to complete their cargoes with fruit, etc.

The number of quarries is estimated at 645, of which 387 are worked. Of these about 229 give Sicilian, 27 statuary, 22 veined, 7 dove, and 2 peacock marble. The quarries give work to 4500 men, and 1000 more men work in the towns, at the sawmills, studios, etc., as sawyers, carvers, rubbers, and polishers.

SULPHUR.

New discoveries of sulphur in the United States in 1896 were reported from Belle Isle, St. Mary's Parish, La., and in the Cascade Mountains, Washington, but as to their importance there is not yet reliable information. No production was made in 1896 by the Nevada or Texas deposits. Increased products were made by the Utah mines, and by the deposits of Calcasieu Parish, La., which are worked by the Union Sulphur Company. The Louisiana sulphur was won entirely by the Frasch process.

In the execution of the Frasch process* a well is sunk by drilling precisely in the same way as for petroleum. This is cased with an iron pipe 10 in. in diameter, which enters the rock overlying the sulphur bed a distance of 10 ft., and the joint being subsequently sealed as well as possible by molten sulphur, water is supposed to be excluded. Inside of the 10-in. pipe is placed one of 6-in. diameter; inside of the latter one of 3-in. diameter; and finally inside of the 3-in. pipe one of 1-in. diameter. The well itself is carried down to the bottom of the sulphur bed, and the 6-in. and smaller pipes are dropped nearly to the bottom of the hole. The inner pipes are recovered when the well is abandoned.

Water heated to 330° F. is forced down the 10-in. and 6-in. pipes under the corresponding pressure, and in coming in contact with the sulphur, the latter (which fuses at 239° F.) is melted and collects in the well as a thin liquid of about sp. gr. 2.

At first the molten sulphur was pumped out through the 3-in. pipe in the same manner that petroleum is raised from a well, the working valve being made of aluminum, which is unaffected by molten sulphur. It was found, however, that aluminum was not strong enough to withstand the shock of the heavy column at change of stroke, and a good deal of difficulty having been experienced on this account the attempt to pump the sulphur was finally abandoned. The 1-in. pipe was then introduced and compressed air forced down this. The bubbles of air mixed with the sulphur reduce the specific gravity of the column of sulphur and it rises with great rapidity.

Although about 2500 tons of sulphur had been produced by this method up to the end of 1896, and about 1000 tons in the first three months of 1897, it can hardly be said to be beyond the experimental stage. The operation in Louisiana has not been regular and the actual cost of production is consequently excessive and difficult to estimate. It is reported that as much as 150 tons of sulphur has been pumped in 24 hours. To work a single hole whether producing one ton or 100 requires about 50 tons of coal to be burned under the boilers, and the labor of a

* U. S. Patents Nos. 461,429, 461,430, and 461,431, each dated Oct. 20, 1891.

large number of workmen. The loss of heat is, of course, enormous. In order to estimate the cost of production correctly it is also necessary to charge off the cost of each well to the sulphur won from it, and to make allowance for depreciation of the plant and property. The yield of a well depends upon the purity or impurity of the bed where tapped, and consequently there is no means of estimating in advance whether a well will produce 100 or 1000 tons. One of those already exploited in a bed of exceptional purity and more than 100 ft. in thickness produced 2100 tons; another produced only 100 tons; so far three wells have been pumped from.

Precisely what occurs at the bottom of the well is, of course, a matter of surmise. We may assume, however, that the sulphur melts out of the bed to form a more or less pear-shaped cavity, and the larger this becomes the more cooling surface is exposed to the water, until finally the loss of heat reduces the temperature below the melting point of the sulphur; while the impurities of the bed collect on the floor and sides of the chamber covering the still unfused sulphur, and protecting it from the action of the water. If the bed is very pure the chamber might become quite large, when the unsupported overlying roof would fall and allow the water to escape through the overlying beds.

The Frasch process is certainly ingenious, and its inventor deserves much credit for the skill with which he has worked out its numerous details, but that it is a commercially applicable process under ordinary conditions is quite open to question. The sulphur that has been produced is practically refined mineral and finds a ready sale on account of its high purity and attractive appearance.

SULPHUR PRODUCTION, IMPORTS, AND CONSUMPTION IN THE UNITED STATES.

Year.	Production.			Imports.								Consumption.	
	Sulphur.			Crude.		Flowers of Sulphur.		Refined.		Totals.			
	Long Tons.	Value.	Value per Ton	Long Tons.	Value.	Long Tons.	Value	Long Tons.	Value	Long Tons.	Value.	Long Tons.	Value.
1891....	1,071	$39,600	$37.00	116,971	$2,675,192	206.00	$6,782	10.00	$1,997	117,187	$2,683,971	118,208	$2,722,026
1892....	1,630	54,750	33.00	100,938	2,189,481	158.00	5,439	26.00	4,106	101,122	2,199,026	102,752	2,253,773
1893....	1,200	26,880	22.50	107,601	1,903,191	240.96	5,746	41.73	1,017	107,885	1,909,954	109,020	1,935,474
1894....	441	7,056	16.00	124,467	1,734,643	165.00	4,145	41.00	1,207	124,673	1,739,641	125,114	1,747,405
1895....	1,650	26,400	16.00	125,950	1,593,148	581.00	12,888	229.00	4,378	126,760	1,610,414	128,410	1,636,814
1896....	3,800	94,200	24.80	145,318	2,085,076	665.00	13,266	447.00	8,226	146,430	2,106,568	148,230	.....

The value of the sulphur produced in Utah in 1896 was calculated as $30.00 per ton; the production of Louisiana was calculated at the average price of seconds in New York, viz., $19.00 per ton.

PRICE OF BRIMSTONE PER LONG TON IN NEW YORK IN 1896.

Month.	Seconds.	Thirds.	Month.	Seconds.	Thirds.	Month.	Seconds.	Thirds.
January.....	$15.37	$15.00	May.....	$16.10	$15.66	September....	$23.12	$22 12
February.....	15.00	14.50	June.....	17.84	17.02	October.....	23.55	22.00
March.....	15.09	14.59	July.....	18.75	18.16	November.....	22.65	21.87
April.....	15.72	15.47	August.....	21.65	21.10	December.....	23.06	22.14

Average for year: Seconds, $19.00; thirds, $18.30.

New Hebrides.—It is reported that a new sulphur deposit of considerable importance is being developed on the island of Tanna, New Hebrides.

Mexico.—Preparations are being made to work the sulphur deposits of Popocatepetl, but no production has yet been made from them. It is proposed to bring the sulphur down from the crater by means of an aerial tramway which has already been surveyed.

Sicily.—The continued decline in the value of sulphur, which began in 1891, brought the market at the beginning of 1896 to a condition wherein there was little or no profit to the Sicilian producers. Wages were reduced so that the miners earned only 7 to 10c. per day, yet it was still complained that there was no profit in producing sulphur, and various plans for bringing about an improvement were discussed, among which were the formation of a combination to limit production, the establishment of magazines for the deposit of mineral on which bankers would make advances, and petitions to the government to remove the export tax of $2.12 per ton. Finally, on November, 1895, the producers resolved to form a "*consorzio*," or general union of all the mines of Sicily, to be operated under the administration of a council, which was empowered with the regulation of the output of each mine, the transport of sulphur to the seaports, and the erection of storage warehouses there, but this scheme did not prove successful.

Early in 1896, however, the concern known as the Societá Anglo-Siciliana, with a capital of £1,000,000, in which English, French, and Italian capitalists were interested, was organized to control the Sicilian sulphur industry. This company made agreements with the miners whereby it was to be the sole purchaser of their product, and had the right to dictate a decrease of production to the extent of 18% in case the annual output should exceed the demand. On the other hand the company guaranteed the miners a price of $15.94 per ton for best unmixed seconds f. o. b. ships for a period of five years, and in case of abolition or reduction of the export tax the miners were to receive the benefit. Furthermore the company guaranteed in case of reduction of output to pay the miner 1% extra for each 3% of reduction. By an agreement among the undertakers of the company, among whom was Mr. A. M. Chance, the production of sulphur by the Chance-Claus process was to be limited to 40,000 tons per annum.

The organization of this syndicate was successfully carried out, and as a result of its control of the market the price of sulphur rose steadily to the end of the year. In May seconds sold in New York at $15.50@$15.75 per ton; in October sales were made at $26. The fluctuations of the local brimstone market are outlined in the review of the New York Chemical Market in the first part of this volume.

The larger part of the Sicilian sulphur is exported from Licata and Girgenti, and comparatively little from Catania and Palermo. Owing to the primitive furnaces still in use at the mines the fusing of the sulphur (except at the modern works at Catania) cannot be done except after harvesting of the crops, which would be ruined by the escaping fumes.

The occurrence of sulphur in Sicily, methods of mining, and treatment of the crude mineral have been ably described by Giovanni Aichino in a monograph, "Lo Zolfo," published in *Enciclopedia delle Arti e Industrie*, 1897.

SHIPMENTS OF SICILY SULPHUR FROM SICILY TO THE UNITED STATES. (IN LONG TONS.)

Port.	1891.		1892.		1893.		1894.		1895.		1896.	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.
New York	29,358	19,665	34,390	14,700	29,146	14,250	33,150	13,725	35,888	19,975	50,557	17,796
Charleston	17,196	4,450	4,010	500	11,665	1,860	3,273	12,023	700	8,450	2,330	5,370
Philadelphia	450	6,406	3,600	6,800	1,900	6,260	350	5,050	1,200	7,150	500	5,500
Baltimore	4,510	6,855	900	11,455	2,050	7,900	600	14,700	1,100	8,620	3,650	10,500
Boston	1,300	650	1,825	1,500	500	.....	1,017	3,300	2,350	2,600	4,600	700
Savannah	850	700	600	570	3,450	1,880	5,695	4,100	3,784	800	8,370	1,025
Wilmington, N. C.	1,900	700	.....	.....	.....	1,140	.....	1,890	.....	650	1,260	1,400
New Orleans	.....	.....	.....	.....	1,900	.....	2,400	.....	1,700	.....	2,100	.....
Other ports (*a*)	1,200	1,330	4,000	.....	.....	.....	800	3,700	1,880	2,380	7,975	1,290
Totals	56,764	40,756	49,325	35,525	50,611	33,290	47,285	58,488	48,602	50,625	81,342	43,581

(*a*) Norfolk, Mobile, and Portland.

TOTAL EXPORTS OF SULPHUR FROM SICILY SINCE 1890. (*a*) (IN LONG TONS.)

Country.	1890.	1891.	1892.	1893.	1894.	1895.	1896.
United States	106,656	97,520	84,450	83,901	105,773	99,227	124,923
France	71,790	56,168	73,176	89,736	56,932	69,696	76,739
Italy	40,231	42,212	38,711	54,486	49,895	49,349	54,009
United Kingdom	26,213	23,408	24,853	27,453	22,165	24,043	21,913
Greece and Turkey	18,103	11,414	*b*14,845	*b*13,840	*b*16,870	*b*16,195	*b*18,556
Portugal	16,695	11,439	13,490	14,545	8,670	14,562	12,001
Russia	17,158	11,930	14,178	19,730	17,977	17,962	18,752
Germany	15,703	10,629	14,326	16,259	16,437	15,472	15,680
Austria	8,746	10,575	9,096	10,169	11,494	12,170	13,799
Turkey	4,231	3,000	(*b*)	(*b*)	(*b*)	(*b*)	(*b*)
Spain	5,679	3,845	7,382	3,499	3,445	5,753	5,910
Belgium	7,279	5,089	5,133	4,358	5,644	6,410	7,527
Holland	.....	.....	2,183	2,957	2,365	3,335	3,834
Sweden	3,314	2,252	4,561	6,579	7,887	5,730	14,540
South America	.....	.....	.....	.....	.....	.....	.....
Australia	.....	.....	1,200	.....	.....	.....	.....
Denmark	400	300	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)
Other countries	2,565	3,542	3,152	1,680	3,376	7,732	8,562
Totals	344,763	293,323	309,536	349,192	328,930	347,636	396,745
Stock in Sicily at the end of the year	106,770	117,037	175,299	210,665	198,513	203,756	222,999

(*a*) From report of A. S. Malcomson, New York. (*b*) Exports to Greece and Turkey combined after 1892. (*c*) Included in exports to Sweden.

In methods of separating sulphur from its gangue little progress has been made. The "*calcerone*" is still the most generally employed, though melting out the sulphur by aid of solutions of calcium chloride is used to some extent.* The proposition to extract sulphur by means of carbon disulphide has not yet come to anything.

In a recent paper in the *Rassegna Mineraria*, R. G. Aichino says that by far the more part of the Sicilian sulphur is extracted from the accompanying gangue by liquation in kilns, the necessary heat being produced by the combustion of part of the sulphur. In other cases coal or wood is used as fuel, or the sulphur is melted by superheated steam, or a solution of calcium chloride, of which the boiling point is slightly higher than the melting point of sulphur. The general method is still that of liquation in kilns, which were originally introduced in 1860. The only difference in those of the present day is the increased size. The kiln consists merely of a circular pit, lined by a wall. The bed slopes

* Vincent, *Bull. Soc. Chim.*, XL., 528.

at an angle of 10° to 15° toward a rectangular aperture 4 ft. high and 9 in. wide. The burning lasts, in kilns holding 250 to 2500 cu. meters, from one to three months. The consumption of sulphur as fuel amounts to one-third or even more of that contained in the rock, a proportion much larger than is theoretically necessary. In practice it is not found expedient to treat rock containing less than 10% sulphur. The cost of treatment averages 18c. per ton. In order to reduce the great loss of sulphur numerous modifications have been proposed. In 1880 Robert Gill patented for this purpose a regenerative furnace which has been largely adopted. The chambers have a capacity of 10 to 30 cu. meters, and a kiln with two chambers can be used for 45 to 90 operations per annum. The steam extractor, which was introduced 25 years ago, has gradually come into general use. It consists of an iron vessel filled with mineral into which steam is conducted. The cylinder holds 5 to 5½ tons, and the operation lasts 24 hours. In a recent installation with a horizontal cylinder holding 4½ tons it was found possible to work 15 to 16 charges in 24 hours.

In 1891 about 74.5% of the sulphur produced was from the old type of kilns, 17% from regenerative furnaces, and 8.5% from steam extractors. In 1895 the proportions were 56.3%, 31% and 12.7% respectively.

Chance-Claus Sulphur.—Of the various processes for the recovery of sulphur from the tank waste of the Le Blanc soda works only the Chance-Claus has proved practicable on a large scale. Probably about 12,000 to 15,000 tons of 2240 lbs. are now recovered annually by this process in the United Kingdom, a large part of it being "flowers of sulphur," which finds a ready sale because of its purity and attractive appearance. In the operation of the Chance-Claus process from 10 to 15% of the sulphur still escapes in gaseous form and in flour, and no improvements have yet been made to overcome this loss.

WORLD'S PRODUCTION OF SULPHUR. (*a*) (IN METRIC TONS.)

Year.	Austria	France. (*c*)	Hungary	Germany	Greece.	Italy. (*b*)	Japan.	Russia.	Spain.	Sweden	United States.
1891.....	45	6,749	40	2,020	1,498	395,528	21,963	336	9,900	23	1,088
1892.....	53	7,231	42	2,155	1,525	418,555	20,521	403	7,822	46	1,656
1893.....	44	3,793	70	2,161	2,400	417,671	23,930	589	4,686	75	1,219
1894.....	76	851	93	2,168	1,946	405,781	.....	1.5	3,417	36	448
1895.....	*Nil.*	.....	102	2,061	1,480	385,197	.....	.....	.....	.....	1,676

(*a*) From the official reports of the respective governments; this table does not include sulphur made by the Chance-Claus process in the United Kingdom. (*b*) Crude. (*c*) Raw mineral; limestone impregnated with sulphur.

TALC AND SOAPSTONE.

THE production of both fibrous talc and massive talc or soapstone increased in the United States in 1896, though owing to the fall in prices the value of the product was less than in 1895. Under the class "fibrous talc" is included the fibrous or foliated variety of the mineral which is mined only at Gouverneur, St. Lawrence County, N. Y. All other varieties of the mineral are classified as "massive talc, or soapstone." The fibrous talc is used chiefly in the form of powder for paper-filling, but a few tons are consumed annually in paint-making and for wall-plaster. The massive talc is either sawed into slabs and used for laundry-tubs, hearthstones, mantels, refractory lining for stoves, etc.; or ground to powder it is used as a lubricator, or in paper-making, and in dressing skins, leather, etc. About 40% of the product is used in powder form. As recently as 1889 a large amount of talc was imported into the United States (from France and Italy), the importation in 1888 having been 24,165 short tons, and in 1889, 19,229 tons, but in 1890 it fell off to 1044 tons, and since then has been very irregular.

STATISTICS OF FIBROUS TALC AND SOAPSTONE IN THE UNITED STATES.

(In tons of 2000 lbs.)

Year.	Production.						Imports.		
	Fibrous Talc.			Massive Talc or Soapstone.					
	Tons.	Value.		Tons.	Value.		Tons.	Value.	Per Ton.
		Totals.	Per Ton.		Totals.	Per Ton.			
1892.....	41,925	$472,435	$11.27	23,208	$423,449	$18.00	531	$5,546	$10.44
1893.....	36,500	337,625	9.25	20,100	366,825	13.50	1,360	12,825	9.43
1894.....	50,500	454,500	9.00	21,044	401,892	19.09	622	6,815	10.95
1895.....	40,000	320,000	8.00	22,133	342,298	15.40	3,165	26,843	8.48
1896.....	45,000	315,000	7.00	24,319	265,806	10.93	1,950	18,693	9.60

Fibrous Talc.—The occurrence, production, methods of preparation and uses of this variety have been fully described in previous volumes of THE MINERAL INDUSTRY, especially in Vol. II. The genesis of these deposits has been described recently by C. H. Smyth, Jr., who says:* "The crystalline limestones of St. Lawrence County are frequently tremolitic, and pass into tremolite and enstatite schist. The talc is clearly due to the alteration of these anhydrous

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silicates. Its secondary origin is proved in most cases by its pseudomorphic character, retaining perfectly the form and structure of the original minerals, and showing every gradation in the process of alteration. The so-called veins of talc are, therefore, simply altered beds of tremolite and enstatite schist. The chemistry of the conversion of enstatite and tremolite into talc is expressed by the following equations:

$$4\,MgSiO_3\,(\text{enstatite}) + H_2O + CO_2 = H_2Mg_3Si_4O_{12}\,(\text{talc}) + MgCO_3;$$
$$CaMg_3Si_4O_{12}\,(\text{tremolite}) + H_2O + CO_2 = H_2Mg_3Si_4O_{12}\,(\text{talc}) + CaCO_3.$$

"The substitution of hydrogen for calcium in the latter reaction is quite in harmony with the generally accepted view that talc is an acid metasilicate, $Mg_3H_2(SiO_3)_4$. The essential agents of alteration are evidently provided by circulating water holding CO_2 in solution, and many occurrences of talc pseudomorphs show that the change may occur under ordinary conditions of pressure and temperature. Considering its origin, the composition of the talc is surprisingly close to that demanded by theory."

Soapstone.—The principal sources of massive talc, or soapstone, in the United States are Pennsylvania, North Carolina, and Georgia. Concerning the Southern deposits Mr. W. M. Brewer contributes the following information: "The southern talc found in the Eastern market has been so far chiefly from North Carolina; while the production of Georgia has found a market nearer home, and the deposits in Alabama have not yet been producers on a commercial scale.

"The talc deposits of Georgia which have been developed are situated in the Cohutta Mountains in Murray County. These have been mined for the past 15 years on a limited scale, and the crude material prepared for shipment in pulverized form as well as sawed into pencils and bricks at Spring Place, the county seat, where a plant was erected several years ago. The disadvantage of being 15 miles distant from the nearest railroad station (Dalton, on the Southern, and Nashville, Chattanooga & St. Louis railroads) has restricted the product. The fact that the mines are situated about 7 miles from the plant has also been detrimental to the industry.

"In the New York market the North Carolina product has been handled for a long time, shipments of both pulverized talc and pencils having been made. In the form of powder it has not been quite satisfactory, because of carelessness in the preparation. The consequence has been that to-day instead of the market being entirely supplied with the domestic product the dealers have to rely largely on the mineral imported from Italy and France.

"But little effort has been made in prospecting for talc in either Georgia or Alabama. This may be accounted for by the fact that the geological formation in which this mineral occurs is situated in the mountainous districts remote from railroads, and the expense of wagon-freighting added to the cost of mining and preparation for market leaves but little if any profit to the producer.

"The present market price of Italian, French, and New York powdered talc varies from 1¼c. to 1¾c. per lb. in New York, and the price of pencils made from North Carolina talc is 4½c. per lb., the size preferred being 5 in. long, 1¼ in. wide, and $\frac{3}{16}$ in. thick.

"In Tallapoosa County, Ala., extensive beds of soapstone occur, which apparently are derived from the alteration of chrysolite, and the same conditions are noticeable near Delta P. O. in Clay County. The only use the Alabama product has found so far was in the walls of the old copper-smelting furnaces in Randolph County, Ala., and for hearths, chimneys, and blacksmith forges.

Talc in France.—At Luzenac, in the upper valley of the Arriège, talc is quarried on an extensive scale in the granite of Saint Barthélemy, a mountain 7700 ft. high, 55 miles south of Toulouse, and 20 miles from the main chain of the Pyrenees. The quarries, situated at an elevation of 5900 ft., are opened in a bedded deposit, 160 to 1000 ft. thick, between mica schist (below) and Lower Silurian slates (above), which dips east and has been followed north and south 6000 ft. The mineral has composition A in the following table, B being that of talc from the valley of Pignerolles in Italy, which has a high reputation for quality.

Kind.	SiO_2.	MgO.	Al_2O_3.	Fe_2O_3.	CaO.	H_2O.	Alkali and Loss.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
A—French	61.85	34.52	2.61	0.25	Trace.	0.60	0.17
B—Italian	60.60	35.30	0.30	0.60	0.40	Undet.	2.80

The best rock is of a brilliant white color, and feels greasy to the touch when ground to a white powder. The principal quarry, at Tremouin, is worked across the direction of the bed, the surface covering 6 to 10 ft. thick having been first stripped off. The broken mineral is hauled in wagons 12 miles to Luzenac, where it is ground by ball-mills. The principal applications of the product, which is used in all parts of Europe and America, are in soap-making and perfumery, paper-making and weaving; it is also used as an ingredient in wagon-grease; and a mixture, called cupro-steatite, made of talc and sulphate of copper, is successful as an insecticide in vineyards. The lump talc is used as an insulator for electric conductors, and for the manufacture of pencils for marking on metal.

TELLURIUM.

THE extraction of the rare element tellurium is a rather significant factor in the treatment of the local ores at the Royal Hungarian Smelting Works at Schemnitz. Dr. D. A. Louis describes the process employed there as follows:

About 300 lbs. of ore containing 3% tellurium is mixed in a cast-iron pot with about 700 lbs. sulphuric acid of a strength of 66° Be. and boiled for six hours. The next day the mixture is transferred to a lead-lined tank and treated with about 65 gals. of hot water, and about 40 lbs. of commercial hydrochloric acid is mixed in during the next six hours. The following day the mass is filter-pressed and the solution containing tellurium chloride is passed into a lead-lined wooden tank, clamped up air-tight, where it is digested with sulphurous acid for two or three days, or until no precipitate is obtained on testing with sodium sulphite. The reactions that takes place are as follows:

$$TeCl_2 + SO_2 + 2\,H_2O = Te + 2\,HCl + H_2SO_4.$$

$$TeCl_2 + Na_2SO_3 + H_2O = Te + 2\,HCl + Na_2SO_4.$$

The Royal Hungarian Smelting Works at Schemnitz produces tellurium on a large scale and puts it on the market in the crude and refined state and in various forms at the following prices, per kilogram of actual tellurium, cash at Schemnitz: Crude tellurium, up to 30% Te, 60 florins; 30 to 60% Te, 70 florins; 60 to 80% Te, 90 florins; 80 to 95% Te, 100 florins. Fused tellurium (94 to 97% Te) is worth 200 florins per kgm., and can be had in sticks or granulated. Special turned tellurium sticks of different lengths, diameters and weight, suitable for scientific collections, can be obtained for 22 kreuzers per gm.

Gold-silver telluride ores occur in the United States in large quantity at Boulder and Cripple Creek, Colo., whence many specimens rich in tellurium may be obtained. Tellurous ores are also found in considerable quantity in the Black Hills of South Dakota and in West Australia.

In Japan tellurium is found in native sulphur, which is a massive, crystalline, red mineral, a variety of the seleno-sulphur found in the Lipari Islands, Naples, and in the Hawaiian Islands. It is semi-transparent, and is indistinguishable in appearance from simple sulphur, except by its beautiful orange color, and occurs interspersed with simple sulphur. An analysis of this telluro-sulphur by Divers and Shimidzu gave: Tellurium, 0.17%; selenium, 0.06%; arsenic, 0.01%; molybdenum and earthy matter, traces; sulphur, by difference, 99.75%.

TIN.

THERE was no tin produced in the United States in 1896; indeed, there was very little done in prospecting for the metal. The Harney Peak Company of South Dakota did some systematic work in this direction, with rather promising results, it is said, but being still involved in litigation the company is unable to undertake active exploitation of its property. The Temescal (California) mines and those of Virginia remained idle. New discoveries of tin ore were reported in Lemhi County, Idaho, and Garfield County, Utah, but presumably they were of little consequence.

In the tin-producing countries 1896 was generally an unfavorable year. Malacca, Banca, and Billiton made a slight increase in production; Bolivia, which has been coming to the front rapidly in recent years, fell off a little; while in Australia, Tasmania, and Cornwall there was a considerable decrease. The reason for this is to be found without doubt in the low price ruling for the metal, and it is noteworthy that those countries which suffered no diminution of output are those which have the cheapest labor.

South Africa is now a small producer of tin ore, a discovery having been made in Swazieland in 1893. In 1893 there were exported 30 tons of ore, in 1894, 141 tons, and in 1895, 246 tons; no figures have yet been reported for 1896. A small amount of tin is produced in Mexico, but only enough to meet domestic requirements, which are small, and there are no statistics of the output. The city of Durango is supplied by the mines in the Cacária district, and Guanajuato, Lagos, Leon, and San Luis Potosi from the mountains in the northwestern part of Guanajuato, and probably other cities are supplied in similar manner, tin ore in small quantity being found in many places in Mexico. These deposits were fully described by Walter Renton Ingalls in *Transactions American Institute of Mining Engineers*, Vol. XXV., 1895. Such tin as is produced in Mexico is smelted by natives in small shaft furnaces, and no exploitation is now being carried on by foreign companies.

In Tasmania no tin mining is done outside of the Mount Bischoff deposit, which is one of the most remarkable in the world. Up to June 30, 1896, the Mount Bischoff Company had produced 48,500 tons of tin and paid £1,419,000 in dividends. The ore is dressed in a mill equipped with 75 stamps, spitzlütten and spitzkasten, jigs and round tables. The average grade of the ore at present is 3 to 3.5%. The cost of mining and transporting the ore to the

works at Waratah is 78c. per ton of 2240 lbs. The mine is worked as a quarry, and the dressing is done for 23c., including sacking of concentrates. The concentrates average about 70% tin, yielding an average of 68% in smelting.

In Cornwall, the famous old Dolcoath mine, having abandoned the old cost-book system and incorporated, made its first report, the half-year ending June 30, 1896, showing a profit of £2931, although the 1030 tons of black tin sold brought only £37 3s. 10d. per ton, which was £2 less than the average of the previous half-year. The mine is now equipped with a 60-stamp California mill, and work is in progress to sink a new shaft 3000 ft. deep.

The attempt to reorganize tin mining in Cornwall on a modern basis, which was begun successfully by the Dolcoath Company, has been checked, however, in the case of two or three others by the action of the "lords" or owners of the realty. They have either demanded excessive royalties, or have imposed conditions which could not be carried out at a reasonable cost, nor without disturbing the plans for improving the mines.

The exports of tin from Bolivia to England, which represents the greater part of the production of that country, are given in the following table, in long tons:

Year.	Bars.	Ore.	Tin in Ore	Total.	Year.	Bars.	Ore.	Tin in Ore	Total.
	Tons.	Tons.	Tons.	Tons.		Tons.	Tons.	Tons.	Tons.
1888.....	440	1,420	923	1,363	1893.....	1,362	2,380	1,547	2,909
1889.....	590	1,229	999	1,389	1894.....	1,305	3,350	2,199	3,482
1890.....	565	1,687	1,096	1,664	1895.....	1,612	4,824	2,485	4,097
1891.....	663	1,379	896	1,559	1896.....	1,240	3,903	2,589	3,829
1892.....	1,076	2,681	2,743	2,819					

In addition to the above some tin ore is shipped from Bolivia to Havre and Hamburg, the metal obtained from which appears in the French, German, and Austrian statistics.

The methods of mining, dressing, and smelting in Bolivia have not yet been described in print, except incidentally; nor have the ore deposits, which geologically and mineralogically are very unique in character.*

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (*a*)

	1891.	1892.	1893.	1894.	1895.	1896.
English production	9,353	9,270	8,837	8,328	6,648	5,000
Straits' shipments to Europe and America	31,457	34,648	39,670	45,640	47,840	47,180
Australian shipments to Europe and America	5,991	5,972	5,579	5,824	4,992	4,320
Banca sales in Holland	5,350	6,300	5,244	5,626	6,221	6,735
Sales of Singkep in Holland	*Nil.*	*Nil.*	*Nil.*	261	644	839
Billiton sales in Java and Holland	5,753	5,560	5,462	4,735	4,539	5,040
Bolivian arrivals in England	1,559	2,819	2,909	3,482	4,097	3,829
Straits shipments to India and China	4,085	3,450	4,285	4,655	4,674	6,118
Totals in long tons	63,548	68,019	71,986	78,551	79,655	79,061
Totals in metric tons	64,565	69,107	73,138	79,808	80,930	80,326

(*a*) This table is based on the statistics compiled by W. T. Sargant & Co., but the figures of English production are taken from the British Blue-Book, except for 1896, where Sargant's have been used. The Straits shipments to India and China are as reported by Messrs. Boustead & Co. This table does not include the production of Germany, Austria, Spain, Portugal, and various other countries.

* The only literature on the tin deposits of Bolivia is an article by John B. Minchin, *Engineering and Mining Journal*, April 18, 1891; and a few paragraphs in THE MINERAL INDUSTRY, Vol. I., and the *Mineral Resources of the United States* for 1894.

PRODUCTION OF TIN IN THE WORLD. (*r*)

Year.	Australia. (*a*)			Austria. (*b*)			Banca and Billiton. (*c*)	Bolivia. (*d*)	England. (*e*)			Germany. (*f*)		
	Met. Tons.	Value. (*s*)	Per M Ton.	Met. Tons.	Value. (*s*)	Per M Ton.	Metric Tons.	Metric Tons.	Met. Tons.	Value. (*s*)	Per M Ton.	Met. Tons.	Value. (*s*)	Per M Ton.
1891..	2,988	$1,308,845	$438	56	$30,585	$546	11,281	1,584	9,503	$4,405,695	$464	287	$131,131	$457
1892..	3,305	1,507,705	456	72	40,888	568	12,050	2,864	9,419	4,473,765	475	684	309,973	453
1893..	2,679	1,115,695	416	66	36,824	558	10,877	2,956	9,124	3,928,705	431	951	348,135	365
1894..	2,653	897,225	338	80	38,812	485	10,527	3,538	18,461	3,022,500	357	896	271,804	303
1895..	2,235	680,400	304	60	24,257	404	10,932	4,166	6,755	2,233,900	331	884	266,292	301

Year.	India (*g*)	Japan. (*h*)	Mexico. (*j*)	Portugal. (*k*)	Russia. (*l*)			Singkep. (*m*)	Straits Settlements. (*o*)	Tasmania. (*p*)			United States. (*q*)			Total Met. Tons.
	Met. Tons	Met. Tons	Met. Tons.	Met. Tons.	Met. Tons.	Value (*s*)	Per M Ton.	Met. Tons.	Met. Tons.	Met. Tons.	Value. (*s*)	Per M Ton.	Met. Tons.	Value. (*s*)	Per M Ton.	
1891..	56	45	23	*Nil.*	9	$3,440	$382	28	36,111	3,329	$1,465,850	$440	57	$25,085	$440	65,357
1892..	70	41	*Nil.*	18	9	3,520	391	86	38,708	3,246	1,453,970	448	65	29,827	459	70,637
1893..	66	39	5	53	7	2,501	357	170	44,658	3,179	1,301,095	409	*Nil.*	.....	.....	74,830
1894..	100	.....	60	26	4	1,440	360	265	51,100	3,195	1,348,000	422	*Nil.*	.....	.....	80,905
1895..	22	.....	30	.....	.....	.....	.....	654	53,354	4,265	1,301,350	305	*Nil.*	.....	.....	83,357

(*a*) From *Report of Secretary of Mines and Agriculture of New South Wales*, which is the only Australian colony that produces metallic tin. Tin ore is also exported from New South Wales. Tin ore is produced in Victoria, Queensland, South Australia, and Western Australia, but its metal contents are reported in the productions of other countries.

(*b*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.* The tin production of Austria is derived partly from domestic ores, and partly from Bolivian ores and impure tin from the East imported for refining.

(*c*) Total sales in Holland and Java as reported by W. T. Sargant & Co.

(*d*) Exports of tin and tin in ore from Bolivia to England. Some of this tin may be produced in Peru. It is all exported through Peruvian and Chilean ports.

(*e*) From *Mineral Statistics of the United Kingdom.* These figures give the amount of tin estimated to be obtainable in smelting the ore of domestic production. They differ considerably from the figures used by W. T. Sargant & Co., which are possibly obtained directly from the smelters.

(*f*) From *Vierteljahrs und Monatshefte zur Statistik des Deutschen Reichs.* By far the larger part of this production is from Bolivian ores.

(*g*) The figures for 1891–93 are by M. H. Warth, from the *Review of Mineral Production in India.* Those for 1894–95 are from official returns by the Department of Revenue and Agriculture, Calcutta.

(*h*) From the *Mining Industry of Japan.*

(*j*) According to Mr. Walter Renton Ingalls. Small amounts produced by natives for domestic consumption are not included.

(*k*) From official returns furnished by the Portuguese Government to the *General Report Upon the Mineral Industry of the United Kingdom*, 1895.

(*l*) From Russian official reports.

(*m*) The figures for 1891–93 are for fiscal years, as reported by O. H. Van der Wyck in *Mineral Resources of the United States* for 1895; the figures for subsequent years represent sales in Holland, reported by De Mouchy & Havelaar.

(*o*) Shipments from the Straits to Europe and America, as reported by W. T. Sargant & Co., and to India and China by Boustead & Co. There are no official statistics fully covering the production of tin in the Malay Peninsula, though there are returns for the most important States—Negri Sembilan (including Sungei Ujong and Jelebu), Pahang, Perak, and Selangor—which are under British protection. According to the *Second General Report of the Mineral Industry in the United Kingdom* (by C. Le Neve Foster) the production of tin in these States in 1894 was 46,631 metric tons, and in 1895, 49,688 metric tons. Not included in these figures is the production of Malacca and Siam. The shipments from the ports of Pahang and Singapore may be regarded, however, as representing the entire production of the Malay Peninsula, except Burma (which is reported under India). The production of Siam in 1894 was 5040 tons.

(*p*) From *Report of the Secretary of Mines.*

(*q*) From reports by the sole producer to THE MINERAL INDUSTRY.

(*r*) It will be observed that this table differs from the statistics usually referred to for the world's production of tin, namely, those of W. T. Sargant & Co., and of the *Metallgesellschaft Frankfurt am Main.* This is because (1) they reckon Australian shipments to Europe and America instead of the Australian production, whereby a good deal of Australian tin consumed at home is not included; (2) shipments from the Straits to India and China and the production of several minor countries are not included. A small amount of tin produced in Spain is necessarily omitted, owing to the absence of statistics for that country.

(*s*) In cases where the statistics in the above table have been taken from official sources, wherein values of product have been stated, a calculation of unit values has been made. This developed some amazing differences. For example, no one will believe that Austrian tin (chiefly from Bolivian ores, which are not exceptionally pure) was worth $103 per ton more than the tin produced from the same kind of ore in Germany. These discrepancies are undoubtedly due to making value-calculations at different stages of the product; *i. e.*, more or less cost of carriage is included. Statisticians should agree upon a uniform system for this class of work.

CONSUMPTION AND STOCKS OF TIN IN ENGLAND, AMERICA, AND HOLLAND. (a)

(In long tons.)

	1892.	1893.	1894.	1895.	1896.
Consumption:					
Deliveries from London after deducting all shipments to America	14,122	18,663	18,050	17,222	19,015
Deliveries from Holland after deducting exports to London and America	8,789	7,853	8,207	9,029	10,150
English consumed at home	2,772	1,473	2,314	988	5,000
Exports of English, minus quantity shipped to America	5,648	6,554	5,686	5,530	
American consumption of all sorts	18,750	19,000	16,650	22,500	21,500
Billiton sent to other ports than Holland	2,605	1,919	862	1,532	1,603
Straits direct to Continent, less reëxports to America and England	5,500	8,900	11,725	7,622	11,400
Bolivian delivered from Liverpool	2,656	2,704	3,323	4,099	4,071
Totals	60,842	67,066	66,817	68,522	72,739
Stocks, Dec. 31:					
Stock of foreign in London	2,776	4,392	8,985	13,539	18,097
Foreign landing in London	1,346	817	1,189	1,025	1,174
Straits afloat for London, including wire advices	2,170	3,470	3,778	2,825	2,792
Australian afloat for London, including wire advices	957	909	650	825	525
Banca on warrants in Holland	868	319	1,006	1,154	1,616
Billiton in Holland	326	640	1,632	1,507	1,638
Billiton afloat for Holland	1,240	1,600	1,129	1,097	1,742
Straits stock in Holland	190	670	767	277	789
Bolivian in Liverpool	220	335	494	492	250
Total stocks	10,093	13,152	19,630	22,741	28,623
Estimated stock in America and quantity floating	5,492	2,730	3,820	6,823	2,925
Grand totals	15,585	15,882	23,450	29,564	31,548
Trading Company's reserves of unsold Banca stock in Holland	3,480	4,200	5,770	6,140	5,953
Floating for Holland	139	.....	.....	.....	.....

(a) From the annual metal circular of W. T. Sargant & Sons.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1891	41,146,123	$8,091,363	1893	40,184,556	$8,007,202	1895	54,252,045	$7,405,619
1892	46,821,958	9,415,889	1894	39,268,628	5,944,065	1896	44,639,324	5,848,933

Tin Plate.—The production of tin plate in the United States increased immensely in 1896, and the domestic make is now in excess of the imports. The following table gives the statistics of the industry in metric tons:

Production and Imports.	1892. (a)	1893. (a)	1894. (a)	1895. (a)	1896. (a)
Tin plate made	2,059	20,859	37,018	54,582	96,430
Terne plate made	4,131	24,529	26,123	33,327	43,835
Total make	6,190	45,388	63,141	87,909	140,265
Made from imported black plate	1,973	25,500	24,156	15,070	1,927
Imports (b)	272,793	257,564	218,521	223,115	121,085

(a) Years ending June 30. (b) Including both tin and terne-plate; these figures are for calendar years.

The manufacture of tin plate has been thoroughly mastered in the American mills, their product withstanding comparison as to quality with any of foreign make. It is especially noteworthy that the increase in home production, meaning a corresponding substitution for the imported, was made in the face of declining prices and unfavorable business conditions, consumption having been very disappointing. Another noteworthy event in 1896 was the sale of American steel in Wales for rolling to black sheet.

Italy, which has been heretofore a large consumer of English tin plate, is now making progress in the industry, and within a few years will doubtless make enough for domestic requirements. There are now three mills in operation, the largest being at Piombino, which turn out about 80 tons per week. The importations into Italy from England are falling off steadily.

In England the tin-plate industry suffered severely in 1896. The exports declined seriously in the early months, the average for the first quarter being only 63,000 tons, as compared with 97,000 tons for the last quarter of 1895. The slack demand caused many of the mills in South Wales to be shut down, so that in the first fortnight of April there were no fewer than 253 mills idle; 91 were working at 10% reduction; 40 at 12½%, and 91 at 15%. Nearly 70 other mills were working at reductions ranging between 5 and 6%. The output at that time was estimated to be only a third of the production when works are fairly employed. At the large works of Cwmfelin about 400 men were thrown idle, owing to their refusal to accept a reduction of 25%.

The *Economist* in July spoke of the tin-plate trade of South Wales as having been in a state of utter collapse during the first six months of 1896, and attributed the fact to the growth of the tin-plate industry in America. Shipments to the United States, which a few years ago averaged at least 25,000 tons per month, in July amounted to less than 10,000 tons. In that month only about one-half of the works in South Wales were working, and these were making little or no profit.

With a view to resuscitating the trade, two plans were discussed—one a reduction of wages, and the other the adoption of a vigorous policy in opening new markets. With respect to the first it was thought that a reduction of 25% in wages would enable the recovery of the American market. In a few cases the men agreed to this, but the mills which had still to pay the old rate were thereby put at a disadvantage, and later in the year many of them were obliged to close down. Several meetings were held during the summer to frame a scheme for developing new markets, and more especially in order to secure a firmer footing in India, China, and Japan. But the mill owners did not agree upon a definite policy, and, although an association was established for the purpose of carrying out a programme of vigorous enterprise in these countries, it had hardly been called into existence when it was dissolved. Various suggestions have, moreover, been made during the year, with a view to stimulate the local demand. One of these was the more general adoption of tin plates for roofing. In discussing the subject a Welsh newspaper brought to notice the fact that the tin roof is entirely unknown in England.

EXPORTS OF TIN PLATES FROM THE UNITED KINGDOM.

(In tons of 2240 lbs.)

	1891.*	1892.	1893.	1894.	1895.	1896
To the United States	325,145	278,479	255,583	226,880	222,901	113,051
To other countries	123,587	117,101	123,650	127,048	143,081	153,904
Totals	448,732	395,580	379,233	353,928	365,982	266,955

Tin in South Africa.

Since 1893 South Africa has appeared in the statistics as a tin-producing country, but hitherto there has been no description of the occurrence of the ore. This has been supplied by Mr. T. Flower-Ellis, in a paper read before the Chemical and Metallurgical Society of South Africa, February 20, 1897, wherein he includes a communication from Mr. S. Ryan, of which the following is an abstract:

The possibility of finding tin occurred to me when I was chief of the Swazieland police in 1889. It was not, however, until November, 1891, that I was successful in discovering the deposits. Several concessions were then acquired, extending over twelve miles along the tin belt, and the whole floated with a small capital as "The Ryan Company (Swazieland), Limited."

The geology of the tin district is, briefly, as follows: Granite boss flanked by metamorphic rocks intersected by elvan dykes, diorite, and rhyolite. The zones of quartz and hornblende schist and steatite may be observed striking north-northwest by south-southeast and dipping slightly west, being apparently of a much older formation that the "Gold Belt" resting unconformably a few miles north. The following is the order in which intrusive rocks have occurred: Intrusive granite which has undoubtedly tilted the schist; elvan dykes striking north and south and dipping at an angle of 30° west; diorite dykes intersecting the formation in all directions; rhyolite cutting through the formation in a northeasterly direction.

The elvan dykes vary in width from a few inches to many feet and are composed of feldspar, quartz, mica and hornblende, in which crystals of cassiterite, garnet, monazite, and tourmaline are contained. Molybdenite and corundum are also found in the schist through which the elvan dykes have cut.

The "stream tin" in alluvial deposits originates from the erosion of the elvan dykes before mentioned, and is deposited in three "beds" as follows: 1, Oldest deposit yielding ¼% tin; 2, yielding from ½ to 1% tin; 3, yielding ¼% tin.

The average thickness of these beds is 4½ ft., though the Lower Embabaan and Usutu Rivers, now being prospected, contain beds of greater thickness. 10% of the tin ore shipped contains 60% Sn, and the remainder assays 72%.

Work was started during the summer of 1894 in the face of great drawbacks, such as the lack of experienced managers, etc., and the difficulties usually attendant on opening a new industry in a new country without transport roads, etc.

At first great difficulty was experienced in obtaining a sufficient number of Swazie laborers, but latterly labor has been plentiful. The Embabaan tin fields are bounded on the west by the Little Usutu River, east by the Umvulusi, and the Babaan River runs through the center; these rivers are fed by a large number of creeks, and the supply of water is therefore plentiful. From January, 1894, to June, 1896, the amount of ore exported was about 600 tons. The ore is sent to London.

The Embabaan tin fields are situated on the eastern slope of the Drakensberg Mountains, 15 miles from the Transvaal border and 25 miles south of Steynsdorp (Komatie gold fields). The chief mining camp is Embabaan, which is about 5000 ft. above sea level. The Swazieland coal fields are situated on the Lebombo Flats, 40 miles from Embabaan.

The New York Tin Market in 1896.

Values for tin were depressed throughout the entire year in consequence of the bad statistical position, which several times during the year became quite alarming. On January 1, 1896, the total visible supplies were 28,265 tons, or 3348 tons more than in the previous year, while on April 1 the excess over the previous year was 6238 tons. Then there was a falling off, which was due mainly to a great deal of tin having been accumulated in the East, where it was held back for speculative account, so that on July 1 the excess had gone down to 2430 tons, but during the next three months shipments increased, and on October 31 the total visible supplies amounted to 35,516 tons, or 4725 tons more than in 1895, and on December 31 the visible supplies were 33,000 tons as against 27,000 tons in 1895, or 6000 tons more.

Deliveries in the United States rather increased, thanks to the sound and healthy development in the tin industry of the country. Not many years ago, it was still considered the monopoly of Wales to produce tin and terne plates cheaper than in any other part of the world, but the energy of our home manufacturers has since proved the fallacy of this theory. And now that this comparatively new industry has taken root in this country it has come to stay, and there is no possibility—protection or no protection—that the foreigners will again take possession of the markets as in former times; and there is no reason why this should not be so, considering that raw material can be had here just as advantageously as abroad, and the slight extra cost in labor is offset by the saving of the freight and boxing.

The consumption on the part of tin-plate manufacturers had a marked influence on our deliveries of tin, and while during the great depression of the summer imports and deliveries would no doubt have shown a not inconsiderable decrease, there was actually an increase during every month of the year.

The stability of prices has been of inestimable benefit to consumers, who, as far as it can be ascertained, continue the policy now followed for several years of buying only for their immediate wants, which they did chiefly for two reasons: First, the general commercial depression during the spring and summer; and, second, the unfavorable statistical position.

At present there are no signs of a large speculation being carried on in tin. The English syndicate, which disturbed prices in '94 and '95, has long realized on its holdings and gone out of existence, with a result not at all gratifying to those interested. The heavy stocks and still large production, coupled with rather dear money in Europe, are not conducive to a repetition of the operation, at least for the moment, but in the event of one or more of these factors changing, a repetition of former operations is not at all unlikely.

Early in January, 1896, prices suffered considerably, in consequence of the failure of one of the largest tin operators on the New York Metal Exchange, and a great many transactions were made at 13c. and a little below.

The better feeling which was noticeable in February did not fail to influence tin, and very soon the market went up to 13½c., but toward the end of the month, with the East a free seller, these prices gave way somewhat, and continued fairly steady during March and April.

In May, with good deliveries, a good advance set in, and for a time prices

ruled very firm between 13½c. and 13¾c., which continued up to the end of July, but from August on prices showed decided tendency to decline, and by the middle of that month not more than 13⅜@13½c. could be realized, which was soon followed by a larger drop. Early in September 13¼c. was the outside figure, and by the end of the month 13c. was reached.

From the beginning of October up to the end of the year, the market remained very steady between 12⅞c. and 13c., one of the main features having been that during the entire six months, from July to December, spot tin remained exceedingly scarce, and during most of the time commanded a premium of ⅛ to ¼c., not at all brought about by speculative operations, but simply on account of the dearth of the metal. The bulk of the heavy shipments which arrived during these months was transhipped from steamer's dock direct to the trade, and only small quantities were put into warehouse to be quickly reshipped before further supplies came in. The closing quotations were 12⅞@13c. for spot and January delivery and 13@13⅛c. for February to April.

The table given below shows the monthly average prices of tin in New York for the last five years.

PRICES OF STRAITS TIN IN NEW YORK, IN CENTS PER POUND.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1892.....	20.50	20.00	20.25	20.50	20.80	22.00	21.00	20.50	20.35	20.50	20.80	20.00	20.60
1893.....	19.99	20.30	20.71	20.81	19.96	19.76	19.15	18.81	20.14	20.84	20.61	20.67	20.15
1894.....	20.16	19.60	19.09	19.75	20.21	19.75	19.22	19.22	16.27	15.35	14.56	13.81	18.08
1895.....	13.25	13.35	13.20	14.00	14.65	14.15	14.40	14.35	14.45	14.65	14.40	13.91	14.05
1896.....	13.02	13.44	13.30	13.34	13.54	13.59	13.63	13.19	13.15	12.94	13.09	12.96	13.29

The speculative dealings on the New York Metal Exchange, which in former years at times assumed very large proportions, have almost entirely died out, not at all to the detriment of the trade.

The London Tin Market in 1896.

This market was, on the whole, steadier than the previous year's, the total range of values for 1896 having been only about £6 per ton, whereas the 1895 values varied from £58 12s. 6d. to £68 17s. 6d., or over £10 per ton. During the year under review, however, the general trend of values proved to be toward a decidedly lower level than in the preceding year.

At the outset the market was affected by the perturbations in the political world, which here, as in copper, tended to repress speculative activity, while the failure of a New York operator assisted the consequent decline. From £60 17s. 6d. at the close of 1895, Straits tin fell during the first week in January to £57 17s. 6d.; but this depression soon gave place to a reaction when the political sea grew smoother, and although there was not much speculative demand, there was, on the other hand, a pretty fair inquiry from consumers, chiefly, however, for alloys, as the tin-plate trade continued in a very unsatisfactory state. Silver was an element of further strength, as it remained steady with an upward tendency, and, aided by these factors, Straits tin rallied to £59, a large government inquiry having at this juncture enhanced the value.

Sellers coming out at these higher values, a relapse occurred to £58 10s. cash; but this in its turn was followed by a recovery to £60 7s. 6d., and so the values continued to oscillate between £58 10s. and £60 7s. 6d.

February was for the most part a colorless month, with no prominent speculative movement either way, and with values ruling steady at £61 5s. cash to £60, the close being at about 15s. per ton above the worst. The American market being dull, our market received no support worth naming from that quarter, but features which tended to steady the tone were the decrease of about 500 tons in the visible supply for the month and the decrease in the quantities of English tin smelted.

The first week in March was quiet at £60 15s. to £60 5s. cash Straits, with Australian at a premium of about 12s. 6d. per ton, and these prices to 5s. lower constituted the range also during the remainder of the month. Consumers continued to buy moderate quantities, but the speculative element seemed to be almost non-existent, there being still considerable uneasiness felt concerning political affairs and the condition of trade and finance in the United States, from which quarter the tin market again received very little support. During the month of March the visible supply in and afloat to Europe, increased to the extent of about 1300 tons, and the market, after opening a shade firmer at the beginning of April, with business at up to £60 13s. 9d., began to fall back steadily, until at the middle of the month it touched £59, the decline of silver from 31$\frac{1}{16}$d. to 30$\frac{13}{16}$d. having assisted the downward movement.

The remainder of the month passed off quietly, with values ranging from the lowest point named up to £59 17s. 6d. cash Straits, while the premium for Australian advanced to about 20s. per ton. The demand for consumption had been fairly good throughout the month, but in the absence of active speculation, and with the large shipments from the Straits, resulting in a fresh increase in visible supplies, while it was well known that large stocks were accumulating at Penang, the comparative weakness of the market is easily understood.

May brought with it no new features of importance, stocks and shipments continuing heavy in comparison with purchases, while speculative interest appeared almost completely dormant. London stocks, in particular, had largely increased, though their being in firm hands obviated any pressing danger to values from that source, while against that fact there was the consideration that America, having so long abstained from active buying, had let her stocks run very low. Values during this month ruled very steadily on the whole, the entire range being comprised between £59 8s. 9d. and £60 12s. 6d. cash Straits.

During the first fortnight of June the value of Straits tin remained steady at £59 17s. 6d. to £60 12s. 6d. cash. In the third week there was a sudden spurt, caused by a certain amount of speculative activity, induced, perhaps, by the stronger copper market, and a rise to £61 15s. took place, the month closing, however, about £1 per ton lower again.

During July the tendency of the article was, in the main, toward easier rates, and the total decline for the month was about £1 per ton.

The closing values for July were steadily maintained during the earlier part of August, at least approximately, cash Straits being dealt in at £60 to £60 2s. 6d., and three months at £60 10s. to £60 13s. 9d., but toward the close of the

second week the liquidation of considerable quantities of spot tin brought about a fall of a few shillings. Silver, too, which had long remained fairly firm at over 31d. per oz., now declined to 30⅞d., though there was a slight rally before the close. A small demand from America for early delivery and fairly good demand here for the same position, improved the value temporarily, but the month closed dull at £59 5s. cash and £59 15s. for three months.

Early in September the uneasiness felt in financial and political circles and the fear of a tight money market were reflected in the advance from 2% (at which it had stood for about two years) to first 2½% and then 3%, in the Bank of England rate of discount, and by this, together with the continued excessive output at mines, and a further fall in silver, the position of tin was greatly aggravated, so that free "bear" speculation was induced, resulting in the value running rapidly down to the record price alluded to at the opening of this report, the figure £56 being reached early in October for spot Straits, a figure so low that authorities consider it certain, if continued for any length of time, to arrest the over-production which had so long been weighing upon the market. Silver had also shared in the downward tendency, and after various fluctuations finally closed for October at 29 15/16d. However, the bottom price of tin, referred to above, was soon replaced by better values, the low level having tempted buyers, especially on the Continent, and their purchases raised prices, before October closed, to nearly £59 per ton.

At the beginning of November the recovery in silver to 30 3/16d. was attended by a further advance to £59 5s. for spot tin, but despite the result of the Presidential election in the States, the improvement was not maintained, a fresh relapse in silver inducing a drop of nearly £1 in the price of Straits. But we should here point out that even with the large shipments made during the year from the Straits, the general statistical position steadily improved.

Further, the shipments for the 12 months against the previous similar period show a decline of about 1000 tons, the condition of the tinplate trade in America has improved, and the Straits production has about attained its maximum, so that the improvement in the statistics will continue.

The month closed with Straits tin at £58 2s. 6d. cash and £59 three months, while Australian still commanded a premium of about 32s. 6d. per ton for good brands. English tin was held for £62 to £62 10s. for common ingots, £63 to £63 10s. for bars, and £64 to £64 10s. for refined ingots.

The following table shows the prices of Straits tin in London for five years.

AVERAGE MONTHLY PRICES OF STRAITS TIN IN LONDON, PER TON OF 2240 LBS.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
1892.....	89 19 4	89 3 1	89 8 1	91 17 6	95 19 4	100 13 5	97 2 6	94 5 0	92 10 9	94 7 6	94 1 3	92 6 3	93 9 6
1893.....	92 7 6	91 16 3	94 11 6	94 5 6	91 14 4	86 14 0	82 19 4	79 10 0	79 18 9	78 15 0	76 3 0	75 16 3	85 7 7
1894.....	73 2 6	71 2 6	64 15 0	68 7 6	70 10 0	71 5 0	70 1 3	65 7 6	71 7 6	71 2 6	67 2 6	62 13 9	68 18 1
1895.....	60 2 11	60 1 1	61 9 5	63 19 7	65 12 8	62 14 2	64 8 4	64 7 10	65 0 8	65 17 6	64 18 10	61 15 2	63 6 8
1896.....	59 6 8	60 13 4	60 8 3	59 13 10	60 0 6	60 14 6	60 17 3	59 16 1	58 10 10	57 7 5	58 11 8	58 0 1	59 10 0

Shipments from the Straits in 1896 were estimated at 47,000 long tons; from Australia 4500 tons; from Bolivia 3800 tons. The production of Banka tin was about 7600 tons; of Billiton, 5600 tons; of Singkep tin 850 tons.

The Metallurgy of Tin.

By Henry Louis.

Tin is a metal of a pure white to slightly bluish-white color. It is highly malleable, but somewhat less ductile, its tensile strength being low, this having been determined to be 4600 lbs. to the sq. in. Its sp. gr. is 7.29 when cast and 7.30 when rolled; that of ordinary commercial tin is about 7.5, most of the impurities usually present being heavier than tin itself. Its melting point is given variously between 227° and 233° C.; it forms a very mobile fluid when molten. At very low temperatures (about —40° C.) tin crumbles down into gray, friable grains, which are generally considered to be an allotropic modification. Tin boils at from 1600° to 1800° C. if heated out of contact with air; heated to this temperature in the air it takes fire, burning with a white flame; heated in the air at lower temperatures it readily gives off white fumes, but loses very little weight; heated for some time in the air it is gradually converted into a yellowish-white mass of stannic oxide, known as "putty powder." It is very little affected by exposure to air at ordinary temperatures, even in the presence of moisture.

Tin alloys readily with most other metals, many of its alloys being useful in the arts; its property of alloying with iron is utilized in the manufacture of tin-plate. The most usual impurities of tin are iron, arsenic, sulphur, antimony, bismuth, lead, copper, etc.; the effect of most of these is to diminish the ductility of tin and to render its color grayer and its luster duller.

The alloy of tin and iron is really the only one that affects the extraction of the former metal, or that needs notice here. As obtained in smelting operations, in the crucibles or forehearths, or on the beds of furnaces, when it is known as "hard-head," it forms a pale to dark-gray, irregularly granular or crystalline, brittle, more or less completely fused mass, and generally consists of more or less metallic tin mechanically intermingled with alloys of definite composition, that appear to be true chemical compounds. The commonest and best known is $FeSn_2$, which should contain: Fe 19%, Sn 81%. This has been carefully studied by Oudemans,* who has isolated it in crystals from Banka tin and finds its sp. gr. to be 7.743. This seems to be the alloy that generally forms when ferriferous tin is melted and allowed to cool slowly, the excess of tin being subsequently dissolved in hydrochloric acid. It appears to have been described and analyzed by Plattner and also by Nöllner in the year 1860.

Caron and Deville have described an alloy of the composition FeSn. Lampadius, Berzelius, Berthier and Stölzel describe alloys from hard-heads of Altenberg of the composition Fe_4Sn (Sn 30.5%, Fe 61.5%). Stölzel has obtained alloys of the formulas $FeSn_6$ and $FeSn_7$ in quadratic prisms from Schlaggenwald, and Lassaigne has described one of the composition Fe_3Sn. Stölzel says that whenever any alloy containing more than two-thirds its weight of tin is melted, tin liquates away, leaving the alloy Fe_4Sn behind.

Tin combines with oxygen to form two oxides, stannous oxide, SnO, and stannic oxide, SnO_2. The latter is practically the only compound of tin that has

**Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1890, I., p. 24: "Over verontreiniging van Banka Tin."

any metallurgical importance. When pure it is white to straw-yellow, practically infusible at furnace temperatures, and is not attacked by any ordinary acid, though it is soluble in solutions of the alkalies. It is reduced to the metallic state by heating to low redness in a current of hydrogen; on the other hand, metallic tin decomposes water vapor at a red heat, forming stannic oxide and hydrogen. It is also reduced by carbonic oxide, and at a low temperature by fusion with potassic cyanide. It is reducible to the metallic state by being heated with carbon, but the reaction is strongly endothermic and takes place only at a high temperature.

Heated with silica, stannic oxide forms silicate of tin. This compound has been most imperfectly studied; it appears to be a yellow substance with a somewhat resinous luster, readily fusible, reducible to the metallic state by carbon only in the presence of a base with which the silica may combine, but reducible by metallic iron. Stannic oxide acts in many respects more like an acid than a base and readily combines with other oxides to form stannates; stannates of the alkalies and alkaline earths are thus readily produced. They seem not to be reducible by charcoal except in the presence of some silicious matter that may combine with the alkaline or alkaline-earthy base.

The metallurgy of tin is in so far comparatively simple, in that the smelter has but one ore to deal with, namely stannic oxide or cassiterite, which is delivered to him in a state of comparative purity; this ore is in practice almost always reduced in the same way, namely, by contact with carbon or carbonaceous matter. The reactions involved are therefore of the simplest kind theoretically, but practically the process is complicated by the facts that:

(1) The temperature of reduction is so high that other metals, the oxides of which are invariably present in greater or smaller quantity, are reduced at the same time, and alloy with the reduced tin; this applies with especial force to iron.

(2) The tin ore must be contained in a furnace capable of resisting the high temperature required; hence the furnace lining must either be acid, consisting of silica or silicates, or basic, consisting of lime or magnesia. In the former case, which is the usual practice, a certain quantity of silicate of tin is produced; in the latter stannates would be formed. Since nearly all ores contain some silicious matter the former alternative is preferred.

There are therefore three stages that may be distinguished in the complete operation of tin-smelting: (*a*) Reduction or tin-smelting proper; (*b*) refining the impure tin; (*c*) cleaning the slags. Each of these stages may be performed in more than one way, and to their various combinations the different local modifications of tin-smelting are due.

Tin-smelting proper is conducted either in shaft furnaces or in reverberatories. The former method requires as an essential condition a supply of very pure fuel —such as wood charcoal—in sufficient quantity and at a reasonable price, and is best suited to ores in not too fine a state of division and of a high degree of purity. The latter method requires fuel capable of giving a hot flame, and can be applied to less pure ores and to ore that has been very finely crushed; it requires, however, a good supply of refractory material, and demands a higher degree of technical skill than the former process. Hence it is that smelting in

shaft furnaces was the original process, both on the Continent of Europe and in Cornwall, when charcoal and stream tin ores were still abundant, and it is still used very extensively throughout the far East, but has gradually given way in part on the Continent and in the East, and entirely in Great Britain, to reverberatory furnaces using coal as fuel, and capable of treating the finely crushed and more impure ores that necessarily result from lode mining.

The slags produced in either method contain tin both mechanically entangled and chemically combined as silicate. The former is separated either by a fusion that allows the molten tin to separate from the slag, or else by crushing and washing the slag. In both methods the greater specific gravity of the metal as compared with the slag is relied on for the purpose of separating it, but as tin is relatively a light metal, while the slags, which, as will be seen, consist largely of silicates of iron and of tin, are comparatively heavy, it becomes a very difficult matter to effect a good separation, and much of the metal remains with the slag and is only recovered in the subsequent operation, the main object of which is to reduce the combined tin. This is usually done by smelting the slags at high temperatures, either in shaft or reverberatory furnaces, sometimes with the addition of strong bases, such as lime, or of iron, or bodies (such as oxide of iron) that will yield iron in the furnace. Generally this operation has to be repeated more than once, and the tin produced is much less pure than that obtained in the smelting process proper; it is in many places spoken of as "slag tin." This is the process of slag treatment proper, but in most methods of conducting the smelting process the richer slags, together with any by-products rich in tin and free from injurious ingredients, are added to the smelting charge. Special methods of cleaning tin slags are also used, and will be subsequently described.

Tin may be refined by two different methods, both of which are employed in the case of impure metal; these methods are (1) "liquation," and (2) "boiling" or "tossing." In liquation advantage is taken of the low melting point of tin; impure tin is heated on the inclined bed of a furnace to a temperature but little above the melting point of tin; comparatively pure tin trickles down, and is received in a large basin or "float," in which it is kept in a molten state. The residue on the bed of the furnace consists of the difficultly fusible alloy of tin and iron, known as "hard-head," which generally contains sulphur, arsenic, copper, and other impurities. Liquation will obviously not remove readily fusible impurities such as lead and bismuth, and the tin is purified from these by boiling or tossing. The former operation consists in thrusting a billet of wood—apple-wood or cherry-wood being preferred—below the surface of the molten tin in the float; steam is evolved together with permanent gases produced by the destructive distillation of the wood, and their escape throws the tin into violent agitation, projecting portions that splash back into the float, so that a large surface of tin in the molten state is exposed to the oxidizing action of the atmosphere. In tossing, the same result is attained by taking out the molten tin by ladlefuls and pouring it back into the float from a height of two or three feet. By either method oxidation is promoted and the impurities in the tin, together with a certain quantity of the tin itself, are oxidized and form a pulverulent scum on the surface of the float, whence they can be skimmed off from time to time. The metal is allowed to stand for some hours before it is finally ladled out and cast into molds, so that

the impure metal may settle down to the bottom of the float, tin being, as already stated, specifically lighter than most of the impurities that are apt to impair its valuable qualities. This fact was already known to Lazarus Ercker, who wrote in 1672.

Tin is invariably poured by this indirect method, *i.e.*, out of a float, and not direct from the furnace, however pure it may be, because the temperature at which it is cast has a powerful influence upon the quality of the metal and needs careful watching and regulating. If cast at too high or too low a temperature, it is apt to be brittle and its color and luster impaired.

The various tin-bearing products obtained in the refining operations are added to the furnace charges in ore or slag-smelting, according to circumstances, so that as little as possible of the metal contained in these various substances may be lost.

SMELTING IN THE SHAFT FURNACE.

This furnace in its simplest form consists of a short vertical shaft, circular, square or rectangular in cross section, and of moderate height. The fuel used is charcoal, which is charged with ore in alternate layers; air is supplied by means of cold blast, commonly forced in through from one to three tuyères, which are usually not very high above the bottom of the crucible, and the reduced tin generally trickles continuously from a small tap-hole into a forehearth. One form of Chinese shaft furnace is, however, worked by natural draft. The exact nature of the chemical reactions is still doubtful. The ore may be reduced by the direct action of carbon in the zone of combustion, where the heat is ample, or partial reduction may take place higher up in the furnace, the reducing agents being possibly carbonic oxide or hydrogen generated by the reaction of the glowing charcoal upon moisture in the air blown in. It seems more probable to the writer that cyanide of potassium is the active reducing agent in the cooler zones of the furnace; the fact that this salt is abundantly formed in the blast furnace is amply proved,* and it is quite possible that one of the advantages possessed by charcoal over coke in this process is due to the large amount of alkali in the ash of the former, which would tend to increase the amount of cyanide present in the furnace. The slag produced consists chiefly of silicate of iron and other bases present, together with titanates or tungstates in many cases, and invariably a certain amount, and often a great deal, of silicate of tin. Since ores in a high state of purity are generally worked in this furnace the proportion of slag to metal is small; the production of silicate of tin is necessary to prevent most of the impurities that might deteriorate the quality of the metal from passing into the latter. As long as the slag contains a notable quantity of tin silicate the reduction of metallic iron is prevented, because any iron that might be reduced would, as already pointed out, reduce the silicate of tin with the formation of metallic tin and silicate of iron. Generally speaking the slags produced in the shaft furnace have to be subsequently retreated, whereby an impure tin is obtained from them, the tin produced in the smelting process proper being at times sufficiently pure to need no refining. While the principles here enunciated hold good in all cases, it

*John Percy: *Metallurgy of Iron and Steel*, 1864, p. 447.

becomes necessary to distinguish a considerable number of different processes, all of which are local variations of the above general method.

The Old Cornish Process.—Pryce* states that the reverberatory furnace was introduced into Cornwall for tin-smelting with fossil fuel in the commencement of the reign of Queen Anne (about 1705), and that up to that time the shaft furnace, known as the blowing-house, with charcoal as fuel, was always employed. He has, however, reason to believe that the original method of smelting in Roman or pre-Roman times was "to dig a hole in the ground, and throw the tin ore on a charcoal fire, which probably was excited by a bellows." Nothing is known as to the period when this primitive device was replaced by some form of the built-up shaft furnace, which maintained its place for so long a period. We shall, however, find ample evidence that such a primitive method, about which Pryce seems to have known nothing whatever, was the original method used in the far East.

The Cornish tin-miner or tinner took his cleaned stream tin to the blowing-house,† "paying the owner of the house twenty shillings for every tide or twelve hours, for which the blower was obliged to deliver to the Tinner, at the ensuing coinage, one hundred pounds gross weight of white Tin for every three feet or one hundred and eighty pounds of Stream Tin so blown; which is equal to fourteen pounds of Metal for twenty of Mineral, clear of all expense."

Pryce describes the smelting process as follows: "The furnace itself for blowing the Tin is called the Castle on account of its strength, being of massive stones cramped together with Iron to endure the united force of fire and air. This fire is made with charcoal excited by two large bellows, which are worked by a water-wheel, the same as at the Iron forges. They are about eight feet long and two and a half wide at the broadest part. The fireplace, or castle, is about six feet perpendicular, two feet wide in the top part each way, and about fourteen inches in the bottom, all made of moorstone (*i.e.*, granite) and clay, well cemented and cramped together. The pipe or nose of each bellows is fixed ten inches high from the bottom of the castle, in a large piece of wrought Iron, called the Hearth-Eye. The Tin and charcoal are laid in the castle, *stratum super stratum*, in such quantities as are thought proper; so that from eight to twelve hundredweight of Tin, by the consumption of eighteen to twenty-four sixty-gallon packs of charcoal, may be melted in a tide or twelve hours' time. Those bellows are not only useful for igniting the charcoal, but they throw a steady and powerful air into the castle; which, at the same time that it smelts the Tin, forces it out also through a hole at the bottom of the castle, about four inches high and one and one-half inches wide, into a moorstone trough six and one-half feet high and one foot wide, called the float; whence it is ladled into lesser troughs or molds, each of which contains about three hundred pounds of Metal, called Slabs, Blocks, or Pieces of Tin, in which size and form it is sold in every market in Europe; and on account of its superior quality is known by the name of Grain Tin, which brought a price formerly of seven shillings, that is further advanced, the last two or three years, to ten or twelve shillings per hundred more than Mine Tin is sold for, because it is smelted from a pure Mineral by a charcoal fire; whereas Mine Tin is usually corrupted with some portion of Mundick, and other

* Pryce: *Mineralogia Cornubiensis*, 1778, p. 282. † *Ibid.*, p. 136.

Minerals, and is always smelted with a bituminous fire, which communicates a harsh, sulphureous, injurious quality to the Metal."

The general appearance of these old castles may be gathered from Fig. 1, reproduced from Karsten's *System der Metallurgie*, published in 1832.

The above description has been quoted *in extenso*, because it not only conveys a good idea of the process as carried on in Cornwall little over a century ago, when the shaft furnace and the reverberatory were working side by side, but may also be taken as a typical account of the process as conducted in other parts of the world. It is noteworthy that the above quotation seems to compel the inference that "grain" tin is not so called, as is generally supposed, because the hot ingots readily break up into columnar grains, but because the metal was smelted from stream tin which occurs in grains, in contradistinction to that produced from ore from the mine. The stream tin must have been very pure to have yielded at least 70% metal; moreover Pryce makes no mention of any pro-

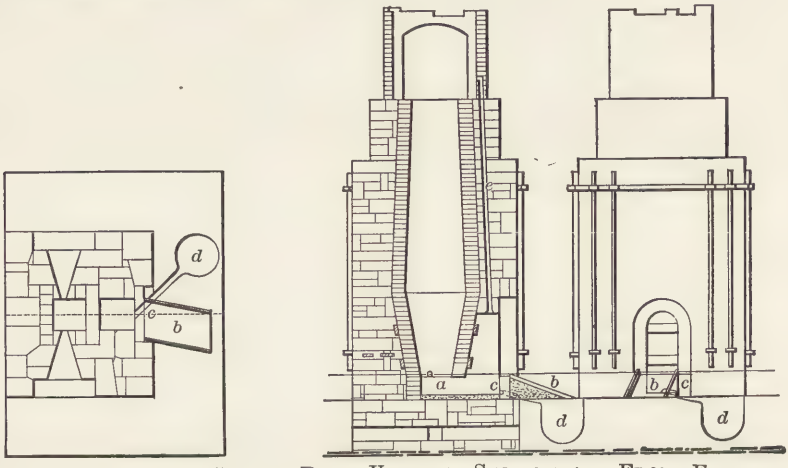


FIG. 1.—CORNISH TIN CASTLE; PLAN, VERTICAL SECTION, AND FRONT ELEVATION.

cess of refining, nor does he refer to any treatment of the slag, though he describes both in his account of smelting in the reverberatory furnace. It seems from the above figures that each shaft furnace was capable of producing about 15 cwt. of metallic tin per 24 hours with a consumption of an average of 315 bu. of charcoal, which, if of hard wood, as seems likely, may have weighed about 48 cwt., the cost to the tinner being about 27s. per ton. The value of tin at that time was about £60 per ton.

The German Process.—As appears from all that can be gathered from the older writers, such as Agricola and Ercker, the shaft furnace used in early times for smelting stream tin did not differ materially from that in use to-day for smelting the concentrates produced by dressing the ores obtained from the granitic district of the Erzgebirge, both on the Saxon and on the Bohemian sides of the mountain range.

The furnace in use to-day in Saxony* is shown in vertical section and in plan

*This description of Saxon tin smelting is the result of a comparison of the older German metallurgical writers, chiefly Kerl and Plattner, supplemented by information as to the practice of to-day from Mr. Albert Doerr, of Freiberg, to whom I am indebted for much useful information on the subject.—H. L.

in Figs. 2 and 3, while Figs. 4 and 5 (after Kerl) show what it was like at the beginning of the century, the changes being almost inappreciable. The furnace proper consists of three walls of granite *b*, protected by massive outer-walling of gneiss or granite *a*. The front wall *c* is made light so that it can easily be torn down and renewed when repairs are necessary. At *d* is the tuyère-arch, through which two tuyères are inserted. The furnace bottom *e* consists of a single block of granite. The tap-hole *f* is always left open, and through it tin and slag flow continuously into the fixed forehearth *g*, which is cut out in a mixture of charcoal breeze and clay held in place by blocks of stone and a plate, as shown in the drawings. In this forehearth the metal collects below a layer of slag. The latter is allowed to flow or is drawn over the plate *m* into the tank *n* filled with water, the object being to make it thereby more brittle, so as to enable it to be readily crushed. The tin is allowed to accumulate, and is tapped off at intervals of from 8 to 12 hours through the tap-hole into the float *h*. Above the furnace there is a

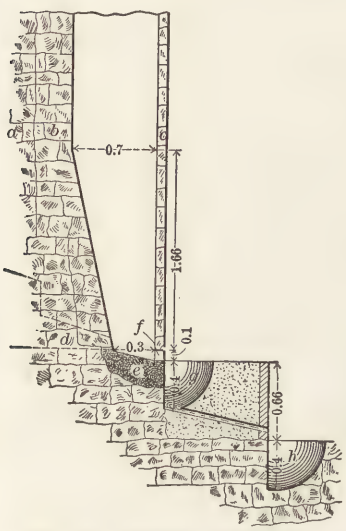


FIG. 2.—VERTICAL SECTION.

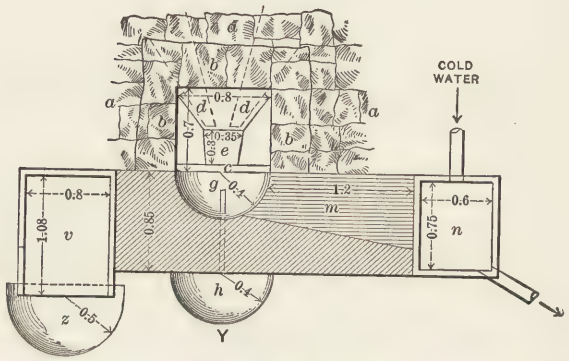


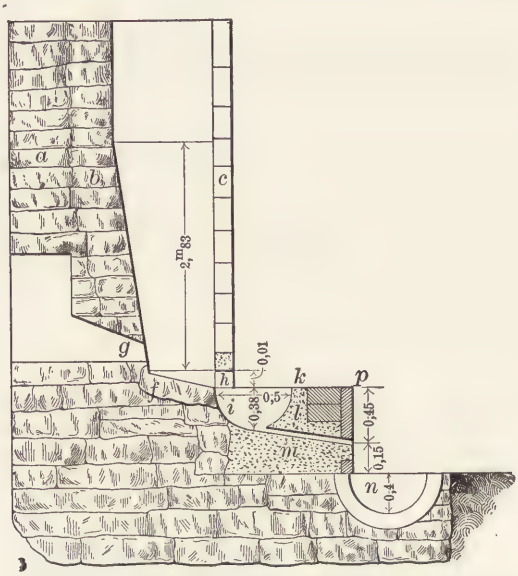
FIG. 3.—PLAN.

MODERN SAXON BLAST FURNACE.

hood, not shown in the drawings, communicating with a series of dust chambers in which any fumes escaping from the furnace, and more especially any fine ore carried up by the blast, are caught and collected.

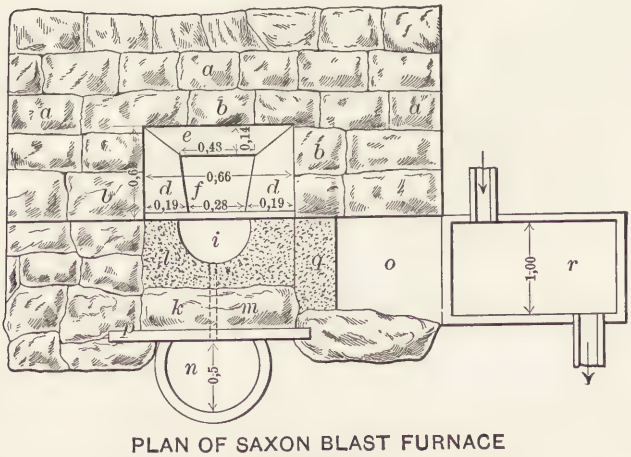
The furnace used at Altenberg is about 9 ft. 6 in. high. Blast is furnished by means of a pair of large bellows worked by a water-wheel. Each bellows is worked on an average of 12 strokes per minute, the total amount of air thus injected into the furnace being about 680 cu. ft. per minute. The blast is delivered through two sheet-iron tuyère-pipes of about $1\frac{3}{4}$ in. diameter made of light sheet iron; these two tuyère-pipes discharge into what may be called the tuyère proper, being simply an aperture in the rear wall of the furnace situated some 4 in. above the furnace bottom; the tuyère-pipes are not clayed in or made tight in any way, and are placed almost horizontal. The furnace and fore-hearth are well heated as a preliminary operation; the former is then filled

with charcoal, which is ignited and the blast then put on. Charges of about 100 lbs. each, containing about two-thirds concentrated ore and one-third slag, are put on every 15 or 20 minutes, enough charcoal being added between each charge to keep the furnace full to the top; the charcoal is always damped before being



VERTICAL SECTION SAXON BLAST FURNACE

FIG. 4.



PLAN OF SAXON BLAST FURNACE

FIG. 5.

charged, and water is sprinkled at intervals upon the charge in the furnace; the object is to keep the top cool and to prevent, as far as possible, the finer particles of ore from being carried off by the current of air. Care is taken to keep the temperature down; the tin, as it runs from the furnace, should be reddish in color, the fumes coming out of the tap-hole should be bluish, and the interior of

the furnace as seen through the tuyères should be yellowish. If the latter appears white hot, the temperature is too high and the blast must be reduced, while thick heavy fumes and a sluggish flow of tin and slag denote the opposite condition. As the tin cools down somewhat in the forehearth, a quantity of hard-head separates out; this hard-head used to be resmelted, but now appears to be sold to works at Tostedt, near Hamburg, where it is worked up with other ores.

The ores treated consist of concentrated lode stuff, containing 60 to 65% of metallic tin, and a furnace charge is composed of this ore together with 25 to 50% (usually about 30%) of slags from a previous operation and 6 to 7% of hard-head, skimmings, and other tin-bearing products. The average capacity of the furnace is 3200 lbs. of ore and up to 1600 lbs. of slags in 24 hours, but varies greatly with the quality of the ore as received from the dressing floors; at present the furnace yields 900 to 1200 lbs. of tin in 24 hours, the fuel consumption being about 1800 lbs. of charcoal per ton of tin produced. The loss of metal is very great; it is said by Lampadius to reach 15%, 8 to 9% being due to volatilization. Two men per shift are needed to each furnace. As now worked, a smelting campaign lasts only some 36 hours, and these are separated by intervals of about three weeks, so that the industry is by no means in a flourishing condition.

The products of the smelting operation are metal, slags, and hard-head which form sows either in the crucible of the furnace or in the forehearth as above mentioned. The following table shows the composition of two typical hard-heads from Altenberg, I. being an analysis by Berthier and II. by Plattner.

Number.	Sn.	Fe.	W.	Cu.	C.	Total.	Approx. Formula.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
I.....	32.22	64.14	1.64	.....	.....	98.00	Fe_4Sn.
II.....	80.89	17.16	.....	0.99	0.96	100.00	$FeSn_2$.

The slags come from the furnace in two conditions, rich and poor; the former contains unreduced ore and is returned to the furnace with the ore. The latter undergoes a special subsequent treatment. The following are analyses of such slags from Altenberg, by Lampadius and Berthier respectively:

Number.	SiO_2.	SnO_2.	WO_3.	FeO.	MnO.	CaO.	MgO.	Al_2O_3.	Total.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I.....	20.05	25.12	.....	30.15	.....	1.10	1.23	5.00	82.65
II.....	16.00	32.00	1.00	41.00	1.70	3.70	1.70	2.40	99.50

These slags also at times contain so much metallic tin in small prills that they are subjected to crushing and washing; or else they are resmelted directly with higher blast pressure in the ore-smelting furnace, which is only kept on ore for two or three days at a time at best. They are smelted together with other tin-bearing residues, the products being slag-tin and a slag that often contains a good deal of metal and has to be crushed and washed. The slag-tin thus produced is generally quite as pure as the tin from the ore furnace. The slags produced contain about 10 or 12% of SnO_2 and are resmelted generally in furnaces similar to the ore furnaces. but having two tap-holes and being only about 6 ft. high, to obviate

as far as possible the reduction of iron. The slags thus produced, which generally contain 5 to 7% SnO_2, are resmelted once or twice more, being finally thrown away when they contain some 3% SnO_2.

The tin that collects in the float of the furnace is at once refined without allowing it to cool. This operation is performed by ladling it out and letting it run slowly over an inclined corrugated cast-iron plate (*v*, Fig. 3) about 3 ft. 6 in. long, 2 ft. 6 in. wide and 1.5 in. thick, sloping about 1 in 7 to 1 in 10, upon which there is a layer some 6 in. deep of red-hot charcoal; the charcoal acts as a filter, retaining the less fusible impurities, while the purified tin trickles through the charcoal and collects under a layer of red-hot charcoal in a float (*z*, Fig. 3) at the lower end of the plate; the residual hard-head is freed from adhering tin as far as possible by beating it with wooden mallets, and is added to the charge in the first slag-smelting. The molten tin is allowed to cool down in the float until it has reached the right temperature for casting, when it is ladled out into molds or slabs, or poured on a copper plate, after which it is hammered up into rounded balls.

The following analyses will give an idea of the changes produced in the metal by these operations of refining:* *Unrefined tin:* Tin, 94.5 to 95.3%; copper, 2.5 to 3.6%; iron, 0.7 to 1.9%. *Liquation residues* (hard-head from liquating plate): Tin, 59.09%; iron, 9.24%; tungsten, 3.35%; copper, 11.80%; silica and slag, 8.47%; intermixed carbon and oxygen combined with the metals, 8.05%—total, 100. *Saxon rolled tin:* Tin, 99.76%; iron, 0.04%; arsenic, trace—total, 99.80%. *Saxon rod tin:* Tin, 99.93%; iron, 0.06%; arsenic, trace—total, 99.99%.

The production of tin by these methods has now shrunk to very small dimensions; in 1894† there were only two mines in Saxony producing tin ore, the total quantity of dressed tinstone being 84 tons; of this amount 82.35 tons were produced at Altenberg, which yielded on smelting 51 tons of tin and 3300 lbs. of hard-head. In 1895 the production is said to have been only 35 tons of tin and 2 tons of hard-head.

In Bohemia, at the chief center, Graupen and Schlaggenwald, the process is practically identical. The ores there are dressed, forming two grades, coarse ore containing about 70% of tin, and concentrates containing 45 to 48%. These are smelted with the addition of about 25% of slags from a previous operation and sometimes of imported (Bolivian) ores. About 2300 lbs. of ore are smelted in 24 hours with a consumption of 350 cu. ft. of charcoal; the average yield of tin from the above charge would be 1000 lbs.

The Bohemian smelting furnace is trapezoidal in section, and about 9 ft. high; the front and back walls are vertical, the side walls slightly flaring. The furnace is built of brick, the bottom being made of fine-grained sandstone; it has one tuyère. The furnace products are similar to those obtained in Saxony and their treatment is practically identical, the slags being resmelted several times, and crushed and washed if necessary; as soon as an ore-smelting campaign, which lasts from three to five days, is finished slag-smelting begins. The following is an analysis of Schlaggenwald tin-slag produced in the first smelting of the ore-slag, by Von Lill:‡ SiO_2, 24.06%; WO_3, 24.33%; SnO_2, 10.41%; FeO, 20.75%; MnO, 5.64%; Al_2O_3, 9.00%; CuO, 3.50%; MgO, 0.37%—total, 98.06%.

* *Berg- und Hüttenmännisches Jahrbuch der K. K. Bergakademie Wien.*, Vol. XIII., 1864: A. Löwe (1842).
† *Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen*, 1895, p. 47. ‡ *Ibid.*

The composition of some samples of Bohemian tin is shown by the following analyses:

Samples of Tin.	Tin.	Copper.	Iron.	Sulphur.	Arsenic.	Total.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Schlaggenwald unrefined tin.....	97.339	2.726	0.684	Trace.	Trace.	100.749
Schlaggenwald rolled tin.....	98.660	1.360	0.060	.....	Trace.	100.080
Schlaggenwald fine tin.....	99.594	0.406	.....	.....	.....	100.000

In the year 1894 there was only one mine and smelting works in operation at Graupen in Bohemia;* the native ores treated were only 24 tons, but 101.2 tons of tin ore and 16.4 tons of crude tin from Bolivia were also smelted, the total output of metallic tin being 80.2 tons. As far as can be gathered from the figures given, only about 3 tons of this output were the product of the native ores.

At Freiberg, in cupelling base bullion containing some tin, a certain amount of skimmings is produced in the first stage of the operation, which contain all the tin as oxide, together with litharge, oxides of arsenic and antimony, and some silver; this product is first desilverized and the residues containing about 12% of tin are smelted in small shaft furnaces; the metal so produced is calcined, producing antimonial lead and oxides of lead and tin still very impure. By a series of alternate reductions in shaft furnaces and calcinations of the product an alloy of tin, lead, and antimony, containing about 33%, ultimately results, which is sold in this condition; the quantity thus produced is quite unimportant.

At Pitkäranta,† in Finland, a little tin ore is mined, which is smelted in a manner similar to that used in Saxony. A variation of the German method is here in use. The ore is practically free from sulphurets, and is dressed in the usual way on shaking-tables and round buddles. Some of the copper ores raised here contain a certain amount of tinstone; such ore is first treated for copper in the wet way, and the residues are then submitted to concentration, the concentrates obtained being smelted with the others. Most of the tin ore is obtained from the No. 4 Omeljanov mine, and is of fairly high grade, the dressed ore containing 55 to 65% of tin. The furnace used is a cylindrical shaft furnace, 10 ft. in height and 18 in. in inside diameter; the fuel used is charcoal. The tin is run into bars, and these are refined by being placed upon a layer of red-hot charcoal contained in an inclined iron cylinder; the purest metal, which liquates through first, is cast into small ingots weighing about 25 lbs., of about the same shape as ordinary copper ingots. Another portion of the tin is cast into rods about 18 in. long. The following is an analysis of the refined metal: Sn, 99.74%; Cu, 0.08%; Fe, Pb, 0.18%—total, 100%.

The slag produced in the ore-smelting is very rich, as shown by the following analysis: Mechanically enclosed tin, 8.33%; Sn (as oxide), 56.19%; Fe (as oxide), 11.96%; silica and other constituents undetermined. These slags are resmelted together with other by-products; the slags thus produced are run out into conical molds and allowed to set, the tin collecting in the apex of the cone; this portion is broken off when cold and refined, the rest being looked upon as worthless.

* *Statistisches Jahrbuch des K. K. Ackerban Ministeriums für* 1894, Vol. II., Part I., p. 72.

† Private communication to the author from Dr. K. A. Moberg, of Helsingfors.

The tin production of Pitkäranta* is small and variable, as shown by the following table:

Product.	1891.	1892.	1893.	1894.	1895.
	Metric Tons.	Metric Tons.	Metric Tons.	Metric Tons.	Metric Tons.
Dressed tin ore	18.42	25.70	21.33	13.18	54.00
Tin	9.60	9.79	6.80	3.93	20.78

The total production from 1894 to 1895 inclusive was 150.4 tons of metal, or an average of 12.5 tons per annum.

Chinese Methods of Tin Smelting.—It seems probable that the Chinese method of smelting was first introduced into the Malay Archipelago, probably at Banka, and that from there it extended into the Malay Peninsula, displacing probably in both places cruder native methods. The original methods of the Chinese have subsequently undergone considerable modification, but it would seem that the change has been least in some of the more remote parts of the Malay Peninsula. It is stated by Van Diest† that the natives first worked tin in Borneo about the year 1700 and that the Chinese made their way into the island in the latter half of the eighteenth century. The natives of the island used to smelt their tin in the same way as they had been accustomed to smelt iron, for which metal the Malay Archipelago had long been famous. They dug a shallow pit about 1 ft. in diameter at the top; ignited charcoal was thrown in and then small quantities of charcoal and ore, and a high temperature was obtained by a blast which was conveyed to the bottom of the pit by a couple of bamboo pipes passing through the ground. The blast was produced by means of a piston working in a vertical hollow tree-stem, into the lower end of which the bamboo tubes opened. The natives do not seem to have attempted to treat their slags, which have been found in several places in the island, *e.g.*, near the village of Kepodarat, in the district of Toboali, where they were found to contain 15% of unreduced tin oxide, in addition to prills of tin.

This primitive process may be better understood from a description of it, as carried on in recent times in Siak, Sumatra,‡ which at one time produced a great deal of tin, though it is now no longer a producer. At Kota-rena the natives smelted the ore in a furnace made by digging a hole in the clayey ground to a depth of about 20 in. and with a diameter of 14 in. at the mouth. The blowing apparatus consisted of two vertical hollow logs of wood, about 6 ft. 6 in. high and 8 in. in diameter; in each was a piston, the two being worked alternately by one man, and from the bottom of each a bamboo tube conveyed the blast into the furnace. The smelting operation lasted 4 to 5 hours, three men being needed, namely, one smelter and two assistants, who relieved each other at the blower; the result of the campaign was from 24 to 26 lbs. of metallic tin. The molten metal collected in the bottom of the hole and was cast into molds made of pieces of split

* *Bidrag till Finlands Officiela Statistik*, XVIII., "Industrie Statistik."

† *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1872, I.; Introduction, p. 27; "Het Tin Smelten of Bangka," *ibid.*, p. 217.

‡ *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1874, I., p. 101: R. Everwijn, "Verslag van eene Onderzoekingereis in het Rijk van Siak."

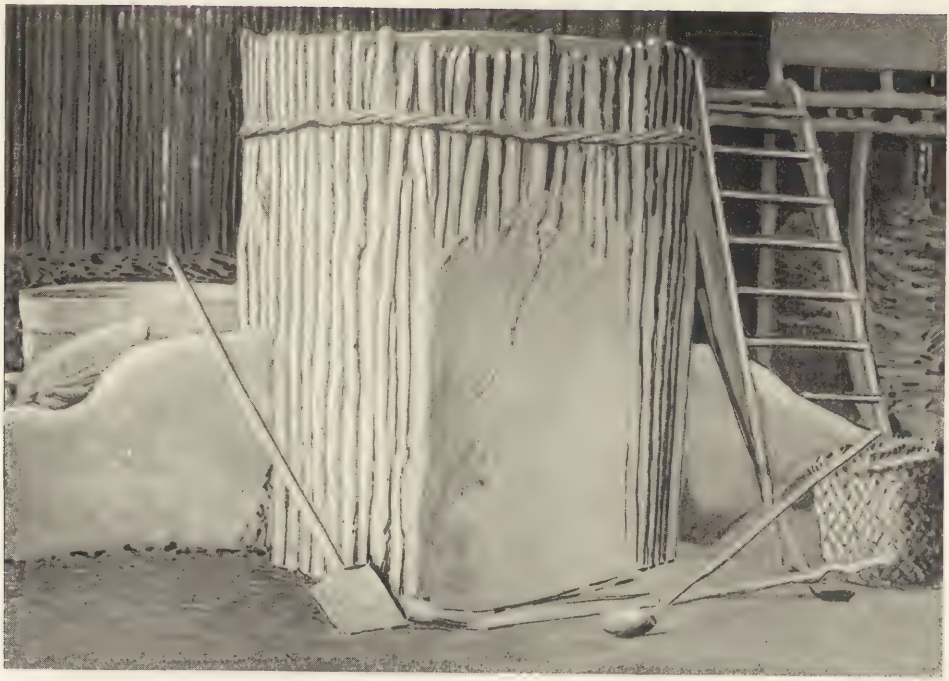


FIG. 7.—VIEW OF CHINESE TONGA FURNACE.



FIG. 13.—VIEW OF CHINESE FURNACE AT DREDA.

bamboo about 18 in. long, stopped with clay at both ends. The semi-cylindrical pieces of tin thus obtained weighed about 6 kati (8 lbs.). The yield of metal appeared to be about 60% of the ore. Most of the tin thus produced was for consumption in the country itself; it was said to cost more to produce in these furnaces than in the usual Chinese furnace.

It is interesting to compare the above description with that given by Dr. John Percy* of iron smelting in Borneo, and also to note the strong resemblance between this process and the method stated by Pryce to have been that probably used originally in Cornwall.

The original Malay smelting process, as practiced up to about 1870 in Perak, where it may possibly have been learned from the Siamese, seems to be a slight advance on the last; as far as can be gathered it seems to have developed on the same lines as the native methods of iron-smelting that are stated to have been in use in and about the Siamo-Malayan province of Singgora. The furnace seems to have been a low clay cylinder, and the blast to have been produced by a crude blowing machine made of bamboos standing on end. The tin was ladled out into molds making slabs about 2 kati ($2\frac{2}{3}$ lbs.) in weight.

All these primitive methods have been displaced by the Chinese method, which itself shows many minor variations.

Though an interruption to strict chronological sequence, it may be convenient to give next a description of the crude Chinese methods as still practiced in various parts of the Malay Peninsula, returning subsequently to the process as successively elaborated, and as now carried out in Banka and Billiton; it is really only in Banka, under the auspices of the Dutch government, that these processes have been at all exhaustively studied.

The furnaces used in the different districts of the Malay Peninsula vary somewhat in detail of construction, in dimensions, and in shape, depending upon the quality of the ore and the charcoal available, and apparently also on the fancy of the smelter. There, are, strictly speaking, two methods or two variations of the general process in use in the Malay Peninsula, the first using an air furnace (*i.e.*, a shaft furnace, worked by natural draft without blast), and the second a blast furnace. The air furnace called "Tonga" by the Chinese, can only be used when very good dense charcoal, such as that of the kompas tree, is available; in such cases it is preferred because it is said to make better tin, besides avoiding the labor of blowing. In most of the tin-mining districts all kompas trees within a practicable distance have long been cut down, and the air furnace has therefore been perforce abandoned. In spite of somewhat extensive travels in the Peninsula, the writer has but rarely seen the air furnace in use, one of the few localities being the district of Tras, in the heart of the mountain range that separates Pahang from Selangor; the tinstone in this district lies in very shallow gravel, is coarse and extremely pure. The air furnace is illustrated in Fig. 6, which shows the details of its very simple construction as seen when not in operation; when it is being worked, two clay tubes about 3 in. in diameter are clayed into the square aperture at the back, and the draft is regulated by stopping these up more or less. The furnace (a general view of which, taken from a photograph, is shown in Fig. 7) consists of a mass of kaolin or china-clay

* *Iron and Steel*, 1864, p. 273.

rammed inside a casing consisting of bamboos or poles stuck into the ground and held together by hoops of stout rattan; hoop iron may exceptionally be used, as near Kuala Lumpor, Selangor. After the furnace has been built it is allowed to dry for some months; when sufficient tin ore has accumulated a fire is lighted in the furnace and some charcoal is thrown in; the shaft is then filled with alternate layers of charcoal and tin ore, the charcoal being thrown on by basketfuls, each basket being about 2 ft. in diameter and 2 ft. high, while tin ore is thrown on by means of a square-mouthed shovel. The ore is damped from time to time, both to prevent the furnace-top from becoming too hot and to prevent the ore

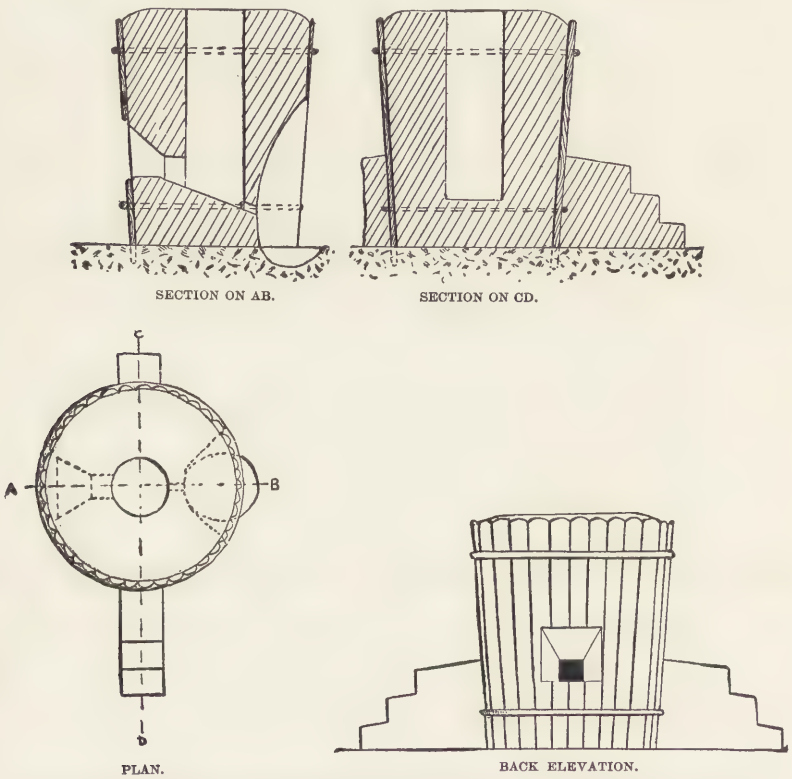


FIG. 6.—DETAILS OF CHINESE TONGA FURNACE.—SCALE $\frac{3}{16}$ INCH = 1 FOOT.

from being carried off by the draft. The tin soon commences to trickle out of the tap-hole and to collect in the float, which consists of a cavity hollowed out in the ground in front of the furnace and lined with china-clay beaten smooth. This float is kept filled with red-hot charcoal; a little slag runs out from time to time, and is drawn off in irregular crusts by means of the slag-rake shown in Fig. 8, the tap-hole being kept open by means of constant poking with the pricker (Fig. 9); a thin sapling is sometimes used in place of the latter. When sufficient tin has been collected it is ladled out by means of an iron ladle (Fig. 10), and is cast in sand molds, these being made by stamping the wooden pattern shown in Fig. 11 into a small bed of sand that is kept close to the furnace

for this purpose. The size of the ingot cast varies somewhat in different districts. In Tras the ingots run 7 to the bhara,* or about 76 lbs. each, and each furnace produces about 20 slabs in 24 hours, or say 1500 lbs. of metal. The ore, which is not always very well washed, yields 64 to 65% of metal; its average assay value is probably not far short of 70%. The consumption of charcoal is about 4 picul by weight for every 3 picul of tin produced, or approximately a weight equal to that of the ore smelted. The charcoal is here worth† $1 to $1.50 per picul (say about 35c. to 50c. United States money per 100 lbs. of ore smelted). There are generally two or three men employed about the furnace, in addition to those carrying in charcoal or ore and carrying out the slabs of tin. The slag as produced is examined by the smelter; if it carries unreduced ore, it is at once thrown into the furnace again with the ore charge; if not, it is kept for retreatment; it is stamped in foot-mills (which will be described further on) and washed to recover prills of metal, and is then

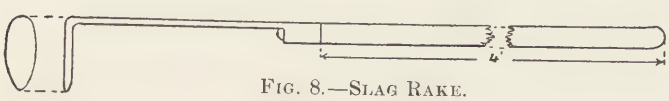


FIG. 8.—SLAG RAKE.



FIG. 9.—PRICKER.



FIG. 11.—INGOT PATTERN.

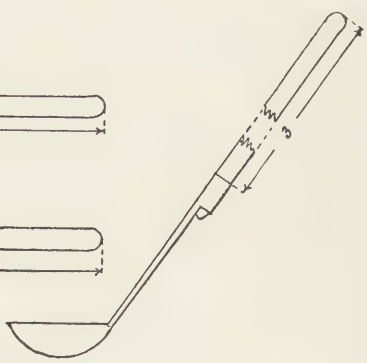


FIG. 10.—LADLE.

SCALE $\frac{3}{4}$ INCH = 1 FOOT.

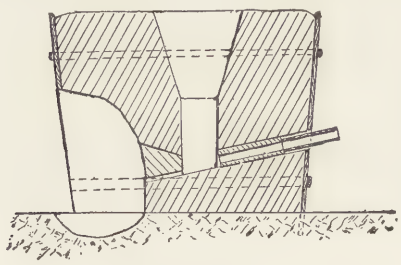
smelted again after all the ore has been treated, or at any other suitable opportunity; the slag thus produced is again stamped, washed, and smelted. It is then as clean as the ore-smelters can make it; in Tras nothing further seems to be done with it, because the district is so inaccessible that the cost of carriage on men's shoulders is practically prohibitive. When this district is opened up by roads this slag will probably, as in other places, be sold to the slag-smelters, of whom there are several near Kuala Lumpor (the capital and center of the tin-mining industry of Selangor), as well as in the other tin-mining centers of the peninsula. A rough analysis by the writer of a sample of the first slag gave about 30% tin (as metal and as oxide), silica, titanic acid, and protoxide of iron as the chief constituents; no tungstic acid could be detected.

The air furnace is used whenever possible; thus it is (or was in 1892) in use at Sungei Besi in Selangor, although suitable kompas-wood charcoal cost $2.80 per picul as against ordinary charcoal at about 80c. per picul. In this furnace the

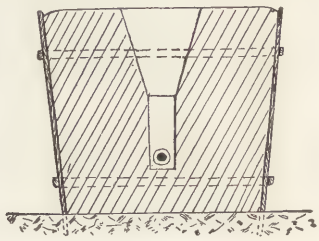
* The weights used by the Chinese in the Malay Peninsula are as follows: 1 bhara equals 3 picul; 1 picul equals 100 kati; 1 kati equals 16 tael. The normal picul weighs 133.33 lbs. avoirdupois, but there are many local differences; *e. g.*, in Tras and the whole of the Hulu Pahang the picul used for weighing bar tin is one-third heavier than the standard picul.

† The dollar here referred to is the Mexican trade dollar, worth at present about 44c. United States currency.

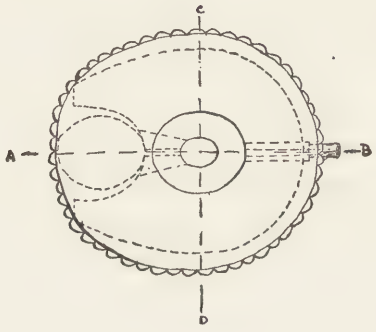
output was about 30 slabs per day. It would seem that though the mode of conducting the campaigns varies somewhat, the usual practice is to put through the whole accumulation of tin ore by continuous working and then to treat the slags as described. The cost may be estimated as follows per 24 hours: 12 picul of charcoal at $1.50 each, $18.00; labor (head smelters, $1.75, and 6 coolies, 50c. each, $3.00), $4.75; tools and wear of furnace, say $2.00; total, $24.75, or about $11 United States currency, making the smelting cost about 73c. per 100 lbs. of metallic tin produced. This does not include the after treatment of the slags. Although the air furnace is decidedly more simple in construction than the blast furnace, there are good reasons for believing that the former is a modification or development of the latter, and that it replaced some form of blast furnace in suitable districts. The air furnace is now, however, rapidly becoming extinct,



SECTION ON AB.



SECTION ON CD.



PLAN.

FIG. 12.—CHINESE BLAST FURNACE AT DREDA.—SCALE $\frac{3}{16}$ INCH = 1 FOOT.

owing to the difficulty of getting suitable charcoal, and to the fact that under British protection attempts are being made to preserve the forests, and to prevent promiscuous destruction of the best timber.

Accordingly the blast furnace is now the more frequently used in the Malay Peninsula. Although varying considerably in details the principle is always the same. The furnace here illustrated (Figs. 12 and 13) is one of a number, all practically identical, erected in a very remote district on the boundary between the Siamese-Malayan States of Reman and Jalor, the center of which is a flourishing Chinese village known as Dreda. This spot has rarely if ever been visited by any European except the writer, hence it may be taken for granted that the furnace there used presents the purest Chinese type unaffected by European influences; it is perhaps rather lower than the average of those in general use elsewhere. It

is built like the air furnace, but is sometimes very roughly hooped, and only lasts for a short campaign. The front of the crucible forms an arch about 8 in. high, which is closed when at work by a lump of clay with a tap-hole about 1 in. in diameter through it, as shown in Fig. 13. In case of need the whole lump can be taken out without destroying the furnace. Through a circular aperture in the back of the furnace a tuyère made of clay, carefully dried and baked, is inserted, and may project to a greater or less distance into the furnace; it is generally about 22 in. long, 2½ in. inside and 5 in. outside diameter, and is connected by means of a short bamboo blast-pipe with the bellows. The latter consists of a hollowed-out tree-trunk 12½ in. in diameter and about 10 ft. long; the piston is a block of wood attached to a long wooden rod that works through a wooden stuffing-box at one end; the piston is neatly packed with feathers or leaves. There are flap valves at either end for the intake, and similar delivery valves opening into a narrow wind-chest that runs along the front of the tree-trunk, so that it forms a double-acting blowing cylinder; all leaks are stopped with clay. One man works the bellows, for half an hour or so at a time, when he is relieved; sometimes two men work it for a short time, probably when the head man judges that more blast is required. The average rate of working is above 10 strokes per minute, so that it may be roughly calculated that the furnace receives about 75 cu. ft. of air per minute. As a rule the blast is stopped for between one and three hours in the middle of the day. The ore in this district is of very varying quality, and the statements of the various smelters are decidedly discordant as to the working of their furnaces. The average of their statements is that a furnace smelts between 1250 and 1670 lbs. of ore per 24 hours with a consumption of between 600 and 750 lbs. of charcoal. The daily production of tin is between 10 and 12 ingots, which here weigh 60 kati each, or say 800 to 960 lbs. of metallic tin. The price of the charcoal, which is of rather inferior quality, is in this district 4 baskets of 30 kati each for $1; this would make the cost of fuel about 22c., United States currency, to each 100 lbs. of metal; according to another smelter's statement the cost is about 35 to 40c. per each 100 lbs. of metal. It will be seen from the figures here given that the blast furnace has a smaller output than the air furnace—the latter having a much greater cubic capacity, but that it works with a much smaller consumption of charcoal, beside using charcoal of inferior quality. There appear to be generally about five men on a shift working the blast furnace. The method of collecting and casting the tin, and of removing the slag, are identical with the corresponding operations in the air furnace. The system of working varies in different districts. In the Dreda district the smelting campaign seems to be carried on continuously from start to finish; in other places the work is intermittent. Only a rough estimate can be made of the cost of smelting in the Dreda district, the cost per 24 hours being probably somewhat as follows: Charcoal, 15 baskets at 25c. each, $3.75; labor (head smelters, $1.75; 10 coolies, 50c. each, $5.00), $6.75; furnace repairs and tools, say $3.00; total, $12.50, or about $5.50, United States currency, making the cost about 60c. for each 100 lbs. of metallic tin produced. The fuel cost is probably too low, and the total cost ought to be probably 70c. to 80c. per 100 lbs.

In Perak the furnace and general mode of working closely resemble the Banka

practice, which will be described in detail below. The furnace consists of a roughly cylindrical or rectangular block of clay or rough brickwork, about 6 ft. square and of nearly the same height, the furnace proper being a cylindrical or slightly tapering cavity lined with clay, about 2 ft. to 2 ft. 6 in. in diameter and 5 ft. deep. The rear wall of the furnace is built up 2 ft. or so above the general level of the furnace top so as to form a kind of screen, apparently to protect the blowing cylinder from damage by pieces of burning charcoal thrown out by the blast; the tap-hole and float are similar to those of the ordinary Chinese furnace. The tuyère and blowing cylinder are identical with those previously described, except that the latter is perhaps a little larger and is generally worked by three men. Mr. Doyle* states that the furnaces work the whole year round, but only at nights; a fresh lining and tuyère are put in every day. He gives the following as the average cost of smelting at the time of his writing, the costs being reckoned on a night's work: Charcoal, 18.58 picul,† $31.96; 1 overman, $1.20; two relays of 3 coolies, $5.40; food and miscellaneous stores, $2.35—total, $40.91. Ore smelted in one night, 32 picul; metallic tin obtained, 19.52 picul. This would make the cost of smelting about 62c. per 100 lbs. of tin; it is right to observe that when Mr. Doyle wrote the value of the dollar in the Malay Peninsula was about twice what it is to-day. Mr. J. E. De La Croix‡ states that although smelting used formerly to be done only at night in Perak it is now carried on day and night at some furnaces. The rectangular brick-built furnace is, according to him, 6 ft. high, 7 ft. 6 in. broad, and 5 ft. 6 in. deep; the furnace proper is semicircular, 18 in. in diameter, and the tuyère is inclined 45°. Such a furnace, with blower complete, costs about $100. It treats 36 picul of ore per night at a cost of $2.85 per picul. Smaller furnaces, closely resembling those in use at Dreda, are also employed. It is stated that quite recently the Chinese have succeeded in replacing a portion or the whole of the charcoal by desiccated or torrefied wood, thus producing a marked economy, but no definite data on this subject are available.

Treatment of the Slag.—The slag collected during a campaign is first treated mechanically to separate the prills of metal that it contains. It is crushed in various ways, but mostly in small foot-mills, which are simply tilt-hammers worked by the foot. The body of the hammer is of wood and the head consists of a piece of hard stone; they are often constructed without a particle of iron. Similar foot-mills have been fully described by the writer§ in a previous paper. The slag is pounded in these little mills and then washed in a short wooden sluice, 5 to 6 ft. long and 12 to 15 in. wide, with a light stream of water, in precisely the same way that the tin ore is cleaned. The tin thus obtained is simply melted, and the slag is collected and smelted in the ordinary ore furnace, more tin being thus obtained. The slag produced in this second operation is treated in the same way once more, and the slag resulting from the third fusion is sold to the slag smelters. The slag furnace is shown in Fig. 14; it consists of a clay body, iron bound, about 36 in.

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*Patrick Doyle: *Tin Mining in Larut*, 1879, p. 19.

†The Larut picul is the heavy or Malay picul, weighing nearly 150 lbs. avoirdupois.

‡*Les Mines d'Etain de Perak*, 1882.

§Henry Louis: *Trans. Am. Inst. Min. Eng.*, "A Chinese System of Gold Milling," Vol. XX., 1891, p. 324.



FIG. 14—CHINESE SLAG FURNACE.

high and 27 to 30 in. outside diameter. The bottom consists of a cast-iron basin into which clay is rammed, and it is carried on short iron legs so as to be 12 to 18 in. above the ground. There is a small tap-hole in front and one clay tuyère about $1\frac{3}{4}$ in. inside diameter and $10\frac{1}{2}$ in. long; the blowing cylinder is of wood quite similar to that used for the large furnace, but much smaller, being about 5 to 6 ft. long and 9 in. in diameter. The slag, after being pounded and washed, is smelted in this slag furnace; good slag is said to yield 100 lbs. of metal from 700 lbs. of slag, and with good charcoal it is possible to get 6 to 8 slabs of tin weighing 50 kati each, say about 470 lbs. altogether, with a consumption of 15 picul of charcoal. From these figures it would seem that slag requires 30% of its weight of charcoal to smelt it. The slag furnace here described is situated at Sungei Besi in Selangor; in this locality charcoal of a quality sufficiently good for the slag furnace is not dear, costing only 60c. to 80c. per picul. Two men appear to be enough to work one of these furnaces. Another slag smelter near Kuala Lumpor stated that he was smelting about 2000 lbs. of slag in 24 hours and was getting a little over 100 lbs. of tin from it; he was evidently working a much poorer slag. The tin drops into a float, which is again merely a hole in the ground lined with clay, the treatment being identical with that in the ore furnace. The resulting slag is again pounded, washed, and resmelted in the same way. The slag resulting from this operation is pounded and washed, and if the smelter judges it to be worth while, he will smelt it once more, though this is not often done. Pounding and washing is always the last stage of the process. Concentrates thus obtained are sometimes treated by heating to redness in an iron pot, when the tin liquates out, leaving a slaggy mass that can be treated further if required.

The average yield of metal from ore throughout the Malay Peninsula is evidently higher than might have been expected, being probably about 65%, including the metal recovered from the slags. The average amount of tin contained in the ore is probably a little under 69%, so that the loss would amount to about 6% of the tin in the ore. A good deal of this is probably left in the slags, so that the loss due to volatilization and to ore mechanically carried off must be comparatively small. The Chinese smelters keep their furnace tops cool by throwing water on them from time to time and by regulating the blast, the pressure of which is of course very light. Most of the volatilization that takes place seems to be at the tap-hole; incrustations, consisting largely of oxide of tin, can generally be scraped from the arch of the furnace above the tap-hole, but no attempt seems to be made to collect this material.

Practically the whole of the tin produced in the Malay Peninsula is forwarded to Singapore or to Penang, whence it is reshipped to the various markets. Malacca was at one time one of the chief ports, whence the name of Malacca tin is derived, but now there is absolutely no tin whatever shipped from there.

The metal produced in ore-smelting is soft and apparently fairly pure,* but needs refining before it is fit for the European market. Slag-metal is generally very impure, and is for the more part simply remelted and recast, and shipped to China, where there is a market for hard tin. The refining is all done by large Chinese or European firms in Singapore and Penang; the tin is

* According to notes kindly supplied by Mr. John McKillop, late of Pulo Brani, Singapore.

melted in open, almost hemispherical cast-iron pans, 4 to 5 ft. in diameter and 1 ft. 9 in. to 2 ft. deep, set over a wood fire. The ingots as received are remelted in these pans, the liquid metal is skimmed, some grease is thrown on the surface, and after being allowed to stand for some hours it is ladled into iron molds. The dross skimmed from the surface, and that deposited on the bottom of the pan, is consolidated into cakes by beating while hot, and is exported to Amoy, where it appears to be utilized in some way; it contains 40 to 50% of tin, partly as metal and partly as oxide. The loss in refining is between 0.4 and 0.8%; the cost including loss is between 20 and 30c. per picul, or about 7.5c. to 11c. per 100 lbs.

Mr. J. E. De La Croix* describes the refining process at Penang in very similar terms. According to him the pot is of wrought iron, 5 ft. in diameter and 25½ in. deep, weighs 1300 to 1700 lbs., and can contain 40 picul (say 5500 lbs.) of tin; it is heated by a reverberatory wood-fire. The cost of the entire refining furnace is about $235, of which the pan costs $70 to $90. Two hours after the fire has been lighted the tin is charged, and is all melted in two hours; after stirring and skimming, etc., in about half an hour more 15 picul of tin are ladled out and cast into molds; 15 picul of unrefined tin are then charged into the pot, and so the process continues. There are thus 205 picul treated in 24 hours, the yield of refined tin being 196 picul, equal to a loss of about 4%. The fuel used is cordwood, of which 200 pieces are burned in the 24 hours; there are six men to a furnace, who are paid altogether 4½c. per picul of refined tin produced. The total cost of refining is given as 52c. per picul. It must be remembered that when Mr. De La Croix wrote the Straits dollar was worth about 80c. to 90c., United States currency.

The following is an analysis of Chinese refined tin of average quality from the Malay Peninsula: Sn, 99.87%; Sb, 0.03%; As, faint trace; Pb, 0.01%; Fe, 0.09%; Cu, none—total, 100.

The Chinese Process at Banka.—As previously mentioned, this process was introduced into Banka about the middle of the last century and continued in universal use until about 1870, when certain improvements, to be referred to below, were introduced by the Dutch government engineers. The original Chinese furnace† was oblong or rectangular in plan, about 13 ft. long by 5 ft. wide, and 4 ft. 3 in. high. It was made of sandy clay well rammed, and the furnace proper and the arch above the float were cut out in it, so that the shape of the interior is very similar to that of the Larut furnace already described, which latter was indeed originally introduced by the Chinese from the Archipelago. The forehearth or float and the horizontal blowing cylinder are also practically identical. The fire is started in the usual manner, and tin ore and charcoal are thrown in as required; the smelter keeps the tap-hole open by constant poking with a thin sapling, and in doing so pulls out a certain quantity of charcoal, unreduced ore, and thick, sticky slag, all of which are at once returned to the furnace. As soon as enough tin has accumulated in the float to make 18 or 20 ingots, it is ladled out and poured into molds; these were originally of

* *Op. cit*, p. 59.

† *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1872, I., p. 217; P. H. Van Diest: "Het Tin Smelten op Bangka."

sand rammed around a wooden pattern, but were subsequently of cast iron; when the tin in the mold is on the point of setting, an iron stamp bearing the word "Banka" is pressed into the top surface.

Smelting is carried on only at night, the usual hour for commencing being from 8 to 10 P.M., and continuing until 6 to 7 A.M. There is only one smelting campaign every year, commencing about November, or October at the earliest, and always concluding by the end of December; the time of beginning the campaign is varied in accordance with the quantity of ore on hand, and its average duration is from 20 to 30 nights. At the beginning of the campaign each furnace will yield about 60 ingots of tin per night, toward the end only about 50, the ingot weighing 65 to 70 lbs. After every 4 nights' smelting, the furnace is laid off for a night and allowed to cool down for any needful repairs. The charcoal used is of good quality and is prepared by the gang of charcoal-burners employed by each mine or smelting establishment; it is burnt in piles about 35 ft. long, 12 ft. wide, 6 ft. high at the front and 9 ft. at the back, composed of stout logs laid parallel to the shorter side of the pile, which is then covered with a layer of earth in which holes are left at various heights. The process of burning takes about a fortnight, and when complete the charcoal burner brings the coals in to the smelting-house. A charcoal burner can supply in one year enough charcoal to keep a furnace going for 5 or 6 nights, *i.e.*, enough to smelt about 150 picul (about 20,000 lbs.) of tin. Fully 7 parts by weight of charcoal are required for the reduction of 10 parts of well-washed ore.

The slag produced contains prills of tin and unreduced oxide of tin; in 5 to 6 nights' ore-smelting there is enough slag produced for a night's slag-smelting, yielding some 30 to 40 ingots. The slags produced in this operation are resmelted, and then sold to the slag-smelters, who dress and resmelt them several times. Their last slag still contains several per cent. of tin. The mines smelt out on the average about 69% of metal from the ore, and the slag-smelters about 1%, so that the total yield is about 70% of the weight of the ore, which actually contains about 75%.

In the year 1854* a commission to hold an inquiry into the amount of tin lost by this method of smelting was appointed by order of the Dutch government, owing to the fact that slags had been found to contain as much as 19% of tin, out of which from 1 to 6% was in the form of prills. It was estimated at that date that Banka was producing annually 80,000 picul of tin and 8822 picul of slag supposed to contain on the average 15.7% of tin, equal to an annual loss of 1386 picul. This commission carefully watched a short smelting campaign conducted by the Chinese in one of the best smelting works on the island, and reported the results fully. The smelting works contained two furnaces; smelting commenced on the night of November 1, 1854, and was continued nightly, there being no smelting on the nights of the 4th, 8th and 12th, so as to enable the furnace to be repaired. Ore was smelted on 8 nights, and then the slag-smelting commenced, the slags produced during the slag-smelting being resmelted three times, or until the Chinese headmen declared they could no longer treat them at a profit. The slags were then sold to slag-smelters, who crushed, washed, and smelted them,

* *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1878, II., p. 29; C. De Groot: "Rapport over de tin slakken, welke op Bangka onbenuttigd worden weggeworpen."

and finally threw them aside as no longer payable. The results of the ore-smelting are shown in the subjoined table, all the weights being in Amsterdam pounds (about 1.07 lbs. avoirdupois):

Number of Night.	Furnace Number.	Weight of Dry Ore Smelted.	Weight of Tin Produced.	Weight of Charcoal Consumed.	Number Hours in Shift.
First.....	1	8,148.0	4,127.2	3,877.7	15¾
First.....	2	7,878.4	4,421.0	3,842.1	16
Second.....	1	6,050.4	3,641.1	3,104.6	14
Second.....	2	5,986.1	3,679.9	3,104.6	14¼
Third.....	1	5,982.9	3,456.0	3,569.0	14
Third.....	2	5,851.5	3,789.7	3,569.0	14½
Fourth.....	1	6,019.8	3,591.1	3,132.0	12¾
Fourth.....	2	6,018.0	3,741.0	3,132.0	12¾
Fifth.....	1	6,462.9	3,796.0	3,936.0	13½
Fifth.....	2	6,404.2	3,877.6	3,660.5	13½
Sixth.....	1	6,361.1	3,624.0	3,186.0	13
Sixth.....	2	6,224.9	3,739.3	3,283.5	13
Seventh.....	1	5,768.7	3,637.0	3,276.4	12¾
Seventh.....	2	5,800.6	3,561.3	3,346.1	12½
Eighth.....	1	6,076.3	2,682.6	3,032.2	12¾
Eighth.....	2	6,115.7	3,788.1	3,103.8	12¾
Totals.....	.....	101,149.5	60,162.9	54,155.5	216¾

The account of the slag-smelting is rather more involved because it occasionally happened that two different slags were being treated simultaneously, and because various furnace residues were smelted with the slags. The following table shows the chief results obtained, the weights used being Amsterdam pounds:

Slags Treated.	Weight of Slags.	Weight of Tin Extracted.	Percentage of Metal Extracted.
I. Slag produced in ore-smelting.....	19,093.5	5,938.8	31.1
II. Slag from first slag-smelting.....	9,733.7	2,517.9	25.7
III. Slag from second slag-smelting.....	7,237.3	1,047.1	14.4
IV. Slag from third slag-smelting.....	3,141.9	325.7	10.3
V. Slag from fourth slag-smelting.....	2,916.5	643.1	22.0
Totals.....	42,122.9	10,472.6	.....

Of these, the first four slags were treated by the mine-owners, the weight of charcoal consumed being 17,375 lbs., and the time employed 73¾ hours. The slag was then sold to the slag-smelters, who used in smelting it 977 lbs. of charcoal and took 13¼ hours in the smelting, using a smaller furnace than was used for ore-smelting. The slag was thrown away in three parcels as follows:

1,663.1 lbs. of slag assaying 20.5% of tin equal to..... 340.9 lbs. of tin.
3,049.1 lbs. of slag assaying 20.33% of tin equal to..... 620.0 lbs. of tin.
2,988.4 lbs. of slag assaying 8.33% of tin equal to..... 249.0 lbs. of tin.

7,700.6 lbs. of slag equal to a total of.....1,209.9 lbs. of tin.

The above results include the tin obtained by the mechanical treatment of the slags. According to G. P. A. Renaud,* slags in Banka are crushed by means of a kind of iron flail, to pass through a sieve made of bamboo with a $\frac{3}{16}$-in. mesh, and are then washed in a sluice-box 18 ft. long and 18 in. wide. The washed

* *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1883, II., p. 68: "Behandeling van de tin slakken op het eiland Bangka."

material is broken down to pass a sieve of $\frac{5}{16}$-in. mesh, and again washed repeatedly; the grains of tin are then separated out, and the residual slag may be smelted.

An analysis of the results above tabulated shows that the average length of the night's shift of these smelting furnaces was 13½ hours, and that they treated per night and per furnace an average of 6743 lbs. avoirdupois of dry ore, with a consumption of 4011 lbs. of charcoal and the production of 3610 lbs. of metallic tin. In the slag-smelting 7 nights' work was done, the average length of the shift being 12½ hours; each night's work cannot be quite fairly averaged, yet by working out the above figures for the first four slags only, it would seem that one of the large furnaces could smelt on an average 6970 lbs. avoirdupois of slag with a consumption of 3089 lbs. of charcoal. The ore yielded in the ore-process 59.480% of metal, and in the slag-smelting 10.353%, giving a total yield of 69.833%. There were thrown away in the refuse slags 1.196%, making the gross amount of metal present 71.029%, not allowing for losses by volatilization, etc. The consumption of charcoal was 102.6% of the weight of the metallic tin produced; upon the weight of the ore smelted it was 53.5% in ore-smelting and 18.1% in slag-smelting, making a total of 71.6%.

These figures illustrate very thoroughly the technical and economic results obtainable in the Chinese furnace, and a few others obtained by a government commission 14 years later will be given presently. From these the cost of smelting under any given set of conditions can be approximately calculated; the following are actual figures obtained from smelting on a rather small scale in the island of Singkep, the furnaces used being similar to those in Banka, but much smaller, and being rented to the owners of the ore, so that the first item represents a small profit besides wear and tear and interest on capital outlay. Cost of smelting per night: Furnace hire, 4.06 fl.;* wages (smelter, 2.80 fl.; helper, 1.68 fl.; 6 coolies, 6.00 fl.), 10.48 fl.; charcoal (1323 lbs. avoirdupois), 27.72 fl.; total, 42.26 fl., equal to $16.90. Ore smelted per night, 2640 lbs.; metallic tin obtained, about 1750 lbs.; cost of smelting per 100 lbs. metallic tin, $0.96.

As previously stated, attempts were made in various ways to improve this original Chinese method, but none have obtained anything like general acceptance with the exception of the modifications introduced in 1868 by Dr. C. L. Vlaanderen; his furnace, known as the rectangular or Vlaanderen furnace, was experimented on and was found to extract about 3% more metal than the old form, with less charcoal, less labor, and in less time. In 1875 there were 90 of these furnaces at work in Banka, and now they are almost exclusively used both in Banka and in Billiton. The main† improvement is the production of the blast by means of a centrifugal fan, thus saving about 0.40 fl. per picul of tin. His own description of his furnace is as follows: A foundation is excavated to a depth of some 7 ft.; the pit is paved with stones, and clay, such as is used by the Chinese in furnace building, is rammed in. On this foundation the furnace is erected by ramming the same kind of clay into a wooden frame 5 ft. 6 in. wide and 8 ft. long to a height of 4 ft. 9. in. When the clay is dry the frame is removed and the furnace proper and the fore-arch are cut out in the block, the arch being similar to that

* Exchange calculated 1 florin equal to 40 cents United States currency.

† *Jaarboek van het Mijnwesen in Nederlandsch Oost-Indie*, 1872, I.

of the old furnace, and the furnace cavity being 3 ft. 9 in. square and 28 in. deep; this cavity is floored with a layer of bricks. In the middle brick of the second row an inclined hole must be cut to form the tap-hole. Upon this floor the furnace is built up of ordinary good bricks to the dimensions shown in Figs. 15 and 16, taken from the above-quoted work, which show both the furnace and the blowing fan. There are three tuyères, holes for which must be left. The inside of the furnace must be lined 3 in. thick with the same clay as is used by the Chinese for daubing their furnaces, consisting of fireclay mixed with a little salt. Clay tuyère-pipes preferably thickened on the top side are used; they are much stouter than those used in the old furnace, and therefore only need renewal every two or three nights. The tuyères are connected to the fan by means of a horseshoe-shaped blast-pipe, which may be of wood, and made air-tight with clay. The blower has the shape of an ordinary centrifugal fan very simply constructed, with four sheet-iron blades; it is driven by means of a very crude rope gearing, passing over grooved wooden sheaves, from a small overshot water-wheel of the usual Chinese type.

When the furnace has been built and well dried by a small fire it is ready for use; after a run it must be well cleaned of all adhering pieces of slag, etc., the clay lining repaired, and the tuyères renewed in case of need. The furnace is then filled with ignited charcoal and the fan started at about 300 revolutions per minute. As soon as the top layer of charcoal is red-hot the first tin ore is charged, and then ore and coal are fed in alternate layers in the proportion of 35 kgms. (77 lbs.) of the former to 30 to 32 kgms. (66 to 71 lbs.) of the latter. The whole of the night's charge of 3000 kgms. of ore (6614 lbs.) is charged in 8 or 9 hours. After 7 or 8 nights of ore-smelting, a night's slag-smelting is done, the slag being broken into pieces about the size of a hen's egg and mixed with 20% of its weight of coral limestone, broken to the same size. This is charged in layers of 50 to 54 kgms. (110 to 120 lbs.) of the mixture and 30 to 32 kgms. (66 to 71 lbs.) of charcoal.

A government commission was appointed in 1868 to examine and report on this furnace. The following comparative test is given in much detail by the commission. Two of the old Chinese furnaces and one Vlaanderen furnace were employed to smelt a parcel of ore which assayed 76.39% of tin. The two Chinese furnaces worked 8 shifts, or altogether 134 hours 22 minutes of the time of one furnace. The quantity smelted was 51,233 lbs. of wet ore, equal to 50,678 lbs. of dry ore, or 6335 lbs. per furnace in a shift of 16¾ hours. The slag, amounting to 5201 lbs., was resmelted twice and then sold to the slag-smelters. The charcoal used was: In ore-smelting, 33,215 lbs.; in first slag-smelting, 2544 lbs.; in second slag-smelting, 1909 lbs.; total, 37,668 lbs. The quantity of metallic tin obtained was as follows: From the ore-smelting, 33,146 lbs.; from the first slag-smelting, 1288 lbs.; from the second slag-smelting, 338 lbs.; by the slag-smelters, 132 lbs.; total tin obtained, 34,904 lbs. Percentage of metal obtained from the dry ore, 68.87. The slag rejected by the slag-smelters still retained 9% of tin.

The Vlaanderen furnace worked for 3 shifts or 48 hours and smelted 21,078 lbs. of wet ore, equal to 20,873 lbs. of dry ore. The amount of slag produced was 2132 lbs., and this was smelted once in admixture with 425 lbs. of broken

coral limestone. The quantity of charcoal used was: In ore-smelting, 13,417 lbs.; in slag-smelting, 2055 lbs.; total, 15,472 lbs. The amount of metallic tin produced was: From the ore, 14,287 lbs.; from the slag, 762 lbs.; total, 15,049 lbs. Percentage of total metal from dry ore, 72.13. The residual slag contained

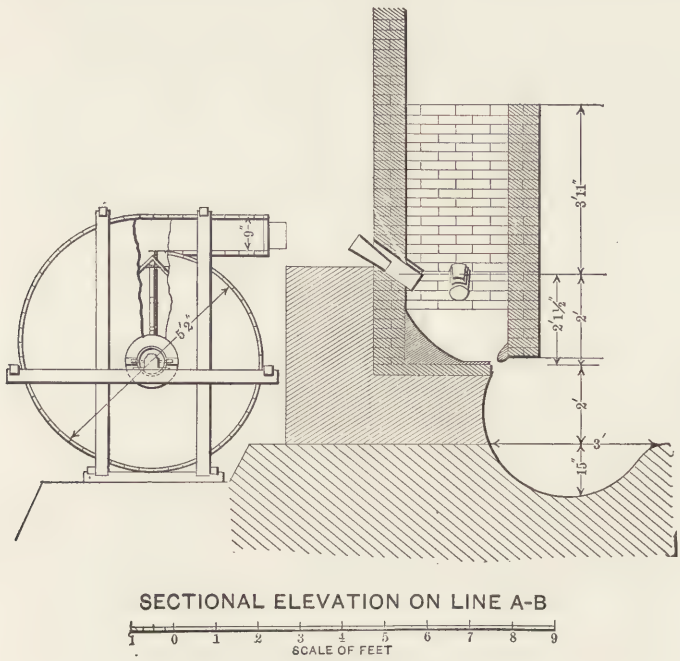


FIG. 15.—VLAANDEREN FURNACE.

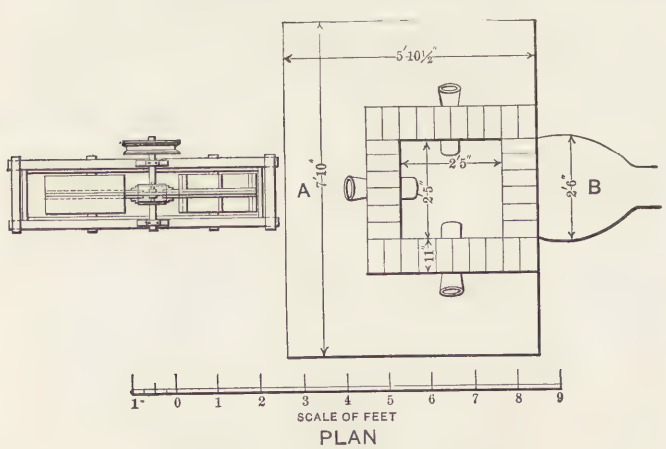


FIG. 16.—VLAANDEREN FURNACE.

only 2% of tin, and neither it nor that obtained in the Chinese furnaces paid to resmelt.

It will be seen that the Vlaanderen furnace produced a saving of time amounting to about 18%, a saving of coal to the extent of about 5%, and extracted over

3% more metal from the ore, these results accounting sufficiently for its very general adoption by the Chinese in the Malay Archipelago. The costs of smelting in this furnace have been calculated by G. P. A. Renaud* in more recent times; in his opinion the centrifugal fan is the only improvement that the Chinese smelters owe to European science. He calculates the cost of a small smelting house with one furnace, fan, etc., complete, as follows, the house being as usual a substantial high-roofed shed, thatched with leaves and open at all four sides: Shed, 500 fl.; furnace, 150 fl.; fan and blast-pipes, 200 fl.; water-wheel and driving pulley, 150 fl.; ditches for the water supply, 150 fl.; one set of furnace tools, 90 fl.—total, 1240 fl., equal to $496.

On this basis he considers that 215 fl. should be written off annually for repairs and depreciation; taking the furnace to smelt 35 picul per night, this amounts to 0.60 fl. per picul if the furnace works for 10 nights in the year, and to 0.30 fl. if it works for 20, the usual smelting campaign lying mostly between those limits. Charcoal is brought in in quantities sufficient for a night's work, say about 53½ picul, at prices that range from 45 to 60 fl. in different districts, so that the price of the picul of charcoal may be averaged at about 1 fl. In round numbers 100 picul of ore require for their smelting 100 picul of charcoal and produce 68 to 70 picul of tin, together with slags that yield some 3 picul more.

The daily cost of smelting, with a production of 35 picul, say 4670 lbs. of tin, may be set down as follows: Tuyère pipes, 1.50 fl.; rope for driving gear, 1.00 fl.; furnace tools,† 18.00 fl.; saplings for pricking out tap-hole, 0.50 fl.; wages (smelter, 7.00 fl.; 2 helpers, 3.60 fl.; 3 coolies, 1.50 fl.; rations for 6 men, 6.00 fl.), 18.10 fl.; charcoal, 53.00 fl.; rations for charcoal-burner, 1.00 fl.—total daily cost, 93.10 fl., equal to $37.24, or 2.66 fl. per picul, which, with depreciation, etc., as above, makes the total cost of tin-smelting from 3 to 3.25 fl. per picul (90c. to 97c. per 100 lbs.) of metal.

The tin produced in the old type of Chinese furnaces in the Malay Archipelago was of exceptional purity, and as such always commanded a high price in the European markets. Its composition is shown by the following analyses:

	Tin from Siak.‡ Per Cent.	Tin from Banka.§ Per Cent.
Tin	99.966	99.90
Iron	0.034	0.20
Sulphur	Trace.	Trace.
Totals	100.000	100.10

The following are the maximum, minimum, and average amounts of the impurities present in six different samples of Banka tin:

	Per Cent.	Per Cent.	Per Cent.
Fe	0.006	0.0196	0.0108
Pb	None.	Trace.	.....
S	0.0027	0.0099	0.0052
C	Trace.	Trace.	Trace.

* *Jaarboek van het Mijnwezen in Nederlandsch Oost-Indie*, 1884, I., p. 5: "Over de Chineesche ontginningswijze van Tinerts op het Eiland Bangka."

† This is as given by the Chinese, but would seem to be too high.

‡ P. Doyle, *op. cit.*, p. 23. § *Berg- und Hüttenmännisches Jahrbuch*, 1864, *loc. cit.*

It was accordingly never found necessary to refine this tin, an occasional remelting being all that was ever required.

Here, however, as in most other cases, improvement in the metallurgical process has been attended with deterioration in the product. The case has been well put by Prof. A. C. Oudemans,* who says: "It is well known that Banka tin always had a high reputation for its great purity, and used in former times to fetch a considerably higher price than other grades. As long as the primitive Chinese method of smelting was followed there were never any complaints respecting the quality of the metal, but during the past 30 years or thereabouts —and only since the introduction of the fan-blast into Banka—complaints have arisen from time to time concerning the impurity of the metal. Sometimes hard needles or grains occur in the soft mass of the tin, which have been determined to be an alloy of tin and iron." This is corroborated by P. Van Dijk,† who says that the main impurity in Banka tin is the tin-iron alloy, of approximately the formula $FeSn_2$, which is formed, not by contact of the molten tin with iron tools, but by the presence of iron compounds in the tin ore, and especially, according to J. P. J. Van der Does de Bije, by iron pyrites. The formation of this alloy is caused by excessive furnace heat, due to the fan running too fast, the temperature being high enough not only to reduce iron ores to the metallic state, but also to form and to fuse the very refractory tin-iron alloy.

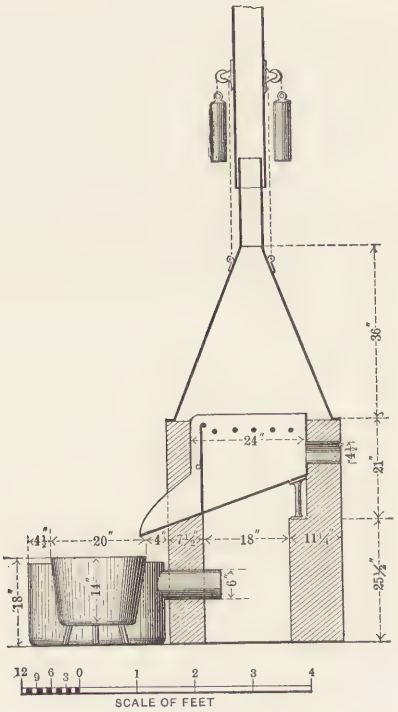
Several parcels of Banka tin were rejected, owing to their impurity, about 1890, and were refined by Van Dijk, who reports his experiments fully. He has in consequence thereof proposed to introduce into Banka liquating furnaces similar to those that have been in use for some time in Billiton, and on the same principle as the Saxon liquating hearth. The furnace used by Van Dijk at Amsterdam is shown in section in Fig. 17, and is practically identical with the Billiton furnace, except that the liquation plate is made of wrought iron and is heated from below, the ingots of tin being placed on this plate on a bed of charcoal; as they are gradually heated the pure tin liquates out, leaving a residue that has to be liquated at a higher temperature; the tin from the first and the second liquations were practically pure, containing respectively 99.84 and 99.79% of pure tin, the rest being practically all iron. The residue was a heterogeneous mass consisting of metallic tin and a dark crystalline alloy, containing 19.2% Fe and 80.8% Sn, corresponding closely to the formula $FeSn_2$. The result of the liquating operation was to show that the original impure tin had contained about 1% of impurity, almost wholly iron, while the loss in liquating was 0.33%, due to water collected in shrinkage cavities in the ingots, sulphur present in small quantities, surface impurities, oxygen in the skin of the ingots, and actual loss in the furnace. Two smaller parcels were liquated in an iron muffle with a V-shaped, sharply inclined bottom, fitted with a perforated iron shelf upon which the ingots were placed; the results were similar, but the total amount of impurity was only about half as great as in the first parcel. In the latter, tungsten was detected in the shape of a couple of small, hard, infusible lumps that formed at the bottom of the float, which consisted in the main of ferriferous tungsten. Tungsten may

* *Jaarboek van het Mijnwesen in Nederlandsch Oost-Indie*, 1890, I., p. 24: "Over Verontreiniging van Bangka Tin."

† *Ibid*, p. 5.

thus be utterly disregarded as an impurity in Banka tin. The cost of refining the large parcel of about 1,359,850 lbs. was: Fuel, 976.32 florins; labor, 3449.12 florins —total, 4425.44 florins, or $1770. The cost of refining per 100 lbs. of metal at Amsterdam was therefore just about 13c., and would no doubt be still less at Banka.

The refining process, as now practiced at Banka, has been quite recently described by Mr. O. H. Van der Wyck;* according to him not all the tin produced is refined, but only the last four or five blocks produced in each night's smelting, together with any that do not pass the usual physicial tests, and all the slag tin. The refining furnace seems to be somewhat similar to that used in



VERTICAL SECTION

FIG. 17.—DUTCH REFINING FURNACE.

Singapore and Penang; it consists of a shallow oval iron melting basin, having a capacity of about 200 blocks of tin, say 15,000 to 16,000 lbs., over which passes a sheet of flame maintained by a hand blower, the fuel used being wood. The tin is piled up in this basin, and as it melts it flows from it into a receiver of the same size as the pan, and thence into the "controlling pan," a small pan with a capacity of about 1100 lbs. of tin. The metal that passes into this latter pan is quite pure; the tin-iron alloy accumulates in the receiver, whence it is removed by means of perforated ladles that allow the pure tin to drain away from the alloy. From the controlling pan the pure tin is decanted into a large pan, out of

* *Seventeenth Annual Report of the United States Geological Survey*, 1895-96, Part III., p. 239.

which it is ladled into molds. The pans are set in sand, which stops any tin that might leak out from cracks in them; it has been found that, in the absence of sand, tin will make its way into the warm ground to a depth of as much as 13 ft.

The tin-iron alloy is melted in small clay furnaces like the ordinary Chinese furnace and cast into blocks; these are then subjected to liquation on a series of iron plates, down which the tin flows into a pan placed at their foot, while the separated tin-iron alloy is alternately left as a sandy mass on the top plate; it is at present looked upon as worthless.

The loss of metal in this refining process is about 2%, but as little more than 10% of the entire tin output is refined, the loss upon the total production amounts to only 0.2 to 0.25%. Refining increases the cost of production of the metal by about 0.11 florins per picul, equal to 33c., United States currency, per 100 lbs.

The liquating hearth used at Billiton consists of three brick walls in which is set an inclined cast-iron plate, terminating in front in a pair of iron steps, below which is set a refining kettle or float at the open side of the furnace. A charcoal fire is built on the plate and is maintained by means of an air-blast through a tuyère in the rear wall. The blocks of tin to be liquated are laid on iron cross-bars a little above the fire, or laid direct on the latter. The molten tin dropping down trickles through the charcoal and runs down the sloping liquation-plate into the refining-kettle, whence it is ladled into molds.

It may be thought that too much space has been devoted to these primitive Oriental processes, but it must not be forgotten that their importance is in reality very great. As far as the writer knows they have never been adequately described, and some indeed not at all, in the English language. Moreover, we have here an almost unique opportunity of tracing the development of a metallurgical process from its crudest beginnings, such as is not easily obtainable in any other branch of the subject, and this series of processes has therefore a high historical value, more especially when it is remembered that the various stages probably present the closest analogy to the mode of evolution of the process of tin-smelting in Great Britain. Finally, far more tin is produced by what has here been called the Chinese process than by any other. In the first place, about three-fourths of the tin production of the Malay Peninsula, or all of it that is not smelted by the Straits Trading Company, Limited, at Singapore, is thus produced, say about 30,000 tons of tin per annum. The whole of the tin produced in the Malay Archipelago, or about 11,000 tons, the whole of that produced in Western Siam, Burma, and the Siamese Malayan States, supposed to be about 5000 tons, and the whole of that produced in the eastern parts of Siam and Southern China, an amount known to be large but that can only be roughly guessed as somewhere about 20,000 tons, is all smelted by one or other of the above variations of the Chinese process. The same may be said of the small amount, some 40 tons, produced by Japan, together with that smelted by other minor producers in the far East. Thus it may be fairly said that this process is responsible for 60,000 to 70,000 tons of tin annually, or some two-thirds of the world's total production.

In addition to the above it may be mentioned that recently tin-smelting has been attempted by the Chinese in water-jacket furnaces of the American type,

the blast being supplied by Root blowers. We have no data as to the result of this experiment, but unless conducted with considerable metallurgical skill, difficulties will no doubt be met with, owing to the strong blast thus obtained, which will probably cause loss of tin, both mechanically and by volatilization, and will also, on account of the higher heat produced, yield a more impure metal, which will need a more thorough refining process than the Chinese have ever yet been in the habit of employing.

A good deal of concentrated tinstone (barilla) is exported from Bolivia as such, but some is also smelted in the country itself in small blast furnaces, charcoal being the fuel employed. The tin thus obtained is very impure, and it is exported as crude tin, the natives making no attempt to refine it, the object being chiefly to save freights. It seems that no attention is paid to any treatment of the slags, so that the loss of metal is very heavy, probably quite 20%. It will be seen that the Mexican smelting process is similar.

A water-jacket furnace has also been erected recently at Potosi to smelt Bolivian ores, and though no details are obtainable, it is stated that the company owning that furnace is working at a profit; the tin thus produced is probably also refined in Europe.

There is much to be said in favor of the method of smelting in the blast furnace, provided that suitable ores and fuel are obtainable, and it seems to-day an open question whether the water-jacket furnace, properly handled, might not do at least as good work as the reverberatory furnace in ore-smelting, while for slag-smelting there seems to be actually some evidence of its superiority.

TIN-SMELTING IN REVERBERATORY FURNACES.

In this process the ore, mixed with some reducing agent, is spread on the bed of a reverberatory furnace and strongly heated; the products are impure tin, which needs refining, and slags of various kinds which generally contain more or less silicate of tin and unreduced oxide of tin, together with mechanically intermixed prills of metal; the slags have to be cleaned by mechanical or physical treatment combined with a special slag-smelting. Two methods of reduction may be distinguished, namely true reduction by means of carbon or some carbonaceous substance, and precipitation by means of scrap iron; in the case of slags the former method is inoperative without the simultaneous addition of a strong base, generally lime.

As already stated, the reverberatory furnace was first employed for the reduction of tin ore in Cornwall, whence its use has spread to all parts of the world, so that all existing methods are, strictly speaking, only variations of the original Cornish method. This was, however, a pure carbon reduction process, the iron precipitation process being a very recent innovation. As typical of primitive reverberatory furnace practice, it may be as well to quote Pryce's* description (slightly condensed), of the original process, as carried on in Cornwall over a century ago, and then to investigate the various modifications it has undergone in recent times. Pryce gives no drawings or description of the furnace beyond saying that it differs little from that used in smelting copper except that

* *Op. cit.*, pp. 273–282.

it is not quite so deep; he figures the copper-smelting furnace and gives its outside dimensions as 18 ft. long, 12 ft. broad, and 9½ ft. high, and as the inside dimensions of the bed, length 7 ft. 10 in., breadth 4 ft. 8 in., and average height 2 ft., and for the fireplace 2 ft. 8 in. long and 2 ft. wide. He proceeds to describe the process as follows:

"The charge for one of these furnaces is from five to six hundredweight of Black Tin, well mixed with a tenth or a twelfth its weight of culm (*i.e.*, pounded anthracite). This furnace is charged through a hole in the side (directly opposite to the tap-hole), through which it is thrown into the furnace with a shovel, and leveled over the bottom with an iron rake or paddle from the mouth. This done, the apertures are immediately closed, and the fire raised to a very great strength, in which state it is left between four and five hours, when the door is taken off, and the whole charge is well stirred together. The foreman of the work at this time examines the state of the Metal, etc., and, if he thinks it convenient, orders an additional quantity of culm, at his discretion, to be put into the furnace, which is closed again and left in this condition, the fire all the time being kept fully up, till the end of about six hours from its receiving the charge; at which time it is again examined by the foreman, and, if he finds it proper, is then tapped and the Metal let out into a fixed bason made of clay, and of a capacity to hold something more than the Metal of the charge: as in some sorts of Tin, the scoria being vitrified to a considerable degree, part thereof will flow out with the Metal; but this is not commonly the case in any large quantity. The scoria remaining in the bottom of the furnace is raked out at the mouth, and falls into a small pit under it made for the purpose, and has generally adhesion enough to form into a cake. As soon as it is cold it is carried to the stamping mill in order to separate the globules of melted Tin (called 'pillion') disseminated through the scoria or slag. Of the pillion so separated, all the rough or grainy parts are considered as Metal, and refined accordingly by being smelted without any flux, and the produce of this smelting refined with the Tin first tapped. The Tin in the bason, or float, as it is called, as soon as it is come down to a moderate heat, is ladled out into the molds, in slabs or pigs, of about three-quarters of an hundredweight; not larger, because they would be too unwieldy to heave into the furnace for refining. The furnace having, by the side of the small float just now described, a larger one capable of holding twenty or more blocks, is for this purpose suffered to cool to a certain degree, and then charged full with the slabs just mentioned, the tap-hole being kept open, so that as the Tin melts in this moderate fire, it makes its exit through it into the float, where while running out it is frequently stirred and tossed by a ladleful at a time held arm high, letting it fall in a stream into the mass of Metal, when the scum which arises is taken off. While the Metal already put into the furnace is melting, more is added, so as to be just enough to fill the float with good Tin; and this, after being tossed and skimmed as before, and suffered to cool to a proper temper, is carried in Iron ladles to molds holding generally somewhat above three hundredweight (then denominated Block Tin).

"There yet remains in the furnace the drossy part with which the Tin was contaminated, and which, not melting with the slow fire made use of, holds with it a considerable portion of good Tin. The fire is, therefore, now increased, so as

to melt the whole; which is then tapped out altogether into the small float, where the Tin subsiding, and the dross rising to the top, the latter, soon cooling, is taken off and set by, and the Tin ladled into small slabs as at first to be again refined. The furnace is now charged again as before; and after cleansing again is generally employed to smelt Tin Ore as usual. The Tin that remains in and about the scoria and dross of the last tappings, etc., is recovered by repeated smeltings, till at last, being almost entirely drained of that Metal, they become what the workmen generally call Hard-heads, consisting of such heterogeneous metals as were included in the first mixture, and esteemed of no further value."

This is a very minute description of this process of tin-smelting and includes all the essential points; in its general outlines it might stand for a description of the operation as practiced to-day. Pryce gives no idea of the cost of smelting, except incidentally. He describes the method of purchase of tin ore as being by promissory notes or warrants for the amount of white tin that the parcel is calculated to produce by assay less the smelting and incidental charges and the smelters profit, and in an instance he puts the produce of 20 cwt. of tin ore at 12 cwt. of white tin and reckons that the smelter would give a promissory note or "tin bill" for 11 cwt. In other words, the smelter would charge the value of 1 cwt. of tin, or about £3 10s., for smelting one ton of tin ore. It is worth mentioning that the method of assay here referred to is the Cornish method, still in use to-day in Cornwall, of heating the tin ore mixed with one-fifth its weight of culm in a black-lead crucible to a full white heat for 20 minutes. It is of course well known now that this method is decidedly inaccurate and generally gives results several per cent. below the true percentage of metal in the ore.*

A Cornish tin-smelting furnace of the most modern† type is shown in Figs. 18 to 21 in full detail. The tools used in working it are represented in Fig. 22, and the various molds, into which the tin is cast, in Fig. 23. The dimensions vary somewhat at different works, but may be averaged as follows: Bed, 16 to 18 ft. long, 8 to 12 ft. wide; fire-bridge, 6 ft. long by 2 ft. wide, 3 ft. deep below the fire-arch, and 15 in. below the fire-bridge. The bed is built over a hollow vault, and together with the fire-bridge, which is also hollow, is cooled by allowing air to circulate underneath it, air-holes being provided in the brickwork of the sides. The bed is carried upon transverse iron bars, upon which rest slabs of slate, or more rarely fireclay tiles; these are covered by a bed of clay from 6 to 9 in. deep, upon which rests the bed proper, consisting of good sound firebrick, laid on end, close together, and well grouted in. Such a bed lasts in ordinary working for about three months and then requires renewal, an operation that takes about two days. Below the bed, just about the ground level, is a floor of firebrick laid on their broad sides, sloping toward an iron pan, into which any tin that may have leaked through the bed proper trickles, and from which it can be ladled out from time to time. The bed itself is shallow, having a "dish" of about 6 in., and sloping from every side toward the discharge or tap-hole. In front of the tap-hole is the cast-iron float, and close to the latter is a second larger float or

* The dry assay of tin ore by various methods has been critically examined in an admirable paper by H. O. Hofman, *Trans. Am. Inst. Min. Eng.*, XVIII., 1889, p. 3.

† I am indebted to Mr. W. H. Borlase, of Penrith, for much valuable information concerning the modern Cornish process.—H. L.

refining kettle set over a small independent fireplace provided with a small flue. Opposite to the tap-hole is the charging or feeding-door that can be tightly closed by a firebrick quarry, and at the flue-end of the furnace is the working-door, or viewing-door, provided usually with an iron roller, to facilitate the use of the rakes, rabbles, and other tools, and also capable of being closed with a quarry. The fire-door is opposite to the last-named door and is placed pretty high up;

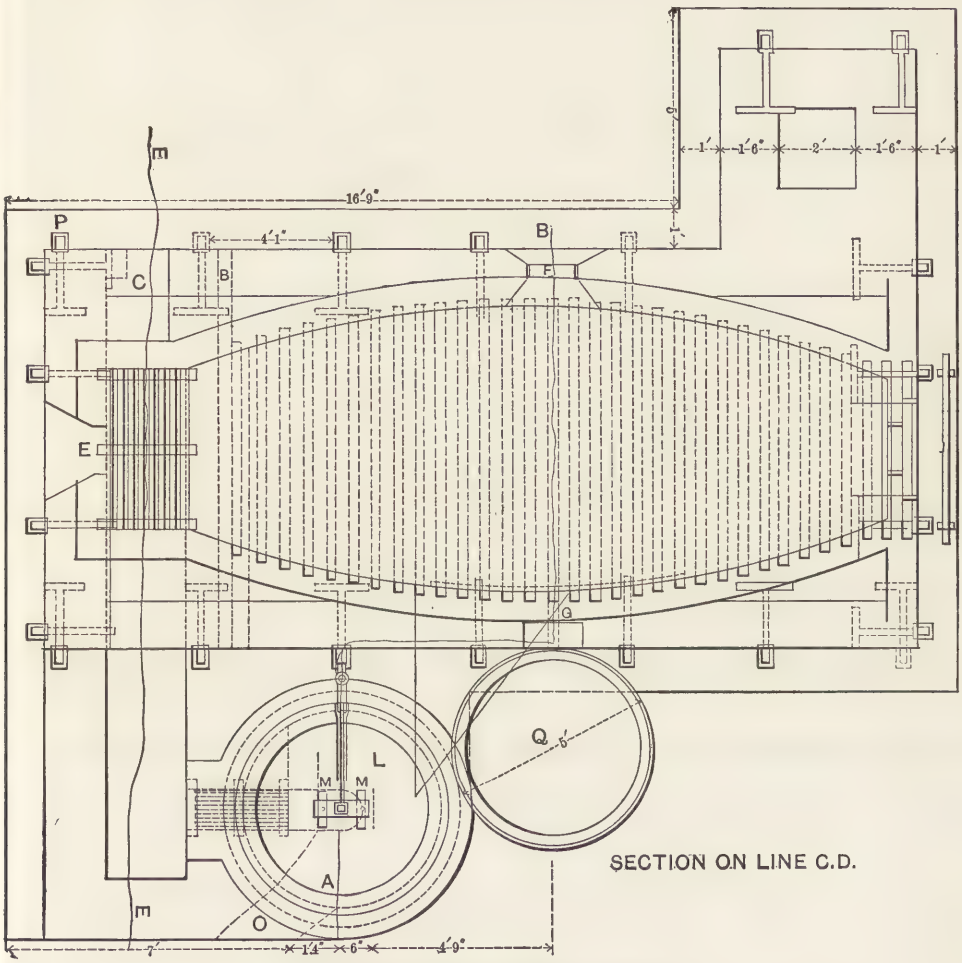


FIG. 18.—MODERN CORNISH FURNACE.

this is only an opening, not a door properly speaking, and is kept closed by the "green" coal while the furnace is at work. A clinker-bed is used, and the fire-place is kept full of coal almost up to the level of the fire-bridge. In some of the older forms there was another door in the spring of the fire-arch, kept always closed by means of a firebrick slab, except during charging, when it was opened so as to diminish the draft over the bed and thus prevent fine ore from being carried off; the same object is now attained by closing the damper during charg-

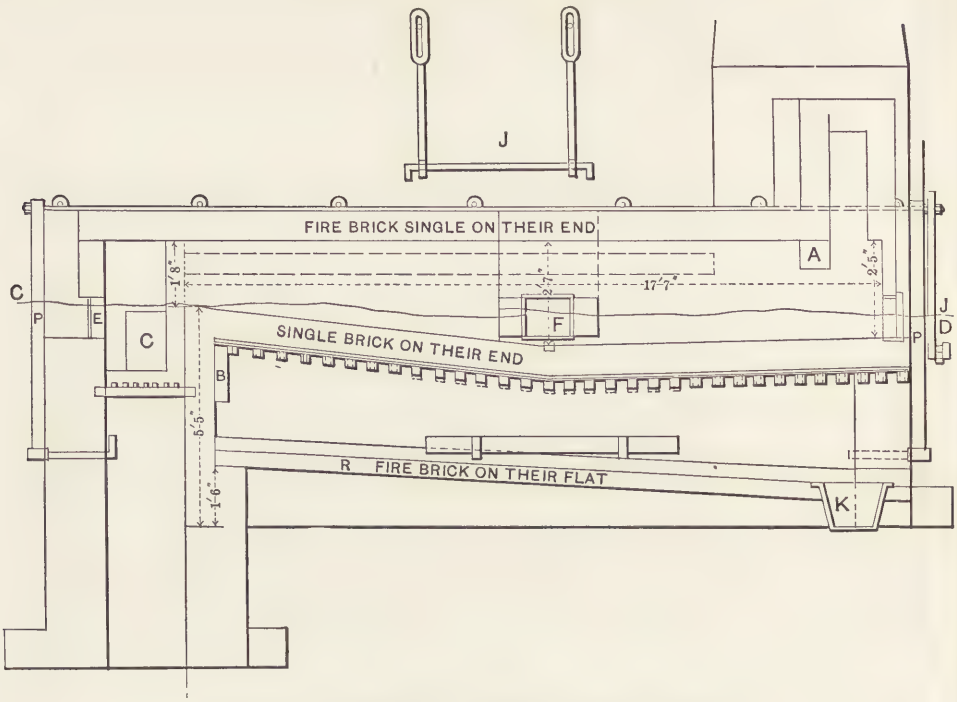


FIG. 19.—LONGITUDINAL SECTION ON CENTER LINE.

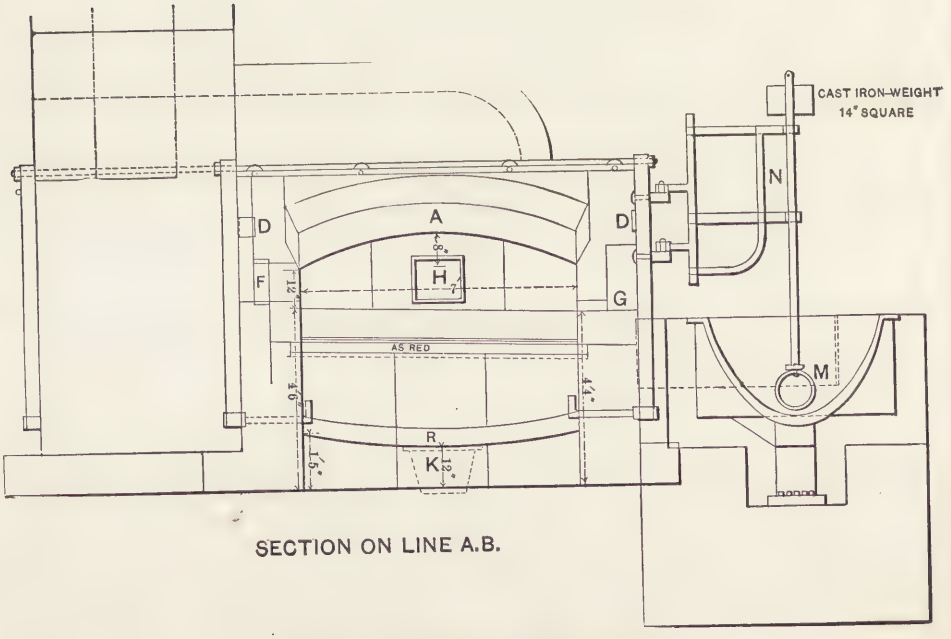


FIG. 20.—MODERN CORNISH FURNACE.

ing. Each furnace has its own stack, about 50 ft. high, fitted with a damper; dust chambers are not used. The furnace is held together by iron plates, buck-staves and tie-bars.

The cost of such a furnace is about £160, of which £90 are for bricks, mortar, iron work, etc., and £70 for labor. The cost of the chimney and flues, everything included, is about £250.

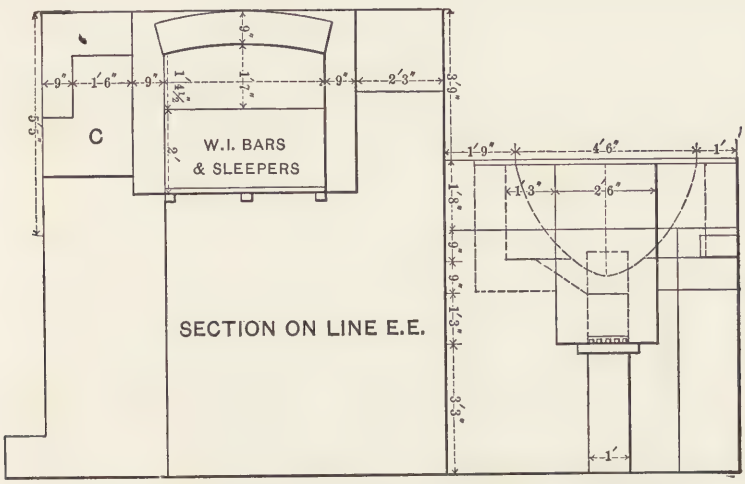


FIG. 21.—MODERN CORNISH FURNACE; SECTION ON LINE EE.

EXPLANATION OF DRAWINGS OF CORNISH FURNACE.—FIGS. 18 TO 21.

A—Arch of firebrick built on their edge, *i. e.*, two rows of 4½ in. brick work.
B—Air hole to keep bottom of furnace cool.
C—Air hole for cooling outside of fireplace.
D—Wrought-iron plate, 6 x ½ in., built at each side of furnace for strengthening arch.
E—Opening to fireplace.
F—Charging or feeding door; to be stopped with brick slab the same as *H*.
G—Discharge hole.
H—Working door.
I—Fireplace of refining kettle.
J—Wrought-iron frame used for resting rake on when working through *H*.
K—Pan to catch any metal that may leak through the furnace.
L—Refining kettle (cast iron).
M—Wrought-iron hoops through which is held a piece of fir-wood, and sunk in the molten metal by the cast weight on *N*.
N—Wrought-iron hinged crane; the perpendicular bar and weight pulled up by a small pulley block high enough for hoops to pass from kettle *L*.
O—Flue from refining-kettle fire to pass off in any convenient place and to any ordinary house chimney.
P—Cast-iron bars for binding furnace, held at the bottom by strong wrought-iron eyes built in the masonry.
Q—Wrought-iron float lined with fireclay to take metal from discharge hole.
R—Bed built of single brick on their flat to catch any metal that may leak through furnace.

The charge for these furnaces varies in different works from 2700 lbs. to 3 tons, 2 tons being about the average. In the most important Cornish works at Penzance, 2-ton charges are smelted, four charges being worked off in 24 hours; at another works 3-ton charges, of which three can be got out in 24 hours, are preferred.

The ores treated are chiefly Cornish and Bolivian. The former, after being

properly dressed, generally contain 62 to 65% of metal (equal to 79 to 82% SnO_2), the remainder consisting of silicious matter, 5 or 6% of oxide of iron and small amounts of tungstic acid and other impurities. Bolivian ores are generally richer, containing up to 72% of metal (92% of SnO_2), 5 to 6% of silicious matter, and 1 to 3% of oxide of iron, etc.

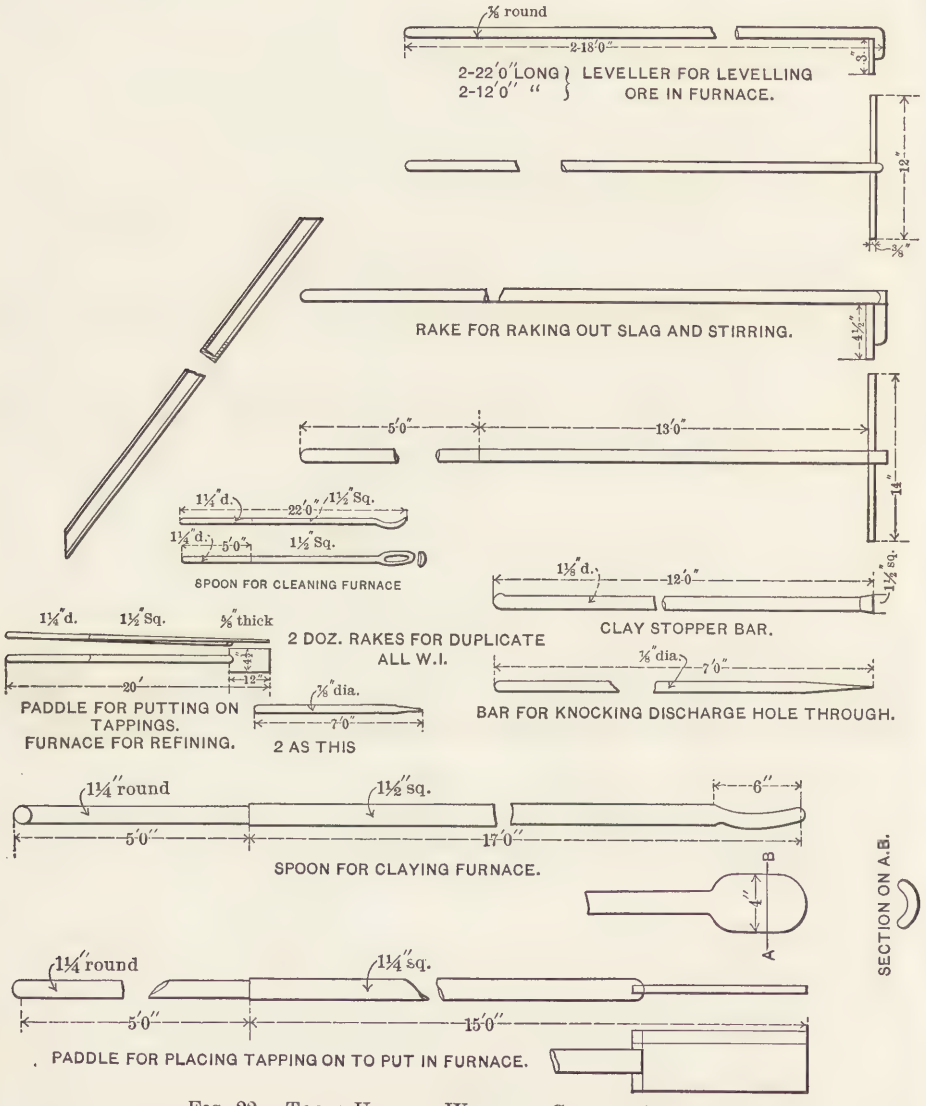


FIG. 22.—TOOLS USED IN WORKING CORNISH FURNACE.

The charge consists of ore mixed with 15 to 20% of culm—spoken of, though incorrectly, as "flux" by the smelters—together with a small amount of slaked lime in varying proportion according to the quality of the ore, occasionally some slag or other tin-bearing products from previous operations, and at times a little fluorspar. These substances are thoroughly mixed and well damped with water.

The charge is wheeled to the furnace in barrows, and tipped on the ground in front of the charging-door. The furnace being all clean after the previous charge has been got out, any weak spots in the bed having been clayed up, and a good heat raised, the charge is shovelled in and spread smoothly over the bed by means of a rabble worked from the end-door. Both doors are then closed tightly and may be luted with fireclay, and a full fire is put on. After from one to three hours the mass should be fairly well melted; it is then rabbled through, the fire being kept up all the time, and rabbling is repeated at intervals as judged necessary. In from 5 to 7 hours the operation is usually completed, the temperature being about the melting point of cast iron. The mass is again well rabbled through and allowed to settle, so that the metal may separate as completely as possible from the slag. The slag is then drawn out through the working-door, by means

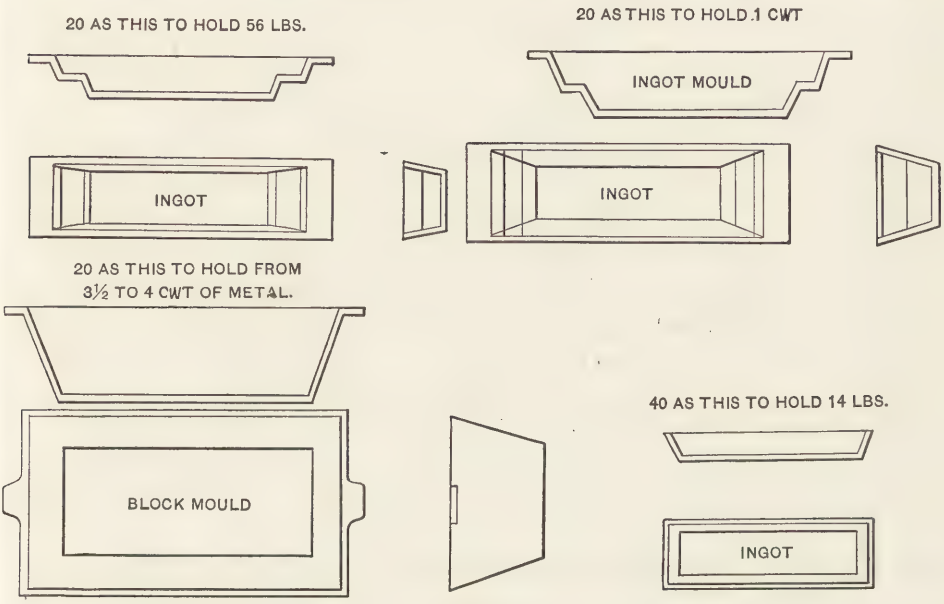
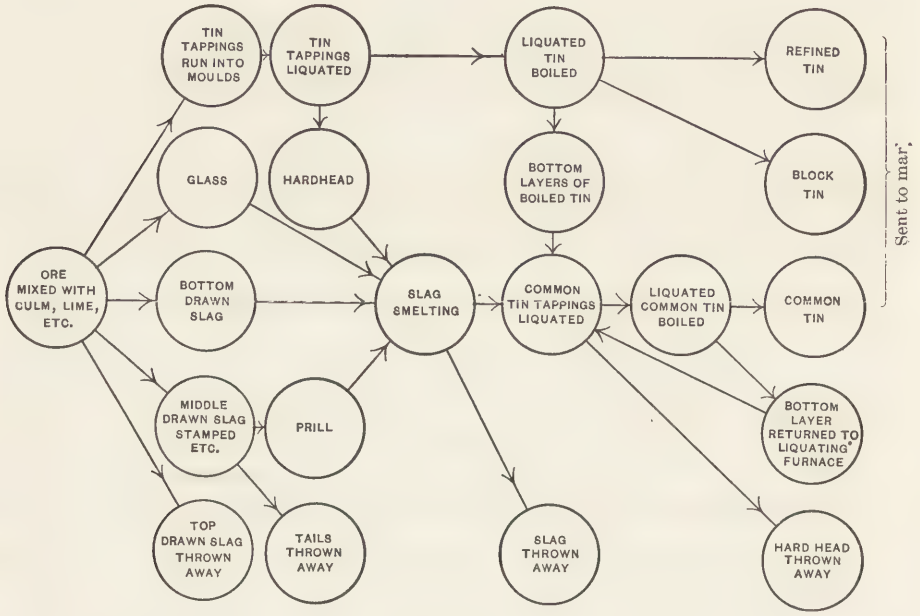


FIG. 23.—BLOCK MOLD AND INGOT MOLDS USED IN CORNWALL.

of rakes or rabbles, and if it appears too thin it is thickened by a few shovelfuls of culm; it is known as "pulled slag." The top layer of slag that is first drawn off (about two-thirds of the whole) is generally considered clean and not to be worth further treatment. The next lot contains a good deal of tin as shot or prill, and is sent to the stamps to be crushed and washed, old furnace beds or other residues being treated in the same way. The remainder of the slag requires to be resmelted together with the prill obtained by the treatment of the previous lot. Not all the slag is obtained in the form of pulled slag; a certain amount remains on the metal in a molten condition and runs out with it into the float when the furnace is tapped; it rises to the top and forms a layer on the tin in the float, and is removed as soon as it has set. This slag is known as "glass" and has also to be resmelted. The tap-hole is always closed during the smelting operation by means of an iron bar of about $1\frac{1}{8}$ in. diameter, known as the "clay stopper bar," which is

clayed into its place. After as much slag as possible has been drawn out, this bar is pulled out and the tin runs into the float together with a certain amount of glass. In order to clear out the furnace, the whole of the clay stopping is finally cut out by means of a pointed bar. After the glass has been removed, the tin is allowed to stand for some hours; any dross that forms on the surface is skimmed off and put aside for retreatment, and the metal is ladled into molds holding about 100 lbs., the flat slabs so produced, known as "tappings," being then ready for refining.

The subjoined scheme shows the general course of smelting operations when ores of fairly good quality are being worked, but variations from it are very often made.

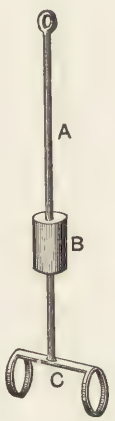


At Penzance four furnaces are worked by 16 men all told, there being eight furnacemen who work 12-hour shifts. There are only the four furnacemen and one helper or tender on the night shift; two charges are worked every shift. The consumption of coal in smelting is about one ton of coal to the ton of ore.

Refining is carried out either in a smelting furnace in some of the smaller works, or else in a special refining furnace, with a flat bed sloping uniformly toward the tap-hole; outside the latter stands a large cast-iron float that can be heated by means of a small fire-grate below it. The slabs of tin are charged by means of a peal, or paddle (see Fig. 22), and are laid close together on the upper parts of the bed and a small fire is put on, so that the tin melts at a gentle heat and trickles through the open tap-hole into the float, leaving the more infusible impurities on the bed; these are drawn aside and fresh slabs are charged in, until the float, which holds from 6 to 10 tons, is full. The tap-hole is then closed and the fire raised until the residue left in the furnace is melted; this is run off into

another float where it is allowed to settle and after a time separates into two portions, the upper and more fusible being an impure tin which is ladled out into molds. This tin requires special refining. The residual hard-head which generally adheres to the sides and bottom of the float is then removed; it is rarely considered worth further treatment and may be added to the charge in slag-melting or may be sold.

Meanwhile the molten tin in the float having been allowed to settle, is skimmed, and then boiled; the latter operation is performed by forcing into the metal a number of billets of green wood held in a cage and weighted down so as to force the wood well below the surface of the molten metal. The arrangement of one of the most usual forms is shown in Fig. 24; *A* is an iron shank 6 to 10 ft. long, upon which is fixed the cast-iron weight *B;* billets of green wood, which may not, however, be wet, are placed in the two hoops composing the cage *C*, and the whole apparatus is suspended by a rope or chain running over a pulley above the float, so that it can be lowered when required, the weight *B* forcing the wood



A, SHANK; B, WEIGHT; C, CAGE. APPROXIMATE SCALE 1″=6′.

FIG. 24.—PROSPECTIVE VIEW OF BOIL CAGE.

down to the bottom of the float; another form is shown attached to the furnace in Fig. 20. A dense wood, like apple-wood, which evolves gases at a comparatively slow rate, is generally preferred, though fir-wood is at times used. The chemistry of the process is not yet completely understood. The wood is slowly distilled by the action of the molten metal, and the gases evolved, together with the steam given off, keep the bath in a state of violent agitation, throwing up spurts of tin that fall back into the bath; a large surface is thus constantly exposed to the air, and it seems probable that the oxygen of the latter, by oxidizing the impurities present, plays the chief part in the operation of refining by boiling. A scum forms on the surface of the tin, which seems to consist of oxides of the various metals present, together with a certain amount of finely divided metal; it is skimmed off and subsequently resmelted with slags, etc. The operation of boiling takes about 3 hours, liquating occupying 1 to 2 hours. After the boiling is finished the tin is allowed to stand at rest for an hour or two, in order that all the heavier impurities still present may settle to the bottom of the float. The

tin is then ladled into molds; the uppermost or purest portion is known as refined tin, the next, which is still sufficiently pure for many commercial purposes, is called block tin, and the remainder, which is not marketable, has to be refined over again.

Refined tin is usually cast in ingots varying from 7 to 100 lbs. in weight, or sometimes into bars or strips of from 3 to 5 to the pound. Block tin, which is largely used in the manufacture of tin-plate, is cast in big blocks weighing 350 to 400 lbs. The various forms of molds used are shown in Fig. 23.

Tossing is sometimes substituted for boiling, the metal being dipped out in ladles holding about 30 lbs. and poured back into the bath from a height of 2 or 3 ft. It is hard work for the men, and appears to present no advantage over boiling.

It must not be forgotten that the refining of tin is at the best an imperfect operation, and that large quantities of impurities cannot be separated by it. The quality of the tin ultimately produced depends therefore very largely upon the purity of the ore charged into the smelting furnace, and it is therefore imperative that any impurities present originally in the ore should be removed as completely as possible in the preliminary operations of roasting, dressing, etc., so that the smelter may receive the ore in the highest practicable state of purity.

The following analyses show the average composition of English tin:*

	Per Cent.	Per Cent.	Per Cent.
Sn	99.76	98.64	99.73
Fe	Trace.	Trace.	0.13
Pb	.....	0.20	.....
Cu	0.24	1.16	Trace.
Totals	100.00	100.00	99.86

Analyses of the various slags are not obtainable. Berthier found 13.02% of tin in a slag free from prill, that was thrown away. He also gives the following analysis of a slag produced at Poullaouen in France in reverberatory furnace smelting: SiO_2, 40.0%; SnO_2, 8.4%; FeO, 20.3%; MnO, 11.1%; CaO, 3.6%; MgO, 1.0%; Al_2O_3, 9.6%—total, 94%.

It is generally considered that slag with about 5% of tin is no longer smeltable with profit in Cornwall. At present a good deal of Cornish tin slag is shipped to Wales and there sold; it is said to be treated by being smelted (in cupola furnaces) with oxidized lead residues and residues from tin-plate works, the result being an alloy of lead and tin that is marketable as such, and a slag practically free from tin.

It is usually stated that the loss in the Cornish method of smelting is about 9%, a certain amount of which is said to be recovered when the furnaces are taken down for repairs, because tin is so readily fusible, and so mobile when melted, that it readily percolates through any cracks or joints in the beds and finds its way into the foundations of the furnace. As a matter of fact, however, no one except the smelters themselves, and possibly not even they, know accurately what the losses in tin-smelting are. Much of the difficulty arises from

* Schnabel, *Metallhüttenkunde*, II., p. 432.

the barbarous method still employed for buying and selling the ores. At the fortnightly ticketings the buyers obtain samples of the various lots of ore for sale, and assay them, and then send in their bids at so much per unit, *i.e.*, the price they give nominally per ton for each per cent. of gross tin in the ore, the highest bidder of course securing the parcel. Unfortunately, however, both assays and weights are misleading; the assay is the old inaccurate Cornish assay and the weights used are not statute weights, but the ton consists of 20 cwt. of 115 lbs. each, while "draftage" is allowed at the rate of 1 lb. in each cwt., 2 lbs. in 7 cwt., 3 lbs. in 8 cwt., and so on, and moisture is allowed for at the rate of $1\frac{5}{8}$ cwt. to the ton of ore, or $8\frac{1}{8}\%$; besides this the smelter deducts an amount that he considers will cover the cost of smelting; he thus gets a great deal more tin apparently than he pays for on the face of the figures returned. It would be difficult to say much in defense of this system of dealing; the usual Cornish argument that this was the method practiced by their ancestors is hardly satisfactory. No doubt an accurate assay showing the real tin contents of the ore would be better in the interests of both buyer and seller, and would go far to remove any suspicion that one party is obtaining an unfair advantage over the other. Moreover correct assays would necessarily force upon the smelters a recognition of the amount of tin that they are really losing, and this, the inevitable first step toward a general improvement of the process of treatment would assuredly benefit the entire tin trade.

Nothing is accurately known as to the cost of smelting; the consumption of fuel, excluding the culm used for reduction, is usually given as 30 to 35 cwt. of coal to the ton of tin. In 1895* the total amount of tin ore produced in Great Britain was 10,612 tons, of an average assay value of about 60 to 65%; the price it brought at the mines was £370,530; this ore is estimated to produce 6648 tons of metal, worth at block-tin price in London £446,780, so that the difference of £76,250, or over £7 per ton of ore, represents the cost of smelting plus smelters' profits less a small amount for carriage to London and agents' commission. In addition to British ore, a certain amount of foreign ore is smelted; the imports in 1895 were 4705 tons, of which by far the largest part was Bolivian ore; of this 3830 tons were imported direct, while it is probable that nearly the whole of the ore imported from Germany and other European ports, amounting to about another 500 tons, came originally from the same source. All the Cornish and imported ores were smelted by seven firms of smelters in the south of England, the works of five being situated in Cornwall, one in Devonshire, and one at Bristol.

In addition to these, smelting operations have recently been commenced near St. Helens, Lancashire, where Bolivian ores are being treated. The process is practically the Cornish process, but many details have been greatly improved. The smelting and refining furnaces are shown in Figs. 25 and 26; the former is an ordinary Cornish furnace, a good example of the usual though not the most modern type, but the latter is essentially novel in design, consisting of two cast-iron trays inclined in opposite directions, each toward a float of its own, set over a small fire, on opposite sides of the refining furnace. This arrangement is said to produce a marked economy of fuel consumption.

On the continent of Europe the Cornish process is only in operation at one

* *Mineral Statistics of the United Kingdom for 1895*, p. 163.

works at the present time, namely, at those of Messrs. Robinson & Bense, at Tostedt, near Hamburg, which employ some 45 hands. The ores treated are chiefly Bolivian, but tin residues of all kinds are also purchased and smelted. Two* reverberatory furnaces are in use, a large and a small one; the charge for the former is 3 to 4 tons, and about 6½ tons are smelted in 24 hours with a consumption of 6400 lbs. of coal, there being three men on a shift. In the smaller furnace the charge consists of 3000 to 3600 lbs. of ore; 7200 lbs. are smelted in 24 hours with a coal consumption of 4800 lbs., there being two men on each shift. The slags produced in this ore-smelting contain 20 to 25% of tin, and they are resmelted in the same furnaces, the smaller furnace smelting 9000 lbs. of slag in 24 hours with a consumption of 4400 lbs. of fuel, and the larger 18,000 lbs. of slag with a consumption of coal amounting to 6000 lbs. Although not so stated, it seems probable that the slag-smelting is an iron-reduction process, as in the Singapore process to be presently described; the owners of the Tostedt works are, however, averse to giving information on such points. The slags obtained in the second fusion are resmelted two or three times more in shaft furnaces. These works have the reputation of producing remarkably clean slags. The effect of their operations may be gathered from the following table of the tin production of Germany, the amounts being given in metric tons:

Year.	1890.	1891.	1892.	1893.	1894.	1895.
Production of tin in all Germany	64	287	684	951	896	884
Production of tin in Saxony	64	49	40	42	51	35
Difference	.....	238	644	909	845	849

The only tin-smelting works in Germany are those in Saxony and at Tostedt; the latter started in 1891, and it may be taken for granted that the figures in the column above, headed "difference," represent accurately the output of tin from these works. Hamburg is the chief importing port of Germany, and most of the tin ore and crude tin sent to that country passes through it; no figures as to the import of tin ore are, however, available.

Tin ores occur at various places in Brittany and have been worked there, but never with any great measure of success. Smelting works have been erected at Piriac, near the mouth of the Loire, and at La Villeder, Morbihan, reverberatory furnaces having been used in both cases, but neither are in operation now. La Villeder was worked some two years ago by an Anglo-French company, which erected dressing and smelting plant at a cost of 80,000 to 100,000f. ($16,000 to $20,000); the whole concern was sold about six months ago for 25,000f. ($5,000). The furnace used had a bed about 11 ft. long by 7 ft. wide; the charge consisted of 1½ tons of ore and 600 lbs. of culm, and in 24 hours 6 tons of ore were smelted with a consumption of 9 tons of fuel (coal). There is still a small quantity of tin ore (about 120 tons in 1894) imported into France and apparently worked up there, but official statistics give no return of any tin-smelting works in the country.

Spain and Portugal have produced tin ores in small quantities for many years

*Schnabel, *Metallhüttenkunde*, Vol. II., p. 414.

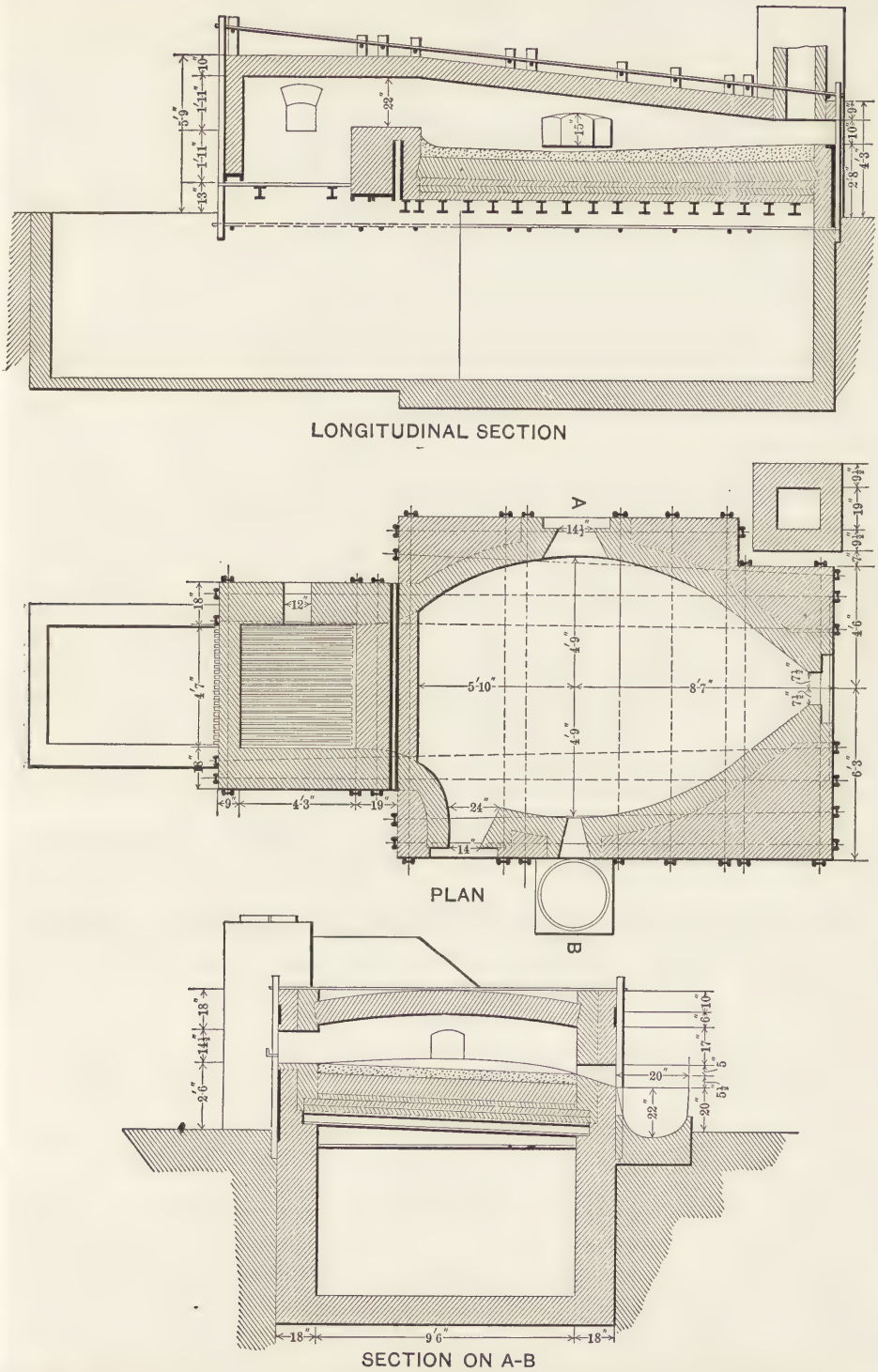


FIG. 25.—TIN SMELTING FURNACE BUILT FOR THE ALMARAZ TIN MINING COMPANY, SPAIN.

but have always exported them raw. Dressing and smelting works are now, however, in process of erection for the Almaraz Tin Mining and Smelting Company, Limited, an English company operating mines at Almaraz, Zamora. The works are being constructed by Messrs. Bowes, Scott & Western, Limited, of London, who are also building the necessary furnaces. The Cornish method of smelting is being adopted, the furnaces being modeled upon the types of those in use in Cornwall (Figs. 18 to 21) and in Lancashire (Figs. 25 and 26). It is also contemplated, should these mines turn out well, to commence operations at other points in the Provinces of Coruña and Pontevedra, so that there seems to be a possibility of Spain becoming a tin producer in the not very distant future.

In all the Australian colonies in which tin ore is found the Cornish method, or a variation of it, is the smelting process employed. The chief tin-smelting districts are Pyrmont in Sydney, Tent Hill on the Vegetable Creek tin field, where the Glen Smelting Company has works, both in New South Wales, Irvinebank in Queensland, and Launceston in Tasmania, where the tin ore from the Mount Bischoff mines is smelted. Tin ore exists in Western Australia, and the government has offered a bounty of £1000 ($5000) for the first tin works started, but none has yet been erected. Numerous smaller works exist or have existed in various places, but the above are the most important. The methods of smelting are, generally speaking, similar, and constitute an improved Cornish process; thus while the maximum charge of the Cornish furnace was for a long time only 27 cwt., in Australia 3-ton furnaces were soon built, and an iron-precipitation process for slag-smelting was also introduced. As it was found difficult to obtain anthracite for mixing with the ore in Australia, the culm used in Cornwall was replaced by ordinary bituminous coal. At Tent Hill and Irvinebank, both of which places lie inland, and are not near any coal field, the use of wood as fuel and of charcoal as a reducing agent was introduced, very good results being thus obtained.

The Vegetable Creek tin field produces annually somewhat over 1000 tons, having fallen off a good deal in recent years; practically the whole of this ore is smelted at the Tent Hill works. These comprise three reverberatory furnaces,* with beds 14 to 16 ft. long and 6 to 8 ft. wide; the fire-bridge is 1 ft. high, the height of the arch being 2 ft. The stacks are 50 to 60 ft. high. The furnace bottoms are carried on iron bars so disposed as to be readily knocked out, when required, for repairs. The furnace charge consists of 3 tons of dressed ore and 1 ton of charcoal moistened with water and spread over the bed to a depth of about 12 in. in the center. The composition of the ore varies according to whether it is stream tin or lode tin. The following is an analysis† of a tin stone of the former class from the Jupiter Mine, Vegetable Creek: SnO_2, 89.92%; TiO_2, 0.69%; Al_2O_3, 6.75%; SiO_2, 0.80%; Fe_2O_3, 2.30%—total, 100.46%.

The furnace, as previously stated, is fired with wood; the charge is rabbled through occasionally and at the end of 12 hours is tapped in the usual way into a large brick-lined float. The metal so produced is comparatively pure; it is ladled into a 5-ton refining kettle, where it is allowed to stand for some two hours; the upper portion is ladled into a smaller kettle, where it is boiled by the

*T. W. Edgworth David: *The Geology of the Vegetable Creek Tin-Mining Field*, 1887, p. 154.
†A. Liversidge: *The Minerals of New South Wales*, 1888, p. 82.

immersion of billets of green "stringy bark" for four hours. It is then generally soft enough for trade purposes and is cast into 50-lb. ingots; these contain 99.5 to 99.8% of pure tin. The slags and other residual products are smelted separately. The Sydney Smelting Company, the property of Mr. T. H. Kelly,

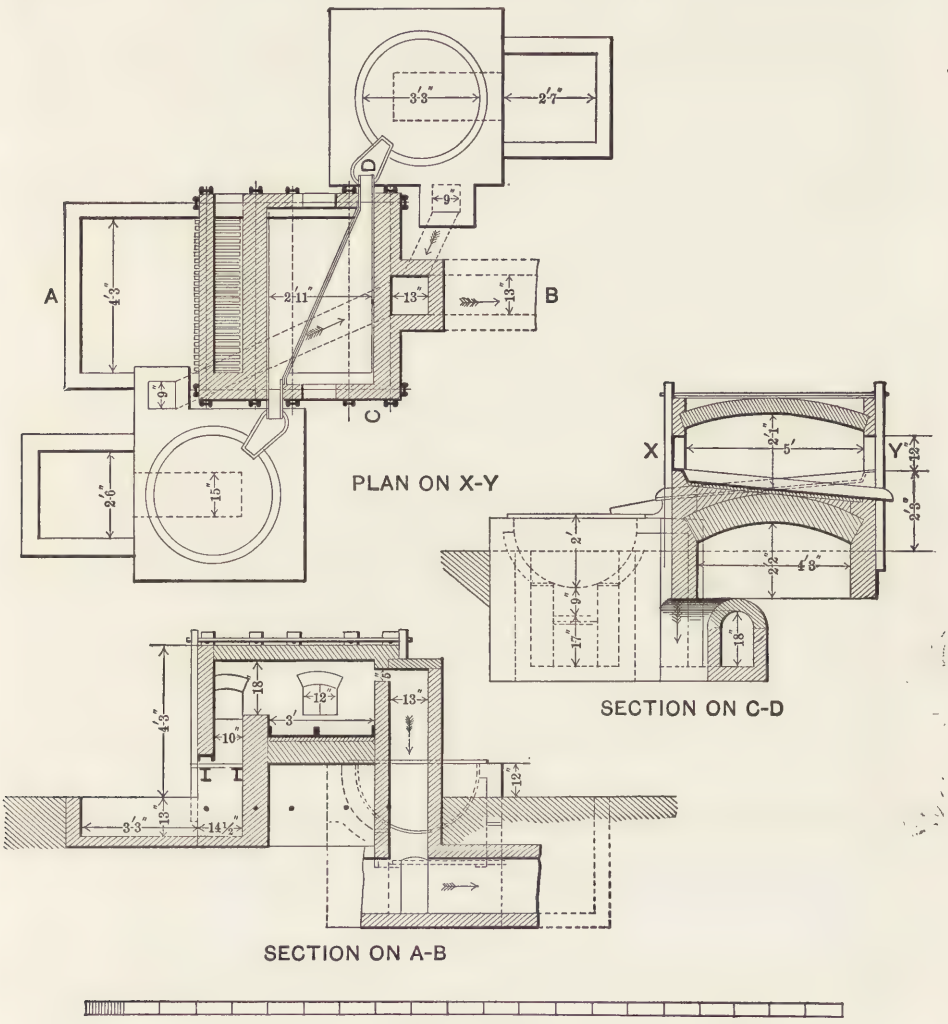


FIG. 26.—TIN REFINING FURNACE.

refuses to give any information respecting the method of tin-smelting there practiced.

The Mount Bischoff Tin Mining Company owns a smelting works at Launceston, Tasmania, where not only the ores raised at the Mount Bischoff mines, but most of the tin ores of the colony are smelted. The smelting plant consists of six small reverberatory furnaces, local coal being the fuel employed. Mount

Bischoff ore as it comes from the mine carries 3 to 3½% of black tin, but it is dressed up to a yield of 64 to 72% of metal, gross assay. In the half-year ending June 30, 1896, the results were as follows: Ore smelted on company's account, 1179.9 tons; metallic tin produced, 797.9 tons: ore smelted on public account, 748.75 tons; metallic tin produced, 519.15 tons; total ore smelted, 1928.65 tons; total tin obtained, 1317 tons. Of the Mount Bischoff ore the various parcels were as follows: No. 1 first quality, average net assay 68.98%, 764.45 tons; No. 1 second quality, average net assay 65.71%, 245.15 tons; No. 2 first quality, average net assay 64.26%, 170.3 tons. In the second half of that year the figures were: Ore smelted on company's account, 1136 tons, yielding 772.3 tons of metal; ore smelted on public account, 854.4 tons, producing 596.2 tons of tin; total ore smelted 1990.4 tons, producing a total of 1368.5 tons of tin. The following were the parcels of Mount Bischoff Company's ore: No. 1 first quality, average net assay 69.06%, 809 tons; No. 1 second quality, average net assay 65.57%, 250.75 tons; No. 2 first quality, average net assay 64.53%, 76.2 tons. The net assay of the first grade is 2% less than the gross, and of the second and third qualities 3% less.*

The ore is mixed with a very little lime and with ground New South Wales coal, the process being practically the regular Cornish process. The final slag assays 5.1 to 5.2% of tin and the loss of tin is averaged at about 4.25% of the total present. The following is an average analysis of the final slag: SnO_2, 6.6%; FeO, 19.7%; MnO, 1.2%; SiO_2, 52.3%; Al_2O_3, 14.1%; CaO, 4.6%; MgO, 1.0%—total, 99.5. The tin is refined in the usual way, the refined metal assaying 99.85% in the first half-year in question, and 99.87% in the second. It seems to vary between 99.8 and 99.9%.

With regard to the cost of smelting, the only information that appears in the accounts is an item "Furnace working account—£2311 14s. 8d." in the first half-year, and "£2084 9s. 8d." in the second. If these amounts represent the cost of smelting the above 1317 and 1368.5 tons of tin respectively, it will work out to £1 15s. 1d. and £1 10s. 6d. per ton respectively; obviously interest and depreciation are not included in this figure, and possibly not even management, but even so it gives an idea of what reverberatory-furnace smelting really costs there. The percentage of cost of the various items is distributed as follows: Fuel, 41; labor, 43; fluxes, 1; general, 15—total, 100.†

Australasia now smelts practically the whole of the tin ore it produces, only a very small quantity being shipped to Europe in the form of ore. The importance of the tin-smelting industry may be gauged by the following table of its approximate annual production, all the tin being returned as metal: Tasmania, 2800 tons; New South Wales, 2250 tons; Queensland, 1400 tons; West Australia, 200 tons; Victoria, 50 tons—total, 6700 tons.

Smelting in the reverberatory furnace has of recent years been employed at one works in the Malay Peninsula, or rather in a small island just to the south of the peninsula; the process as there carried out deserves detailed description, first of all because these works are by far the largest in the world, producing more tin than the whole of Cornwall, and secondly because the process is in many

* From the half-yearly reports of the directors of the Mount Bischoff Tin Mining Company, Registered.
† *Sixteenth Annual Report of the Director of the United States Geological Survey*, p. 506.

respects quite novel, and has, thanks to the skill and ability of Mr. J. McKillop, the constructor and for many years the manager of the works, attained a greater degree of scientific and technical perfection than can probably be claimed for any

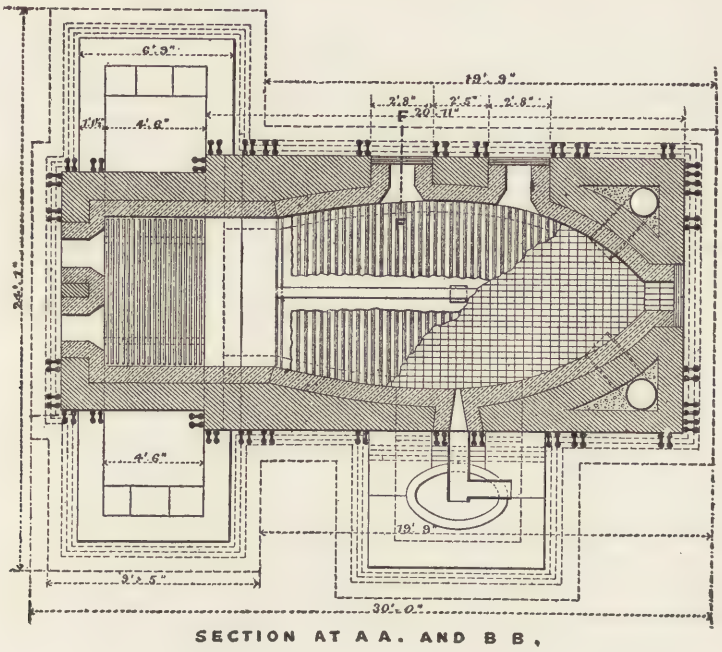


FIG. 27.—TIN SMELTING FURNACE AT PULO BRANI.

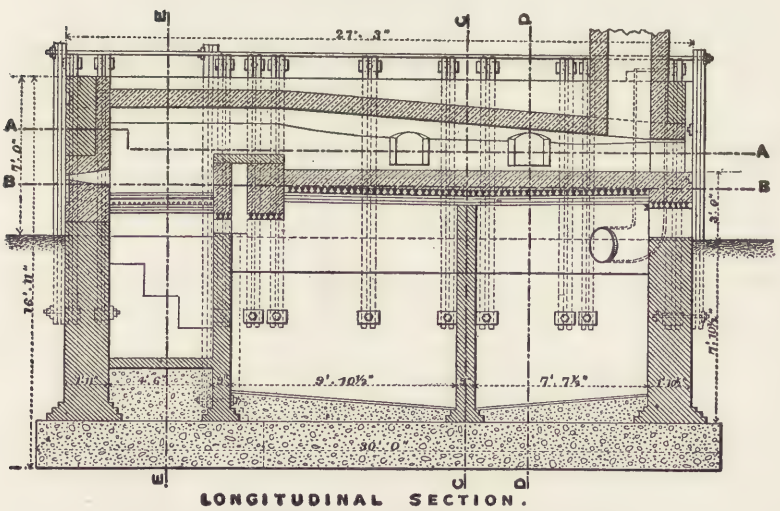


FIG. 28.—TIN SMELTING FURNACE AT PULO BRANI.

other tin-smelting establishment. Much of the following information has been obtained direct from Mr. McKillop, who has also described his process in a paper

communicated to the Institution of Civil Engineers.* The first attempt to smelt tin in reverberatory furnaces in the Malay Peninsula was made in Perak by the Shanghai Tin Mining Company of Perak, which was, however, not successful, and had to be wound up. In 1885 the Straits Trading Company was formed and recommenced operations at the abandoned works of the Shanghai Tin Mining Company at Teluk Anson on the Perak River; the new company succeeded in securing a most valuable monopoly, namely the sole right to purchase and export tin ore from Selangor and Perak, and two years later, at the end of 1887, commenced smelting operations with one furnace on the small island of Pulo Brani, which practically forms a portion of the almost landlocked harbor of Singapore, so that every facility for unloading tin ore and coal, and for shipping the ingots, was readily obtainable. This company has been as successful commercially as it has technically, and to-day has in steady operation fourteen 4-ton smelting furnaces, with a well arranged refinery, producing about 1200 tons of ingot tin per month.

The reverberatory furnaces used for smelting present a good many novelties, the lining of the walls, the roof and the bed being so arranged that each is capable of independent renewal. The outer main walls carry the lining of firebrick, against which the edge of the bed abuts, and the arch of the roof springs also from the main walls and not from the lining, the walls being securely tied against the thrust thus caused by iron buckstaves and tie-rods. The bed is carried on transverse iron rails of a length equal to one-half the width of the bed, one end of which rests on the brickwork of the main walls and the other end on a heavy longitudinal rail; the latter is in two halves, the outer ends carried by the brickwork of the furnace, the two inner ends on a brick pier in the middle of the furnace. When this pier is knocked down, the iron-work on which the bed rests at once falls in, and the bed, of course, with it. The bed is composed of firebrick laid on end as close up as possible, and grouted with fireclay. It is then dried and heated, and when ready a charge of cast iron is melted on it that binds every part firmly together. Every part of the bed slopes toward the tap-hole, the fall amounting to 3½ in. The fire-bridge, which is 8 in. above the bed, is hollow and is also carried on rails; the working dimensions of the bed are: Length 16 ft., width at center 9 ft. 9 in., and at fire-bridge 6 ft., area about 125.5 sq. ft. A bed requires about 3000 firebrick, and lasts for 120 to 150 charges. A special feature is the water-vault; the lower part of the furnace foundation below the ground line is built so as to form a water-tank extending the full length of the bed and bridge, and the full width of the bed and float. This vault is kept full of water to a depth of 8 ft.; any tin that leaks through the bed, or through any joint, drops into the water and is thus granulated; once a week the water is pumped out and the granulated metal collected. Any steam generated escapes through a couple of tubes, 18 in. in diameter, built into the angles of the vault nearest the flue end; and it is found that as long as the water-level in the vault is maintained there is no danger of explosion. As it is absolutely impossible to prevent the leakage of so mobile and easily fusible a metal as tin, this water-vault seems to present the best method of dealing with the leakage and of promptly recovering the metal.

* Vol. CXXV, 1895-96, Part III.

The dimensions of the fireplace vary, according to the kind of coal that is being used, from 4 ft. by 6 ft. to 4 ft. 6 in. by 6 ft. 9 in., the depth being 2 ft. 6 in. from the top of the fire-bridge to the bars; there are two fire-doors in the back wall high up close to the roof; these have no door proper, but are kept sealed by the coal thrown in. There is also a series of holes just above the fire-bar level, through which a bar can be introduced to keep the fire-bars clear and to work the fire as required. The furnace proper has the usual working-door in front under the flue and two charging-doors. The construction of the furnace is shown in Figs. 27 to 29, taken from the above quoted paper.

Coals of various kinds have been used at these works, namely Welsh, Australian, Japanese, Borneo, and Tonquin; now Japanese and Tonquin coals are used almost exclusively, since, although inferior to Welsh coal, they are a great deal

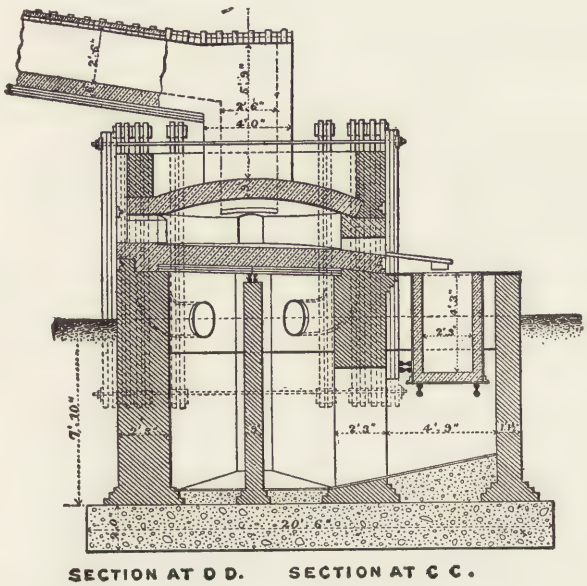


FIG. 29.—TIN SMELTING FURNACE AT PULO BRANI.

cheaper. Welsh anthracite is imported, however, for mixing with the charges. Soft coal has been used, but is not quite as satisfactory, besides 10% more being required. The area of the flues is about 3.75 sq. ft. and the stacks, one to each pair of furnaces, are 100 ft. in height.

The cost of a furnace complete with overhead flues, but without stack and including also one-half of the old form of refining pot and ingot stand (used when refining was done in these furnaces and when one kettle was arranged between each pair of furnaces) may be taken at about £350 (exchange taken at 2s. to the dollar).

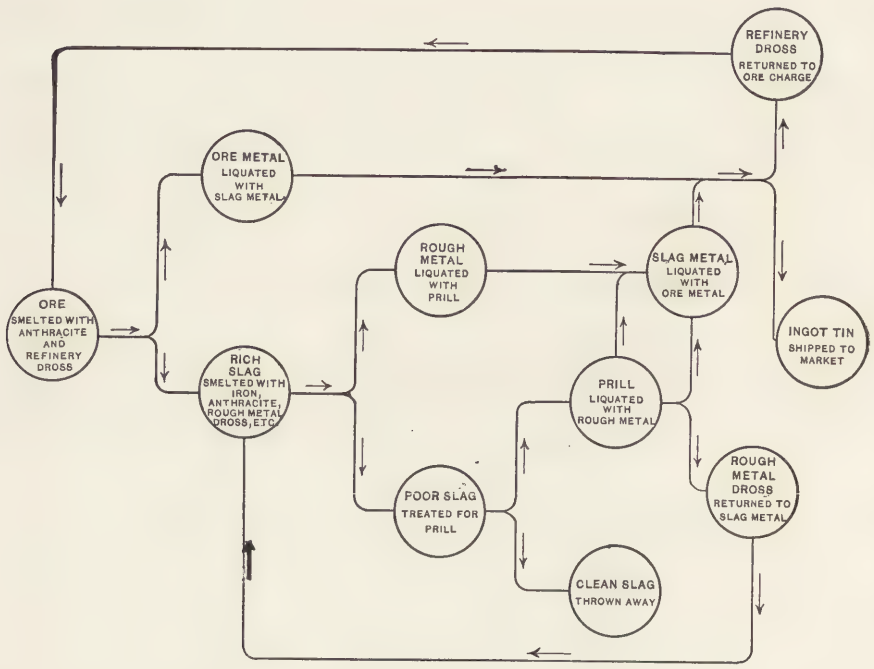
The complete process of smelting may be classified into the following four operations:

(*a*) "Ore-smelting," producing "ore metal" and rich slag."

(*b*) "Rich slag-smelting," producing "rough metal" and "poor slag."

(*c*) Treatment of "poor slag," producing "prill" and clean slag.

(*d*) Refining "ore metal," "rough metal," and "prill," producing various grades of marketable tin and "refinery dross," which is added to the "rich slag-smelting" charge or to the "ore" charge. The following diagram illustrates the cycle of operations:



(*a*) As soon as a charge has been got out of the furnace, the latter is cleaned and fettled in case of need with a mixture of fireclay and bauxite. The materials for the next charge have meanwhile been wheeled to the furnace and deposited in front of the charging-doors. The furnace being sufficiently hot, the damper is closed and the charge is shoveled in and spread over the bed, so that it is thickest near the fire-bridge, 2 ft. from which it is 12 to 14 in. deep, though less than 6 in. deep at the front door. The doors are then tightly closed, the damper opened, and the fire made up. The composition of the charge varies somewhat according to the composition of the ore, the following being average mixtures: (1) For ore assaying 65 to 71%: Ore, 80 cwt.; culm, 10.4 cwt.; refinery dross, 2.4 cwt. (2) For ore assaying above 71%: Ore, 80 cwt.; culm, 12 cwt.; refinery dross, 2.4 cwt. At the end of 2 to 2¼ hours a fresh fire is generally needed; if the coal is good the charge can be rabbled first, if not, another fire is first put on, lasting about 1½ hours; in either case the charge, when well rabbled, should be all off the bed, liquid near the fire-bridge, not too pasty toward the center, and frothing freely. Another fire is then put on, lasting about an hour, and the charge is again well rabbled; all should now be well melted,

uniformly bright all over, and free from floating masses of unfused material. At this stage another good fire is put on, and the furnace is tapped by withdrawing the tapping bar that closed the tap-hole; all the tin runs out in a thin stream into the float in about ¾ hour. The bar is then put back and a gutter placed under the tap-hole so arranged as to deliver into a series of sand-beds that have been made up beside the furnace. The whole of the clay stopping of the tap-hole is then knocked out, and the slag is allowed to run in a thick stream into the sand-beds. The furnace is then ready for a fresh charge. The average time from charge to charge is 7½ to 8 hours. The products are 4500 to 4800 lbs. of ore metal, containing about 99.5% of tin, and 2200 to 2500 lbs. of rich slag, assaying 20 to 40% of tin. An analysis of this slag gave: Tin, 35%; silicon, 15%; aluminum, 18%; iron, 9%; also Mg, Ti, Ca, Mn, etc. The tin produced is allowed to cool in the float until its temperature is sufficiently low to enable it to be ladled into iron ingot molds; the metal is dull grayish and somewhat hard and brittle, and it is sent into the refinery to be subsequently refined.

(*b*) The rich slag is next smelted, this operation being performed in the same kind of furnace that is used for ore-smelting; it is usual to put a newly built or repaired furnace on slag charges; slag-smelting requires a higher temperature than ore-smelting, and the metal produced has a considerably higher melting point, hence this smelting is of use in filling up any crevices or stopping any leaks in the new work. Similarly, when a furnace is getting worn out and is leaking seriously, a few charges of slag will often remedy the evil, at any rate temporarily. An average charge is made up as follows: Rich slag, 30 cwt.; rough metal dross, 12 cwt.; scrap iron, 2.75 cwt.; anthracite, 6 cwt.; coral, 2.4 cwt. The "rough metal dross" is a product of the refining of the metal produced in this fusion.

A good fire is put on, and then usually a second, after which the charge is rabbled in some three hours or so from the start, and in another hour the charge is usually well off the bed and pretty well ready for tapping. The entire operation takes about 7 hours. The products are 1800 to 2000 lbs. of "rough metal" and about 2400 lbs. of "poor slag." The former is either run into the float whence it is ladled into molds, being stirred continuously while settling, or it is run direct into sand molds or granulated in water, the last being apparently the most satisfactory mode of procedure. Good rough metal contains about 95.5% of tin, the rest being chiefly iron; it is dull blackish-gray in color, and very brittle. The slag is black and glossy, very variable in composition, but averaging about 60% of silica; it should not contain more than 2.5% of tin as silicate, but may contain up to 10% in the form of prills of rough metal. As a rule all the prills are contained in the two first pigs of slag; the others are examined, and if found free from prills are thrown away. Two-thirds of the slag produced is thus got rid of at once.

(*c*) The slag that contains prills of metal is broken up and any large lumps of metal are extracted; the slag may then be stamped and washed, but the brittle rough metal is thus apt to be slimed and lost. It is hence found preferable to fuse the slag with the addition of a little culm and lime. An average charge would be: Poor slag, 40 cwt.; culm, 2.5 cwt.; coal, 2.5 cwt. This is melted and rabbled once or twice, after which the slag is tapped off, the process taking 5 to

6 hours. The metal is allowed to accumulate in the furnace and is tapped every two days or so, being treated exactly like rough metal, which it greatly resembles in appearance. It contains, however, rather more iron and is often spoken of as "hard-head." Its average composition is: Tin, 80.5%; iron, 19.5%—total, 100%. The slag thrown away amounts to about 27% of the weight of the ore originally started with; on the average it may be taken as containing 5% of tin, though a good deal is much poorer than this. This average would correspond to a loss of tin equal to 1.35% of the ore, or taking the latter to contain 70% of tin, a loss of about 2% of metal. There are no losses, except such as are due to splashing, etc., in any other parts of the process, since all the products of refining are continuously returned to the furnaces. The whole of the iron added in the slag-smelting is thrown away in the slag, the iron in the rough metal being practically derived from the rough metal dross, and simply circulating, so to speak. The consumption of iron amounts to 4.7% of the tin produced, and that of culm amounts to about 27% of the tin produced.

(*d*) The metal produced from 100 parts of tin ore is now distributed somewhat as follows: (1) Ore metal 58 parts, containing 57.7 parts of tin; (2) rough metal 9 parts, containing 8.6 parts of tin; (3) rough metal from prill 2 parts, containing 1.6 parts of tin; together with slag 27 parts, containing 1.4 parts of tin—total, 69.3. The last item is, of course, the loss. The ore metal is ready for the refinery, while the rough metal obtained from operations (*b*) and (*c*) undergoes a preliminary refining in the smelting-house. A smelting furnace is charged with either granulations or ingots that have been stirred while setting, and the heat is brought up to dull redness using wood as fuel. The results are 90% of slag metal, containing about 99.5% of tin, and hence practically identical with ore metal, and "rough metal dross;" the latter contains much metallic tin and iron and oxides of these metals, together with those of most of the impurities present in the rough metal. This dross contains on the average: Tin, 65%; iron, 25.5%. As previously stated, it forms part of the slag-smelting charge, and thus all the tin in it returns into the cycle of the furnace operations. The slag metal is cast into molds like the ore metal and the two are sent to the refinery together.

The refining furnace or liquating furnace consists of a reverberatory furnace with a fireplace at either end and a flue in the center of the arch. The bed is 10 ft. long and 6 ft. wide, with two shallow transverse depressions or troughs running across it, opposite to each tap-hole, and inclining in the direction of its width with a fall of about 1 ft. At the lower side there are two tap-holes with spouts leading to floats 3 ft. 6 in. in diameter, capable of holding about 7 tons of tin each. At the higher side of the furnace there are four charging-doors. The arch is 6 ft. high in the center. The fireplaces have a grate area of 6 ft. 6 in. by 11 ft., and each has a fire-door in its respective furnace-end; the fuel used is mostly wood, but coal has been used. The temperature is maintained as little as possible above the melting point of pure tin, and the furnace body is kept full of smoke so that the atmosphere shall be a reducing one. The fuel consumption, when coal is used, is about 100 lbs. to the ton of tin refined. The ingots of tin, weighing about 100 lbs. each, are piled on the upper portion of the bed, and, as they melt down slowly, the liquated tin runs into one or other of the floats which are placed in the pouring house, a gray pulverulent mass, known as refinery dross (which

is added, as before stated, to the ore-furnace charges), being left behind on the furnace bed. This dross contains: Tin, 65%; iron, 11.5%, a considerable proportion of these metals being in the state of oxide. Every 100 parts of ore metal will give about 96.5 parts of refined metal and 4.5 to 5 parts of refinery dross. The tin in the floats is allowed to stand at rest for about one hour and is then ladled into one of a set of four large kettles set over a small fireplace, and situated at a level about two feet below the smaller floats. Each of these large kettles is 8 ft. in diameter and holds about 30 tons. The tin is thus made to undergo a tossing process during the operation of being ladled from the one kettle to the other. If the tin is not now sufficiently soft, it is "boiled" in the usual way, but this operation is more often performed before the metal is sent to the refinery if at all. The metal is allowed to stand for 24 hours in the large kettles at a temperature just above its melting point; all impurities are thus given sufficient time to settle to the bottom of the kettle; the surface is then skimmed and the tin ladled out into molds, the ingots weighing about 100 lbs., at least 25 tons of the tin being so cast. The bottom 12 in. of metal in the kettle is ladled into separate molds and is sent back to the refinery to be liquated over again.

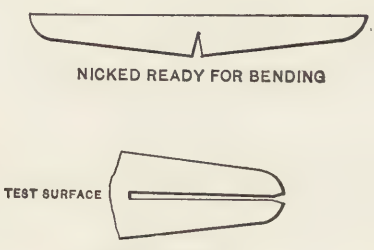


FIG. 30.—TIN TEST PIECE—WIDTH 1¼ INCH.

The metal thus produced should be white, bright colored, and should show a slight depression in the center. A sample cut from any of the corners should, if cut half-through and then bent, show a smooth, almost silky, surface, free from harsh projecting lumps or grains. During the process of casting, samples are continually taken and tested, the shape of the sample being shown in the accompanying figure (Fig. 30).

The following is an analysis, made in 1892, of an average sample of Pulo Brani tin: Sn, 99.76%; Sb, 0.07%; Pb, 0.02%; Fe, 0.14%; Cu, As, none—total, 99.99%.

The cost of smelting operations cannot be given with absolute accuracy; at the works it was always taken as ranging between $1.15 and $1.25 per picul (say 39s. to 42s.), but it varies of course considerably with the rate of exchange, and the very variable price of coal, which ranged from 18s. to 27s. per ton. The following shows very nearly the cost of operations about the year 1888, and this does not seem to have undergone much modification since that time, allowing for the depreciation in the value of silver: Fixed charges,* including rent, maintenance, office expenses, management, European supervision, etc., 7s. 5d. per ton of 2240 lbs. of ore; native labor of all kinds, 4s. 8d.; coal, 0.98 tons at £20

*The Straits dollar is throughout these calculations taken at 2 shillings or 48 cents United States currency.

6s. per ton, 20s. 1d.; culm, 3.4 cwt. at £27 per ton, 4s. 7d.; materials, iron, lime boilwood, water, etc., 11d.; repairs to tools and furnaces, 1s. 6d.—total cost per ton of ore, 39s. 2d., equivalent to about 56s. per statute ton of metallic tin, or $12 per ton of 2000 lbs.

It has been already stated that all the ore is bought, generally through agencies in the various native states, the following being the system of payment adopted by the Straits Trading Company: The ore is weighed and 3 to 5% is deducted for moisture. It is then assayed by the cyanide method, and a deduction made for the supposed loss in smelting as judged by the softness and general character of the assay button, this first deduction being from 0.25 to 5%, or sometimes more. From the percentage of gross tin thus obtained, a second deduction for loss in smelting is made according to the following scale: From ores giving over 70% gross, 2% is deducted; from 67.5 to 70%, 3% is deducted; from 65 to 67.5, 4% is deducted, and below 65% the scale of deductions increases still more rapidly. If the assay shows the presence of very much impurity, a still higher deduction is made. From the weight of the dry ore and reduced assay value thus obtained, the weight of tin is calculated and is paid for in cash at the current market rate less a deduction of $1.75 per picul of ore (about £2 18s. 10d. per statute ton), of which $1.50 per picul (about £2 10s. 5d. per ton) is supposed to be for smelting charges and the remainder for profit. From the figures quoted above, the company has evidently allowed itself a liberal margin, although it must not be forgotten that it has to pay agency, transport, and a few other similar charges in addition to smelting charges proper.

In 1893 the Pulo Brani works smelted 13,500 tons of ore and produced 10,470 tons of tin; in 1894 the output rose by about another 500 tons, and is still increasing steadily.

It will thus be seen that the Pulo Brani practice presents the highest stage of development of the process of tin-smelting. Beginning with the most rudimentary methods, which were evidently the primitive ones in all parts of the world where tin has been found, we have been able to trace their step by step development and the causes that led to the adoption of the more scientific reverberatory furnace, and the few improvements which this process has since undergone. It is in the highest degree significant that the chief improvements have originated in the Pulo Brani works, which are by far the largest works in the world, smelting half as much tin again as the whole of Australasia, and just about as much as the whole of Great Britain turns out from some seven or eight works. There is no doubt that we have in these facts the clue to the real reason why the metallurgy of tin has not kept pace in any measure at all with that of the other common metals, it being remembered that of all the common metals tin is by far the rarest. Comparing, for instance, the world's annual output of these metals, namely: Lead 600,000 tons, zinc 360,000 tons, copper 340,000 tons, with that of tin, amounting to about 90,000 tons, the difference is sufficiently striking; at the same time it must be borne in mind that one single copper-smelting company (the Anaconda) produces annually some 45,000 tons of copper, or half as much as the world's output of tin, and over four times as much as the tin production of the Pulo Brani works, eminent though the latter is, as we have seen, among tin producers. As is always the case, when metallurgical

operations are conducted on a small scale, improvements come slowly, and ancient methods survive with surprising tenacity of life. So far no electrical or wet methods have proved of the slightest use in the extraction of tin from its ores, and in spite of the proverbial danger of prophesying, it seems as though it may fairly be expected that a good many years will yet pass away before any very radical advances in the art of tin-smelting will have to be chronicled.

Mr. Walter Renton Ingalls contributes the following notes of tin-smelting in California and Mexico:

"At the Temescal mines in California, which were exploited in 1890–92 by an English company, the ore was smelted in Cornish reverberatory furnaces, of which the only novel feature was the use of petroleum as fuel. They produced 135 tons (2000 lbs.) of tin before the mines were closed down, the lode having proved unprofitable.

"The ore averaged 3 to 3½% black tin. It was crushed by Husband pneumatic stamps to 1 mm. size and washed on vanners, the concentrates of which were washed again on a stationary, convex buddle. About 20 tons per diem were dressed when the mill was in operation, for which a crew of 8 men was required. The concentrates assayed about 65% Sn. They were smelted in a Cornish reverberatory furnace with hearth 10½ by 11 ft., using California petroleum for fuel and New Mexican coal for reducing agent. Coal cost $10.50 per ton of 2000 lbs. at the mines, and petroleum 4.2c. per gal., which was probably equivalent to coal at $6 per ton. The concentrates were charged in lots of 1800 lbs., mixed with 20% coal. Three charges were smelted per diem. The slags assayed about 5% tin, and were restamped. The cost of smelting was probably about $9 per ton, without allowing for general expenses, etc., which would be misleading if included, since the furnace campaigns lasted only a few days at the end of each month, when the accumulated stock of dressed tin ore was smelted. The tin was of good quality, being equal at least to the best Cornish.

"In Mexico a small amount of tin is smelted by the natives in small blast furnaces constructed of uncut stone, plastered inside with clay. The hearth is sharply inclined toward the front, which is closed by a clay breast, through a hole in which slag and metal run into an outside basin scooped in the ground. The slag chilling in this is lifted off the metal with hooks, and the metal is cast into slabs, or allowed to set in the basin and marketed in the mass finally lifted out. From this it may be surmised that operations are conducted on a very small scale. One of these furnaces measured 5 ft. from hearth to top, 9 by 9 in. at the tuyère and 6 by 9 in. at the top; another was 4 ft. high, 9 by 7½ in. at the top and 6 by 7 in. at the tuyère. In the former the tuyère was 18 in. above the hearth, and in the latter 12 in. These furnaces were blown by a pair of ordinary smith's bellows, working alternately through a single tuyère in the back of the furnace. Each furnace required two men at the bellows and one smelter. The fuel was charcoal, and the slags were rich, both in prillions and in silicate of tin. Precisely the same kind of furnace is used by the natives for smelting silver-lead ore. It will be noted that thsi is practically identical with the more preimitive Chinese process.

"An American company operating at Potrillos, in the State of Durango, erected two Cornish reverberatory furnaces, fired with wood, but enough ore was never

smelted to establish a practice. A noteworthy fact in connection with this venture, however, was the remarkable percentage of antimony in the metal that was produced, one lot of 310 ingots showing tin 90.6047%; antimony, 9.2850%; arsenic, trace; lead, 0.0555%; bismuth, 0.0276%; iron, 0.0270%—total, 99.9998.%* A sample of metal from a native furnace at Sain Alto in Zacatecas gave 6.83% antimony.

"There is no tin produced in Mexico at the present time except by the natives, who smelt in the little furnaces described at a number of places for metal for local needs. The uses they have for tin, however, are few, and the smelting is consequently carried on only spasmodically."

*Ingalls, "The Tin Deposits of Durango," *Trans. Am. Inst. Min. Eng.*, Vol. XXV.

ZINC AND CADMIUM.

By Walter Renton Ingalls.

The chief features of the zinc industry in the United States in 1896 were a combination of a large number of the smelting companies, and the exportation of spelter and ore to Europe. The first was consummated in January, when the Cherokee Smelting and Refining Company (Pittsburg, Kan., and Weir City, Kan.), Robert Lanyon & Co. (Pittsburg, Kan., and Nevada, Mo.), S. H. Lanyon & Bros. (Pittsburg, Kan.), Pittsburg & St. Louis Zinc Company (Pittsburg, Kan.), Girard Zinc Company (Girard, Kan.), Rich Hill Mining and Smelting Company (Rich Hill, Mo.), Scammonsville and Columbia Smelting Companies were united, partly by purchase, partly by lease, in the Cherokee-Lanyon Spelter Company. These works had then a capacity of 35,000 tons per annum, at the most, while works with capacity of 40,000 tons, including the best equipped of all, viz., those of Peru and LaSalle, Ill., remained outside the combination. The new company promptly adopted a policy of concentration of smelting in a few works and restriction of output, the effect of which is outlined in the market review subjoined.

At the beginning of September the price of zinc had fallen to 3.25@3.30c. at St. Louis, which was below the parity of the European market, where conditions had been steadily improving since the first of the year, and it became possible to export American spelter, which was done until the first of November, when it became more profitable to market the product at home. The total exportation for the year amounted to 10,130 tons.

Shipments of zinc ore to Europe also attained important proportions, about 15 carloads going from Joplin to Swansea, a little from Tennessee, and a good deal from New Jersey, the total amounting to 2075 tons.

It is unlikely that exports of zinc ore from Missouri will attain large volume in the near future, owing to high freight rates to the seaboard, but it is probable that those from New Jersey will become more and more important. The business originated through the production on a large scale of zinkite-willemite concentrates by the Wetherill process at Franklin Furnace, N. J. These concentrates average 49 to 51% Zn, 2 to 3% Fe, 4 to 5% Mn, and scarcely more than a trace of lead, wherefore they are very desirable for the manufacture of high-grade spelter. For this purpose a market for them has been found at the Grillo works in Westphalia, and the exportation, which was begun about the first of September, now amounts to from 700 to 1000 tons per month, which is shipped in bulk via Antwerp or Rotterdam.

It is not easy to understand how zinc can be produced at a profit when the price in St. Louis is as low as 3.25c., or even 3.50c., which was more than the average in 1895. Yet new works have been built at Iola, Nevada, and Upland, in the Missouri-Kansas district, and others are planned. Probably there are few which are making money except the larger works, especially those of Illinois, which have rolling mills and sulphuric acid factories in connection with them. The Illinois Zinc Company, of Peru, Ill., followed the example of its neighbor at LaSalle and installed an acid plant last year. Probably the only other noteworthy improvements in the Western works was the introduction of new mechanical roasting furnaces, the Brown, Pearce, etc., which are doing good work and clearly controverting the European opinion as to the advantage of automatic blende-roasting.

PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1891.	1892.	1893.	1894.	1895.	1896.
Illinois and Indiana	28,660	30,227	29,725	28,948	33,748	31,656
Kansas	21,460	22,953	22,085	26,060	25,916	36,566
Missouri	16,205	16,169	13,737	12,000	10,726	
South and East	13,938	14,733	10,708	6,996	11,468	9,415
Total tons of 2000 lbs.	80,263	84,082	76,255	74,004	81,858	77,637
Total tons of 2240 lbs.	71,662	75,073	68,525	66,074	73,088	69,319
Total metric tons	72,816	76,258	69,159	67,135	74,245	70,432

PRODUCTION OF ZINC OXIDE IN THE UNITED STATES.

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1893	25,000	22,679	$1,875,000	$75.00	1895	22,690	20,498	$1,588,300	$70.00
1894	22,814	20,697	1,711,275	75.00	1896	16,799	15,240	1,259,925	75.00

IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES. (IN POUNDS.)

Year.	Sheets, Blocks, Pigs, and Old.		Manufactures.	Total Value.	Oxide.	
					Dry.	In Oil.
1892	410,896	$23,307	$22,709	$46,016	2,442,014	111,190
1893	425,998	22,301	20,756	43,687	3,900,749	254,807
1894	512,932	17,271	12,342	29,613	2,850,771	113,540
1895	864,113	29,352	12,183	41,535	4,546,038	129,343
1896	856,044	25,904	15,728	41,632	4,572,781	311,023

EXPORTS OF ZINC AND ZINC ORE FROM THE UNITED STATES. (IN POUNDS.)

Year.	Ore and Oxide.		Plates, Sheets, Pigs, and Bars.		Manufactures.	Total Value.
1892	2,058,560	$41,186	12,494,335	$669,549	$161,794	$877,529
1893	109,760	1,271	7,446,934	413,673	225,357	640,301
1894	.....	5	3,621,934	144,278	99,418	243,701
1895	48,000	1,008	3,141,285	155,975	50,126	207,109
1896	4,150,000	47,408	20,260,169	1,013,620	51,216	1,112,244

The zinc industry in Europe was more prosperous in 1896 than in the two years previous, consumption being large and prices having an upward tendency. As a

consequence production was increased about 3000 tons in Silesia and 8000 tons in Belgium (the latter being attributable chiefly to the Vieille Montagne Company). In the Rhine district there was a falling off in production from 27,125 to 26,420 long tons, and in England from 29,495 to 25,880 long tons, but this was a good deal more than made up in France, where the Malfidano Company more than doubled its output, and the Real Asturiana made a large increase. The Spanish works of the latter company, however, which is the only producer in Spain, put out only the same amount of metal as in the previous year. The small works at Limburg, Holland, reported no production.

In Upper Silesia there was the same number of works in operation as in the previous year, but the industry is now concentrated in the hands of eight companies. The "Bergwerksgesellschaft Georg v. Giesche's Erben" began the erection of a new works, which is expected to be in operation by the middle of 1897. The Grillo works in Westphalia began the manufacture of high-grade spelter from New Jersey ore in rivalry with the Vieille Montagne Company, and a small amount of electrolytic spelter from Duisburg was on the market. A new zinc-rolling mill was established at Budapest, Hungary.

PRODUCTION OF ZINC IN THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Austria.	Belgium.	England.		France.	Germany	Russia.	Spain.	United States.	Totals.
			Native Ores.	Foreign Ores.						
1890.....	5,449	82,701	8,692	20,922	19,372	139,266	3,774	5,919	61,111	347,206
1891.....	5,006	85,999	9,037	20,846	20,596	139,353	3,677	5,592	72,816	362,922
1892.....	5,237	91,546	9,500	21,302	20,609	139,938	4,374	5,925	76,258	374,689
1893.....	5,870	95,665	9,585	19,397	22,419	142,956	4,522	5,752	69,159	375,325
1894.....	6,810	97,041	8,260	24,448	23,387	143,577	5,014	5,100	67,135	380,772
1895.....	6,456	107,664	6,760	23,241	24,200	150,286	*b*5,039	5,845	74,245	403,736

(*a*) Compiled from the official reports of the respective countries, with the exception of the column of zinc in England from foreign ores, which is arrived at by deducting the zinc as reported in the official blue books from the total output of the smelting works as stated in the reports of Messrs. Henry R. Merton & Co.
(*b*) Merton's figure.

PRODUCTION OF ZINC IN THE WORLD. (*a*) (IN METRIC TONS.)

Year.	Belgium, France, and Rhenish Prussia	Upper Silesia.	Great Britain.	Spain.	Austria.	Russia (Poland).	United States	Totals.
1895.....	192,305	95,519	29,967	5,845	8,489	5,039	74,245	411,409
1896.....	205,511	97,409	26,294	6,000	9,403	6,264	70,432	421,313

(*a*) This table is based on the figures of Henry R. Merton & Co. Those for Belgium, France, and Rhenish Prussia do not permit of segregation in advance of the official statistics, since certain companies own works in two or three countries and report their production in the aggregate. For Spain the figures reported by Señor Don Roman Oriol have been adopted. The statistics for the United States are those collected from the producers by THE MINERAL INDUSTRY. It will be observed that Merton's figures for Austria in 1895 are higher than the official report gives, and since this has been uniformly the case for several years it may be assumed that those for 1896 are correspondingly high, notwithstanding that the Austrian statisticians include the make of zinc-grey with the spelter. The discrepancy appears to be due to the Galician works, which give higher figures to Messrs. Merton than to the Government, those for the three remaining works in Austria agreeing closely. Merton's figures for Belgium, France, and Germany in 1895 give a total of 287,824 metric tons, against 282,150 tons reported by the three governments for that year.

ZINC MINING IN THE UNITED STATES.

The most important feature in zinc ore mining in the United States in 1896 was the final adjustment of the litigation between the New Jersey companies and the probability of the speedy inauguration of operations by them on a larger scale.

Missouri-Kansas.—The Joplin correspondent of the *Engineering and Mining Journal* writes that 1896 was a prosperous year in that district, the only drawback being a succession of heavy rains in May, which interrupted work in many mines. This, however, is a regular annual happening there, and will be so long as mining is carried on in Joplin in the present fashion; consequently it did not prevent an increase in the output of ore, which was 153,082 tons of 2000 lbs., being 8595 more than in 1895. This was due partly to the opening of new mines and partly to the high price of ore maintained throughout much of the year, the range of prices having been as follows, in cents per pound:

Month.	Average Price of Ore at Joplin.		Average Price of Zinc in New York Per Pound.	Difference Between Prices of Ore and Spelter.	Prices of Spelter in St. Louis.	
	Per Ton.	Per Pound of Zinc.			Highest.	Lowest.
January	$24.00	2.00c.	3.75c.	1.75c.	3.90c.	3.65c.
February	23.50	1.96	4.03	2.07	3.90	3.75
March	23.00	1.92	4.20	2.28	3.95	3.88
April	23.00	1.92	4.09	2.17	3.93	3.70
May	21.50	1.79	3.98	2.19	3.90	3.70
June	21.00	1.75	4.10	2.35	3.93	3.73
July	21.50	1.79	3.97	2.18	3.88	3.70
August	21.00	1.75	3.76	2.01	3.80	3.50
September	20.00	1.62	3.60	1.98	3.65	3.35
October	20.50	1.71	3.72	2.01	3.70	3.45
November	23.50	1.96	3.99	2.03	4.05	3.75
December	25.50	2.12	4.14	2.02	4.10	3.90
Average	$22.33	1.86c.	3.94c.	2.09c.	3.89c.	3.67c.

The estimated value of a pound of zinc in ore in the above table is only approximate. Ore is not sold at Joplin by assay, although of course its price is determined by its metallic contents. The average price quoted in the table is supposed to be for first-class ore containing 58 to 62% zinc. In calculating the second column the ore is assumed to average 1200 lbs. zinc per ton.

New mines at Galena, Kan., and in the vicinity of Joplin, Mo., contributed largely to the production of the district. At the close of the year there was no stock of ore in the local market, against 2000 tons more or less at the beginning.

Arkansas.—In Arkansas ore has been mined near Lion Hill, Marion County, by the Chicago & Little Rock Mining Company, which has a rough dressing plant in operation. The mines are situated near the head of steamboat navigation on White River (which, it is said, never freezes), and 100 miles above Batesville. At Batesville the zinc ore is reshipped by the Iron Mountain Railroad to destination. This river is claimed to be navigable for steamboats eight to nine months each year.

New Jersey.—The chief event in zinc mining in New Jersey in 1896 was the final decision by the Court of Errors and Appeals in the case of the New Jersey Zinc and Iron Company *vs.* Lehigh Zinc and Iron Company. The property in question was the well-known Mine Hill ore deposit near the village of Franklin Furnace, which was transferred in 1848 by Col. Samuel Fowler to different parties, one deed conveying the right to mine the zinc ore, locally known as "red zinc ore," and another the right to the franklinite ore, which was at that time held to be of value only as an iron ore. The willemite, which occurs in connection with the franklinite, was not then considered valuable. The litigation first began in 1857, shortly after the discovery of the value of the zinc in the franklinite, and in one form and another was carried on until 1877, when the United States courts finally decided that the franklinite title, at least so far as it referred

to the southern half of the deposit, was vested in Moses Taylor, of New York, who had purchased it from the original owners. At the same time the zinc title covering the entire tract was held to be with the New Jersey Zinc Company. After the decision these parties consolidated into the New Jersey Zinc and Iron Company, which thus acquired the franklinite title for the southern half and the zinc title for the whole mine. The franklinite title to the northern half was then owned by the Lehigh Zinc and Iron Company. The New Jersey Zinc and Iron Company, however, claimed that as the greater part of the value of the franklinite had been found to be in the zinc obtained, the old title was no longer valid, and the zinc title really covered all. The action in its last form—a suit in ejectment brought by the New Jersey Zinc and Iron Company against the Lehigh Zinc and Iron Company—was tried in the Circuit Court for Sussex County in May, 1894, ending in a disagreement of the jury. Another trial resulted in a verdict for the defendant, the Lehigh Company, and this judgment was confirmed by the highest court of the State.

Not long after this decision concluding the litigation the two contestants, together with the Passaic Zinc Company, consolidated as the New Jersey Zinc Company, which took over the zinc mines and works of the three. The new company is making arrangements to exploit its vast property on a larger scale than ever, which has become possible by the unqualified success of the Wetherill process for concentrating the ore, but in all probability will confine itself to the manufacture of zinc-white and high-grade spelter rather than attempting the control of the Eastern zinc market.

Pennsylvania.—The Correll mines, situated in the Saucon Valley, at Friedensville, which contain valuable deposits of zinc ore, and have been operated under various leases since 1858, by Samuel Wetherill, Franklin Osgood and others, but have not been worked recently, have again come into the hands of the heirs of the original owner, the late Jacob Correll, of Bethlehem.* The ore taken from these mines formerly had the reputation of producing a very high grade of metal.

ZINC MINING IN EUROPE.

Algeria.—New explorations for zinc ores have been undertaken in the Djebel Belkif, commune de Morsott, where there are promising outcrops of galena and calamine not far from the Youks and Morsott stations of the Bone-Guelma line of railway from Soukarras to Tébessa.

Greece.—The production of zinc ore in Greece in 1896 showed a falling off, owing to the gradual exhaustion of the great reserves of the Laurium mines. The calamines, which formerly assayed 60% Zn after calcination, are now very scarce, and the more part of that which is being mined at present is lower in zinc and more ferruginous. The production in 1896 was 22,700 tons.

Italy.—The production of zinc ore in Italy keeps fairly steady. By far the largest part comes from the Province of Cagliari, in Sardinia, but about 15,000 tons per annum is mined in Bergamo on the mainland.

Upper Silesia.—The production of calamine in Upper Silesia in 1896 was 263,224 metric tons; of zinc blende, 274,850 tons; of lead ores, 31,198 tons; of iron ores, 7556 tons; and of pyrites, 3543 tons. The coke-blast furnaces produced 6857 tons of zinc dust, furnace crusts and fume, which were sold to the zinc

* *Engineering and Mining Journal*, Nov. 7, 1896.

smelters. The production of zinc, lead (by zinc smelters only), and cadmium are given in the accompanying tables. The zinc-white works in addition turned out 875 tons of zinc-white, 56 tons of zinc-gray and stone-gray, 11 tons of lead, and 175 tons of zinc-white residues, etc. The rolling mills produced 39,490 tons of sheet zinc, 655 tons of lead (from the refining furnaces), and 860 tons of zinc ashes and other by-products. The blende-roasting works produced 39,916 tons of 50° sulphuric acid, from which 28,720 tons of 60° acid were concentrated, and 6934 tons of 66°. The production of sulphurous acid amounted to 927 tons. The number of persons employed in the zinc and lead mines at the end of the year was 9528; in the spelter works, 7599; in the zinc-white works, 28; in the rolling mills, 714; in sulphurous and sulphuric acid fabrication, 789.

Russia.—Rich discoveries of zinc ores have been from time to time reported from Southern Russia and the Caucasus. A Belgian company has now encountered a rich deposit at Alagar, about 50 versts from the Wladikawkas railway, and is considering the production of spelter. Oleboff & Co. (incorporated) have been for two years investigating the treatment of various ores in Rostow on the Don, but their experimental work is not yet concluded. A deposit of blende and galena has recently been uncovered at Petrowsk in the Caucasus. Blende occurs in the Kutaïs government near the river Tschorok between Artwin and Bortschka.

The Société Minière Franco-Russe, with a capital of 6,000,000f., has been organized to exploit 8 zinc mines and 4 coal mines in the Bendin district (government of Petrokow) and Olkusz district (government of Keletz) in Poland, for which it has an imperial concession for a period of 60 years. It is also given possession under lease of the existing works on this territory, which covers an area of 823 hectares. The company has been conducting explorations since 1892, and has proved the existence of an immense body of ore, of which 75% is estimated to be calamine and 25% blende. It is proposed to erect modern smelting works.

The Polish zinc deposits and coal seams are extensions of the Upper Silesian, and with the introduction of improved methods of exploitation the zinc industry in Poland ought to assume increased importance. It can certainly supply the entire demand of the Russian Empire. At present there are only two producers of spelter in the kingdom.

Spain.—The more part of the blende and calamine of Spain comes from the Province of Santander, where the Real Compañia Asturiana in 1896 produced 24,000 tons of crude calamine from the Reocin mines and 7200 from those of Udias and La Florida. Reocin turned out 810 tons of blende besides the calamine. In Leon the Real Compañia Asturiana obtained 1000 tons of calamine, while the production of Murcia was 10,000 tons.

The production of spelter was confined to the works at Arnao in Asturias, which produced 6000 tons, *i.e.*, about the same amount as in 1895.

Tunis.—Five calamine deposits are worked under concessions in this province, all situated in the northern part.*

1. Kanguet et Tout lies about 30 kgms. from Beja, on the road from Beja to the port of Tabarka, which at present is only finished from Beja to the mine. The ore occurs in an irregular stock-work. The mine is worked open-cast. The

* E. de Fages, *Rev. génér. des Sciences; Glückauf*, March 27, 1897, p. 245.

ore is sorted into lump and fines, the former being calcined in kilns with charcoal. The calcined calamine is freighted by mule or camel to Beja, and thence to Tunis. From Tunis the ore goes to Antwerp, where in 1896 it brought an average of 100f. per ton. The production was between 3000 and 4000 tons.

2. The Sidi-Ahmet concession, belonging to the Real Compañia Asturiana, lies north of the Sidi-Ahmet Mountains, about 40 kgms. from Beja. It contains at least three deposits, of which the first has yielded already about 35,000 tons of good calamine. The ore is hand-sorted and calcined in kilns. The annual production is about 3500 tons calcined ore, which brings about 90f. per ton in Antwerp.

3. Fedj-el-Adoum lies 20 kgms. southwest of Tebursuk, in the highest part of the Jouaouda Mountains, which rise to an elevation of 907 meters. There are three groups of deposits, of which only one is exploited. This is very extensive and already about 50,000 tons of ore are in sight. The ore is calcined, after hand-sorting, and shipped by mule to Pont-de-Trajan, but a road will be built to Tebursuk. The production is not yet normal, amounting to only 1500 tons in 1896 against 2500 in 1894.

4. Zaghouan lies about 60 kgms. south of Tunis, near the village of Zaghouan. There are two important deposits, and already about 40,000 tons of carbonate ore, averaging 40% Zn, have been proved. Zinc silicate appears more frequently here than in the other localities. The mines are connected by a mule-path and a 15-kgm. tramway with the stone road between Tunis and Zaghouan, over which the ore is transported in two-wheeled carts. There are three shaft furnaces and one reverberatory at the mines for calcining the ore. The annual production is 5000 tons of calcined ore, which brings 80f. per ton at Antwerp.

5. El Akhouat lies about 32 kgms. southwest of Tebursuk. The ore is carried in two-wheeled carts to Madjez-el-Bab, and thence by rail to Tunis. This concession was only granted in 1896, and its production has not yet become important.

PRODUCTION OF ZINC ORE IN EUROPE. (IN METRIC TONS.)

Year.	Algeria	Austria	Belgium.	Bosnia.	France.	Germany.	Great Britain.	Greece. (*b*)	Italy.	Norway. (*a*)	Russia.	Spain.	Sweden
1891	13,636	28,828	14,280	47	56,300	793,544	22,580	28,344	120,685	498	3,676	78,216	61,591
1892	21,907	33,944	12,260	16	69,236	800,167	27,311	27,695	129,731	576	4,369	74,265	54,981
1893	24,400	30,531	11,310	*Nil.*	74,400	787,919	23,880	22,589	132,767	(*c*)	4,501	62,616	46,623
1894	29,703	28,491	11,585	*Nil.*	80,065	728,616	22,170	20,830	131,777	200	1,440	58,964	47,029
1895	.....	25,862	12,230	.....	.....	706,423	17,758	24,031	121,197	.....	.....	54,109	31,349

(*a*) Zinc-lead ore. (*b*) Including blende and calamine, calcined. (*c*) Not reported in Government statistics.

PRODUCTION OF ZINC AND LEAD ORE AND OF ZINC IN UPPER SILESIA. (*a*) (IN METRC TONS.)

Year.	Blende.	Calamine	Total Zinc Ore.	Iron Pyrites. (*b*)	Iron Ore. (*b*)	Lead Ore.	Zinc.	Zinc Sheets.	Zinc-White, etc.	Lead. (*c*)	Average Value per Ton of Zinc.
1892..	291,617	368,230	659,847	2,520	9,371	29,049	88,175	33,266	895	659	$97
1893..	287,395	348,654	636,049	2,107	7,166	30,742	91,716	35,187	211	693	79
1894..	251,040	323,295	574,335	2,874	5,808	33,898	92,546	34,518	1,267	697	69
1895..	264,746	272,947	537,693	2,316	7,920	31,904	95,440	35,677	1,250	1,002	275
1896..	274,850	263,224	538,074	3,543	7,556	31,198	97,826	39,490	1,002	1,139	297

(*a*) From *Statistik der Oberschlesischen Berg- und Hüttenmännischen Vereins.* (*b*) Products of the zinc-lead mines. (*c*) Production of the zinc smelters.

METALLURGY.

In the old system of distillation but few novelties were introduced in 1896. Lynen's furnace in which the retorts are connected with a common condensing chamber, cooled to from 415° to 550° C. and retaining a layer of liquid zinc on the bottom—a proposal which has merit—has not yet come into practical use. Nor have the Francisci muffles, with which experiments have been made in Upper Silesia.

Blende Roasting.—According to Prost only a small percentage of zinc in roasted blende, *e.g.*, 0.20% out of 57.80%, is present as sulphate, while nearly the whole of the lead (5.02% out of a total of 5.60%) is in that form. Both iron and zinc sulphides are almost completely converted to oxides, lead sulphide becomes sulphate or silicate, while calcium and magnesium form sulphates. There is evidence that ZnO enters into combination with Fe_2O_3 as $ZnFe_2O_4$ in the roasting furnace. Three samples treated with ammonium tartrate gave the following results:

Number.	Iron.	Zinc.	Zinc Soluble in Amm. Tart.	Total Zinc.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I	15.20	45.29	28.82	63.64
II	1.50	62.86	59.18	95.42
III	5.28	57.80	43.16	74.67

Distillation.—The chemical behavior in the retort of zinc oxide mixed with impurities ordinarily present in ores has been investigated by E. Prost.* He finds that the reduction of the oxide is unaffected by presence of Fe_2O_3, Al_2O_3 and PbO, taking place at 1075° C. in case of pure ZnO, and at 1200° to 1250° when mixed with the other oxides in equal molecular proportions. Zinc sulphide is decomposed by iron, the action at any given temperature depending upon the proportion of iron, which should always be in excess of the amount theoretically requisite for combination with the sulphur. With a mixture of ZnS and 190% excess of iron the zinc was wholly volatilized at 1200° to 1250° C., while at 1075° considerable remained in the residue. Lime proved more efficient than iron, an excess of 75% over the theoretical quantity decomposing the zinc sulphide completely at 1250° C., while with sufficient lime the same result can be obtained at the melting point of gold (1200° C.). These experiments were made with a chemically pure, crystalline sphalerite.

At 1200° to 1250° calcium sulphate exerted no retarding influence on the reduction of zinc oxide, even in presence of lead and iron silicates, which might convert reduced calcium sulphide to silicate and facilitate the formation of zinc sulphide. Magnesium sulphate, which at high temperature is decomposed by carbon into magnesia and sulphur dioxide favors the production of zinc sulphide.

Experiments with a mixture of ZnO, PbO, and SiO_2 heated together at 1000° C. and reduced by carbon at 1200° to 1250° C. indicated the formation of a double silicate irreducible at that temperature, but was completely reducible at 1450° to 1500.° The conclusions drawn from these experiments are that no

* *Bull. de l'Assoc. Belge des Chimiste*, X., 6, pp. 246-263; *Journ. Soc. Chem. Ind.*, 1896, p. 813.

combinations are formed during the reduction by carbon, which cannot be reduced at the ordinary working temperature of the zinc furnace; loss in lead-bearing residues may be reduced by judicious admixture of lime in the retort charge; the reduction of zinc oxide is affected to an important extent by the intimacy of mixture of the charge.

Retort Residues.—The residues from the muffles of the distillation furnaces in Upper Silesia contain from 3 to 4.5% Zn, together with much valuable coal, since of the coal mixed with the ore only that is consumed which can obtain oxygen from the ore and is burned at the opening of the muffle—not a large part of the whole. Two distillation tests were made, the data and results being as follows:*

Number of Test.	Roasted Blende Under 2 mm. Size.		White Calamine. (a)		Red Dolomitic Calamine. (a)		Average Zinc Tenor of Charge.	Reduction Coal.			Loss of Zinc in Distillation.	Zinc Tenor of Residues.
	Amount	Zinc Tenor.	Amount	Zinc Tenor.	Amount	Zinc Tenor.		Small Coke, 4–30mm	Coal Dust, 1–15mm	Total Coal.		
	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
I.....	29.1	34.0	23.6	14.0	47.3	9.5	16.29	28.4	17.7	46.1	23.49	3.20
II.....	21.0	34.0	26.3	14.0	52.7	9.5	15.41	26.0	12.2	38.2	28.06	4.12

(a) The calamine was dried, but was not calcined.

Five analyses of muffle residues were as follows: I., 4.12% Zn; II., 3.20%; III., 4.01%; IV., 5.41%; V., 5.16%; VI., 5.36%. In washing V. and VI. concentrates assaying 13.80 and 14.60% Zn, respectively, were obtained, but only 7.92 and 8.15%, respectively, were in the form of ZnO. Ashes amounting to 7063 kgms. from these experiments were washed for unburned coal, yielding 1505 kgms. of 4 to 30 mm. size, at a cost of 12.71 marks, or 0.845 per 100 kgms., which is too dear. The conclusion reached by Dr. Steger was, therefore, that it would be unprofitable to dress the Silesian muffle residues, and the hope that the zinc in the old heaps would be available is not likely to be realized.

The condition of the zinc remaining in retort residue was also investigated by E. Prost.† In a sample containing 4.76% Zn, 2.36% was found to be in the form of sulphide.

Retort Fabrication.—The chief requisites for retorts are that they shall be highly refractory and shall burn dense and free from rents. Such material is obtained by mixture of chamotte with just enough clay to bind the chamotte grains, the proportion of clay being kept as low as possible, since in burning the retort the clay shrinks, and favors the formation of cracks. The best clay is chosen for this purpose, but here the difficulty presents itself that the most refractory clays are those which have the greatest shrinkage, while the dense burning clays—the sandy clays—have less durability. This is overcome by a judicious mixture of the two kinds, remembering always that the fusing point of the mixture is lower than the arithmetical mean of the fusing points of the two components.

The properties of the clays used in retort and muffle fabrication in Upper Silesia and Westphalia have been described by Steger,‡ from whose data I have compiled the following table:

* Steger, *Zeits. für Berg- Hütten- und Salinenwesen im Preuss. Staat.*, XLIV., No. 2, 1896, p. 147.

† *Bull. de l'Assoc. Belge des Chimistes*, X., 5, pp. 212–225.

‡ "Die Retorten der Oberschlesischen und der Rheinisch-Westfälisch Zinkhütten," *Zeits. für Berg- Hütten- und Salinenwesen im Preuss. Staat*, XLIV., No. 1, 1896, pp. 1–12.

	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.	X.	XI.	XII.	XIII.	XIV.
	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Al_2O_3.....	39.25	39.93	39.00	39.60	39.76	39.31	32.66	36.75	27.00	29.65	29.00	22.72	28.38	26.70
SiO_2.....	44.76	44.88	45.46	44.64	44.87	45.61	50.41	49.00	68.00	53.02	53.86	65.39	58.10	59.18
MgO.....	0.36	0.08	Trace	Trace.	Trace	.....	.....	0.56	}	0.78	.....	0.23	.....	0.89
CaO.....	0.26	0.21	.....	.....	0.76	0.37	0.50	.....		1.15	0.46	.....	.....	1.55
Fe_2O_3	0.48	0.99	0.011	0.014	1.14	1.13	3.23	0.80	} 5.00	a3.40	a1.81	0.91	0.57	1.93
K_2O.....	1.55	0.52	.....	} Tr.	0.67	0.66	1.56	0.41		0.55	.....	0.86	} 2.49	1.32
Na_2O.....	.....	.....	.....		.....	.....	.....	0.37	}	.....	.....	1.84		
Loss.....	13.41	13.03	14.26	14.96	12.95	13.25	11.64	11.87	.....	10.91	14.87	7.77	10.54	8.00
Totals....	100.07	99.64	98.731	99.214	100.15	100.33	100.00	99.76	100.00	99.46	100.00	99.72	99.08	99.57

	XV.	XVI.	XVII.	XVIII.	XIX.	XX.	XXI.	XXII.	XXIII.	XXIV.	XXV.	XXVI.	XXVII.	XXVIII.
	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Al_2O_3.....	29.33	27.23	43.00	36.69	42.69	44.92	38.3	45.21	46.39	27.62	34.78	29.53	19.49	17.62
SiO_2.....	57.55	61.69	54.70	49.42	55.73	54.00	b43.84	52.50	51.87	51.25	c49.64	57.95	71.79	73.58
MgO.....	0.26	.....	.....	Trace.	0.23	Trace	0.19	0.54	0.45	0.45	0.41	0.36	0.06	0.40
CaO.....	0.74	0.38	0.30	0.09	0.28	0.48	0.19	.....	.....	0.23	0.68	1.72	.....	1.24
Fe_2O_3	2.60	1.24	1.70	0.73	0.37	0.80	0.46	0.81	0.52	d1.14	1.80	1.88	0.85	1.29
K_2O.....	.....	.....	0.30	0.43	0.50	0.60	0.42	0.51	0.32	1.89	0.41	.....	} 0.85	.....
Na_2O.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.85	.....	.....		
Loss.....	9.33	9.61	.....	12.70	0.43	.....	17.78	0.78	.....	14.75	12.00	8.45	7.47	5.50
Totals....	99.81	100.15	100.00	100.06	100.23	100.80	99.18	100.35	99.55	100.22	99.72	99.89	100.91	99.63

(*a*) FeO. (*b*) Sand, 4.90%; chemical combined, 38.94%. (*c*) Sand, 9.95%; chemical combined, 39.69%. (*d*) This sample in addition contains 2.04% FeO.

I., Briesen clay, average sample, Bischof;* II., Briesen clay, purest sort, selected with magnifying glass, Bischof;* III., Briesen clay, analyzed in laboratory of Paulshütte, Schoppinitz, O. S.: IV., *idem;* V., Briesen clay from Ferdinand shaft, analyzed at the experimental station of the Royal Porcelain Factory at Berlin, 1888;† VI., Briesen clay from Anton shaft;† VII., Saarau clay, Kerl;‡ VIII., Saarau clay, analyzed at Paulshütte, O. S.; IX., burned clay from Striegau; X., gray clay from Striegau; XI., dark-blue clay from Striegau, near Saarau; XII., clay from Grojec; XIII., kaolin from Saarau, analyzed at Paulshütte, O. S.; XIV., *idem*, by R. Schumann; XV., kaolin from Pentsch; XVI., kaolin from Ruppersdorf, near Strehlen; XVII., kaolin from Ruppersdorf, puddled and burned; XVIII., kaolin from Göppersdorf, dried at 100° C.; XIX., burned shale from Neurode, analyzed by Aron; XX., *idem*, by R. Schumann; XXI., raw Neurode shale, Bischof; XXII., shale from Rakonitz, Bohemia, analyzed by Aron; XXIII., *idem*, by Seger; XXIV., shale from Morgenroth mine at Schoppinitz; XXV., blue clay from Andenne, Bischof; XXVI., black clay from Andenne, Schmidt; XXVII., clay from Namur;‡ XXVIII., clay from Andenne, Bischof.

I.–VI. Briesen clay possesses a quotient of refractoriness of 60%, fuses with Seger cone No. 35, and is therefore exceeded by no other clay in infusibility; only the Neurode, Rakonitz, and other shales (*Thonschiefer*) and some puddled kaolins stand higher. A rational analyses of V. and VI. showed in the former, pure clay (*Thonsubstanz*) 99.07%, quartz 0.32%, feldspar 0.61 to 100%; and in the latter, clay 99.67%, feldspar 0.33 to 100%. This clay comes in stone-hard lumps; it is ground to 1 mm. size, wetted, and allowed to rot several months, after which it forms excellent retort material.

VII., VIII. Saarau clay, which comes in large wet clods (with 20 to 30% water), is a fat, strong-binding clay, but stands considerably behind the Briesen in infusibility and non-corrosibility with slags. About 14% of its silica is present as sand, the remainder being chemically combined.

IX., X., XI. The Striegau clay is plastic, but inferior in infusibility to the Saarau. It shrinks much in burning, as do both the Saarau and Briesen.

XII. Grojec clay is a non-shrinking material, which is mixed with the preceding varieties to counteract their contraction. This clay is always used raw. At moderate temperature it burns dense and firm; at white heat it fuses. It can only be used, therefore, in very small proportion.

XIII.–XVIII. These kaolins have been used as a substitute for the Grojec

* Bischof, *Die feuerfesten Thone*, S., 181. † Hecht, *Thonindustrie-Zeitung*, 22, May 26, 1888.
‡ Kerl, *Thonwaaren-Industrie.*

clay. They are quartzose and very refractory; to reduce their sand contents they are sometimes slimed. They are very lean and fragile. In the fire they shrink until their chemically combined water is driven out, but then expand on account of their high quartz content, and withstand very high temperatures without fusing, though on this account they are susceptible to changes of temperature. Their high silica tenor, however, as is the case with the Grojec clay also, makes them likely to slag with the basic material of the charge. On account of their leanness these kaolins contribute practically nothing to the binding of the mass. To use them raw also contravenes their utility, since they would serve their purpose much better if burned to chamotte, when more fat raw clay could be used, while the kaolin-chamotte would be a component promoting the infusibility of the mixture. The coarse grains of free quartz will still prove a defect in these kaolins, however, since they break under rapid variations of temperature, to which the retorts are subject in the furnace. The slimed kaolins, which seem best adapted to replace the binding clay, have proved entirely too lean.

XIX.–XXIV. The Neurode shales and equivalents have proved the best lean material for admixture in the retort mass in Upper Silesia, being cheaper than chamotte, and superior in infusibility and non-corrosibility with slags. Together with the shale, old retort material dressed clean from adhering slag is used. This material has undergone a complete change in the furnace through the action of the zinc vapor, which forms in it zinc spinel and tridymite. Investigations in the laboratory of the Paulshütte (Schoppinitz) showed the following percentages of zinc in old retorts: Light green, 6.45; light brown, 0.50; light blue, 8.55; dark blue, 9.00.

Degenhardt gives the following analyses of old retorts at Bethlehem, Pa.:

Color.	SiO_2.	Al_2O_3.	Fe_2O_3.	ZnO.	MnO_2.	CaO.	MgO.	Totals.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Blue.....	41.13	33.48	2.84	21.47	0.37	0.92	0.47	100.68
White.....	50.10	38.28	3.42	6.10	0.41	1.13	0.73	100.17

Old retort material is of limited refractoriness, and muffles containing a high percentage of it bend and break, and those in which it is replaced by good chamotte are significantly stronger. The use of this old material consequently limits the temperature in the Silesian furnace below the point used in the Rhenish-Westphalian, where but little of it is employed. The Neurode burned shale, which is used in Silesia as a partial substitute for the old material, to increase the refractoriness of the mass, and work against softening, is an excellent material, fusing with the highest Seger cone, No. 36. The particles of shale hold out amazingly in the mass, only their surfaces are acted upon chemically, while their interiors remain unaltered. Even after being used several times, first by direct admixture and subsequently in the old material, they are still highly refractory.

XXII. The Rakonitz shale is equal in quality to the Neurode, but is more expensive.

XXIII. A shale from the coal measures at Schoppinitz, which proved ill adapted as a substitute for the Neurode on account of its high fluxing components.

XXV.–XXVIII. The clay from Andenne, near Namur, Belgium, occurs as two

distinct kinds: The blue to black-blue, less silicious, extraordinarily fat and binding; the light colored, very silicious, less fat. The former is used raw, and the latter (for lean material) burned. The Andenne fat clay corresponds to Seger cone No. 33.

The average mixture of clays used in Upper Silesia, varying somewhat at each works, is: Fat clay (raw), 2 parts by measure; sandy clay (raw), 1 part; lean material (old retorts, shale), 5 parts. As fat clays are used the Briesen, Saarau, and Striegau, each alone or a mixture of two or three; as sandy clay the Grojec or Silesian kaolin, or a mixture of both; as lean material old retorts and shale in varying proportions. The clays are used in 1 to 3 mm. grains; the old material and shales as a mixture of coarse (up to a maximum of 9 mm. diameter) and fine, just as is delivered by the Chilean mill, which is commonly employed for grinding them.

The coarse chamotte increases the mechanical strength of the raw muffles, which in view of their weight is important. The coarse grains also help the muffle to withstand rapid changes of temperature, and makes them less liable to chemical corrosion owing to the less surface present for attack. The fine chamotte increases the density of the mass and its heat conductivity. Chamotte fines would not serve as the sole lean material, however, since then the muffle would have a tendency to crack.

In the Belgian-Rhenish-Westphalian practice anthracite or coke dust (about 10%) is now almost always used in the mass. Opinions as to the action of this vary, but whatever it is there is no doubt that coke is an excellent lean-making material, not shrinking even at white heat, and to it is also due that so much fine chamotte may be used without danger. As to the use of coke Dr. Bischof says:* "Anthracite, coke, and even charcoal dust, are more easily ignited than graphite, yet so long as they are protected from burning by a clay envelope they increase the refractoriness of the clay and counteract the tendency to crack." Obviously the coal must be prevented from burning and therein the hydraulic presses are important. Moreover the retorts are glazed outside with a mixture of fat clay, sand, and rock salt (or soda), while inside a natural slag-glaze is soon formed.

The muffle mixture varies considerably among the Belgian-Rhenish-Westphalian works. One uses 40 parts by measure of dark, fat, raw Belgian clay; 50 of light-colored, sandy Belgian clay burned to chamotte; and 10 of coke. Another, 36, 54, and 10 parts of the same substances respectively. Both are very durable. A third uses 30 parts of fat, Belgian clay (raw); 10 of sandy Belgian clay (raw); 50 of old retorts, and 10 of coke, which has less durability on account of the old material, while the use of the sandy clay raw instead of in the form of chamotte is a mistake, even though it, like the old material, reduces the cost of the mass. A Belgian works used 10.5 parts Tahier clay (raw); 10 of the same burned; 8 of old material; 1 of quartz; and 0.5 of coke. The amount of the last is too little to be of influence.

Zinc Smelting in Wales.—Data relating to the practice in the metallurgy of zinc in Wales are furnished in the report for 1895 by the English Crown Spelter Company, which is one of the most enterprising of the Welsh producers. The amount of ore smelted was 13,349.695 long tons, of which 8752.606 was cala-

* *Die feuerfesten Thone.* S., 206.

mine, 4020.336 blende, 373.800 zinc ashes, and 202.953 flue dust. The make of metal from this was 5727 long tons. The consumption of coal on the grates was 24,802 tons, and in the retorts 6886 tons, the former being an average of 4 tons 728 lbs., and the latter 1 ton 448 lbs. per ton of metal made. Of retorts, 12,515 were used, an average of 2.18 per ton of spelter against 2.31 in the previous year, while the breakage of condensers amounted to 16.7 per ton of spelter against 16.9 in 1894. Further progress was made in changing the old Welsh furnaces into gas furnaces, and most of those in use now are of the latter type.

The English Crown Spelter Company operates mines and dressing works in Italy, from which it derives a part of its ore supply, purchasing another part in the open market. The items of cost in its annual report are not segregated so that it is possible to arrive exactly at the cost of smelting. It is stated, however, that at an average price of £14 11s. 6d. obtained per ton of metal sold at Swansea there was a good margin of profit over the cost of smelting, which is attributed to the improved yield of metal from the ore and the smaller quantity of coal consumed and the less price paid for it. Although it is not so stated in the report, the increase in salvage of metal and decrease in fuel account is undoubtedly due to the introduction of the gas-fired furnaces.

The mines in Italy produced 8296 tons of raw calamine, which yielded 6294 tons net, dry weight, at Swansea, assaying 47.88% Zn. The dressing works produced 1971 tons net, dry weight, at Swansea, assaying 44.62% Zn. The sum, 8265 tons, averaged, therefore, 47.08% Zn, containing 3891.23 tons of metal. This ore cost $12.305 per ton of 2240 lbs. delivered at Swansea, or 1.167c. per lb. of metal in the ore. The price realized for spelter at Swansea was $70.92 per ton, or 3.166c. per lb. The profit on the year's business was $49,339.96, or $8.615 per ton, equal to 0.385c. per lb. The cost of smelting, including loss of metal in treatment, rents, taxes, interest, depreciation of plant, and all general expenses, was consequently 1.614c. per lb. of spelter produced from ore mined by the company.

Electrolytic Methods.—The chief progress in the metallurgy of zinc recently has been in the direction of electrolytic production, for which numerous processes have been invented. These are summarized in an article by B. Neumann,* who states that the cost of electrolyzing zinc sulphate and chloride solutions with a current of 2.5 volts is 18.96 marks per 100 kgms. of zinc when steam power is used, and 13.33 marks with water power, which leaves a very small margin for roasting, leaching, etc. Moreover, the leaching of basic zinc sulphates formed in the roasting is not easy, and in any case iron goes into solution, and involves difficulty in removing it, since FeO, unlike Fe_2O_3, is not thrown down by ZnO, while it cannot be completely peroxidized even by blowing air through the hot solution for 24 hours.† This solubility of interfering substances is an objection to Luckow's process, in which the roasted ore is made the anode. Given a solution of zinc sulphate, the formation of H_2SO_4 at the anode soon causes a rough cathode deposit, and a co-deposition of hydrogen; but if the solution be basic a zinc sponge comes down, which prevents the rolling of the metal and oxidizes on remelting. The same sponge is apt to be formed if the

* *Chem. Zeit.*, 1896, XX., pp. 36–38. † *Ibid.*, 1895, XIX., p. 221.

current be less than 150 ampères per sq. meter. Copper, antimony, and arsenic interfere with the precipitation.

After roasting the ore Ashcroft leaches the zinc with a ferric chloride solution; Heinzerling with magnesium chloride;* Höpfner with carnallite; and others with calcium chloride. Alkaline zincate solutions have been recommended by Kiliani, Burghardt, and Höpfner; Nahnsen adds K_2SO_4 or $MgSO_4$ to the solution of $ZnSO_4$, but in the trial this process had at Lipine, O. S., the cost proved to be prohibitive; Lorenz proposes the electrolysis of fused $ZnCl_2$, which presents great technical difficulties in the way of evaporating the solution, producing anhydrous salt, and sufficiently durable material for the furnace.

Following are some of the patents issued in this line in 1896:

C. A. Burghardt, Manchester, and G. Riggs, Eccles (English Patent No. 19,934, October 23, 1895), treat roasted ore with a concentrated solution of ammonium carbonate, sesquicarbonate, or bicarbonate, or mixtures of these with ammonia, whereby oxides of zinc and copper are dissolved. The solution drawn off is heated to 40° C., iron precipitated by hydrated oxide of tin and prolonged agitation, and copper thrown down subsequently by metallic zinc. The zinc is then precipitated by electrolysis in wooden vats at a temperature of 40° to 50° C., oxygen being liberated at the positive pole. The oxide of tin is recovered by dissolving the accompanying oxides of iron with H_2SO_4. The solutions are kept covered with a layer of paraffine oil to prevent loss by volatilization.

C. Höpfner (English Patent No. 8449, April 29, 1895,) roasts, dissolves ZnO with H_2SO_3, neutralizes excess of acid, decomposes $ZnSO_3$ with $CaCl_2$, obtaining $ZnCl_2$ in solution, and $CaSO_3$ precipitated; the latter is soluble in H_2SO_3 and is available for wood-pulp manufacture. Zinc is precipitated by electrolysis from the chloride solution.

Lyte proposes to leach with dilute H_2SO_4, convert zinc to chloride by addition of $CaCl_2$, draw off the solution of $ZnCl_2$, evaporate to dryness, volatilize and condense to free from impurities, melt, and electrolyze at a temperature somewhat over the melting point of zinc, using coal as anode and molten zinc as cathode, tapping off the zinc automatically, and collecting the chlorine.

R. O. Lorenz (English Patent No. 25,074, December 24, 1894) proposes to obtain zinc from mixed lead and zinc ores by converting the metals into chlorides, and subjecting the chlorides to fractional electrolysis at a temperature from 450° to 680° C.

Siemens & Halske (English Patent No. 1575, January 22, 1896) dissolve zinc oxide in neutral aluminum sulphate, $Al_2(SO_4)_3+18\ H_2O$, electrolyze between insoluble anodes at 3 volts pressure, whereby zinc is deposited at cathode, and oxygen at anode, while the basic aluminum sulphate is reconverted to neutral sulphate for subsequent use. The bath is best prepared by dissolving the ZnO in a warm 10 to 15% solution of neutral aluminum sulphate.

With all this enterprise in investigation of a long-time, puzzling problem some practical results bid fair to be reached soon. The company owning the Ashcroft process, which on an experimental scale is claimed to have given good results, is building works (now nearly completed) at Newcastle, New South Wales, to treat Broken Hill ore, while the Smelting Company of Australia, Limited, is building

* *Chem. Zeit.*, XVI., 1892, p. 1864.

works at Illawarra, New South Wales. At the latter it is proposed to electrolyze a zinc sulphate solution, which Siemens & Halske, who are the undertakers of the works, guarantee to do at a cost of 72s. per ton of zinc, or 92s. including royalty, while zinc-white will be produced by the Marsh & Storer process, namely, precipitation from the sulphate solution by means of magnesia. Lately it has been announced that Dr. Höpfner has made arrangements with Brunner, Mond & Co. for the exploitation of his process in England. This process involves the electrolysis of a solution of zinc chloride, wherein chlorine is set free and is made available as a by-product. It is expected that the works for this will be in operation early this year.

Electrolytic zinc was produced by the Schlesischen Actiengesellschaft für Bergbau und Zinkhüttenbetrieb at Lipine, O. S., its product assaying 99.98% Zn and 0.01 to 0.03% Pb, and selling for 24.50@25.50 marks per 50 kgms. f.o.b. Breslau; electrolytic zinc from works at Duisburg am Rhein was also on the market, this being produced directly from ore, whereas that of the Silesian is obtained by refining crude spelter. The Duisburg works turn out about 90 tons per month, and it is said the output is to be increased this year. The process employed is the invention of Dieffenbach, but as to its nature nothing definite has been made public. It is, so far as known, an electrolysis of a solution of zinc chloride, which is obtained by leaching ore roasted with salt, and according to a German technical paper* its operation is so successful that only 0.5% Zn is left in the residue, the material treated being pyrites cinder containing 15% Zn.

For a consideration of the more important chemical methods (using the term "chemical" as distinctive from the "electrolytic" and "fire" methods) reference must be made to a paper by S. H. Emmens in Vol. I. of THE MINERAL INDUSTRY; while the combined smelting-volatilization method in use at Cañon City, Colo., is described by F. L. Bartlett in a subsequent article in this volume.

Assay of Zinc Ores.—This is usually made in the United States by the ferrocyanide method; in Europe it is done by the Schaffner sodium sulphide method in more or less modified form. The latter is described by H. Nissenson and B. Neumann,† who call attention to the numerous details that must be observed to insure accurate results from it. The sodium sulphide titration is made with an ammoniacal solution, using as indicators ferric hydroxide suspended in the solution or acetate of lead paper.

Determination of Sulphur in Blende.—To determine the sulphur contained in roasted blende as sulphide, J. G. Heid‡ recommends the following procedure:

Boil 1 gm. in an Erlemeier flask of about 200 c.c. capacity with 50 c.c. of a 5% sodium acetate solution; let it settle and decant the liquid above the precipitate through a fine filter; repeat this operation and boil in the same manner twice with distilled water. By this means all the sulphur occurring as sulphates is extracted. The filter is then added to the precipitate remaining in the flask, and treated with fuming nitric acid to transform the sulphur into sulphuric acid, the sulphur being determined as barium sulphate in the usual manner.

In seven samples of roasted zinc blende the following results were obtained:

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* *Zeits. Elektrochem.*, 1896, III., p. 244.

† *Chem. Zeit.*, 1895, pp. 1624-1626; *Journ. Soc. Chem. Ind.*, January, 1896, p. 52.

‡ *Engineering and Mining Journal*, Aug. 22, 1896.

	I.	II.	III.	IV.	V.	VI.	VII.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Total sulphur	2.15	1.25	1.80	0.95	2.75	1.10	1.65
S as sulphide	0.42	0.55	0.13	0.26	0.36	0.07	0.12

USES OF ZINC IN THE ARTS.

Sheet Zinc for Roofing.—A series of valuable articles on "The Applications of Sheet Zinc for Roofing and Other Purposes," by Prof. W. H. Seamon, was published in the *Engineering and Mining Journal,* Oct. 24, 31, Nov. 7, 14, 1896. He estimated the consumption of spelter in the United States in 1892 at 78,040 tons, divided as follows: Galvanizing, 35,000; brass manufacture, 20,500; sheet metal, 15,500; desilvering lead, 3500; miscellaneous and unaccounted for, 3540. In Europe more than 100,000 tons of sheet zinc are used annually for roofing. The business of zinc roofing is conducted there by the various zinc manufacturers, and partly by roofing companies. The sheets are usually corrugated and stamped at the rolling mills, which are ready to execute special orders, though some of the roofers cut and stamp their own sheets. The usual lengths of the sheets are 7 or 8 ft., but even 10 ft. may be had by special order, and they are from 2 ft. 8 in. to 3 ft. in width. For roofing purposes the following thicknesses are recommended by the Vieille Montagne Zinc Company:

Gauge.	Thickness.		Weight Per Square Foot.		33 in. by 72 in.		36 in. by 84 in.		36 in. by 96 in.	
					Weight of Sheets		Weight of Sheets		Weight of Sheets	
	In.	Mm.	Lbs.	Ozs.	Lbs.	Ozs.	Lbs.	Ozs.	Lbs.	Ozs.
No. 13	0.029	0.740	1	....	19	1/2	22	3 1/2	25	6 1/2
No. 14	0.032	0.820	1	2 1/2	21	1 1/2	24	9 1/2	28	2
No. 15	0.038	0.950	1	5 1/2	24	7 1/2	28	8 1/2	32	10
No. 16	0.043	1.080	1	8 1/2	27	13 1/2	32	7 1/2	37	2
No. 17	0.048	1.210	1	11 1/2	31	2	36	5 1/2	41	8 1/2
No. 18	0.053	1.340	1	14 1/2	34	8	40	4 1/2	46	1 1/2

The various clips and fastenings may be had direct from the zinc rollers; the only labor required to be done by the roofer is that of bending the lateral edges of the sheets, and the actual operations of laying them.

Professor Seamon estimates the original cost and maintenance of roofs of different material as follows:

	Zinc.	Tin.	Slate.	Galv. Iron.	Tiles.	Lead.	Copper.
Original cost per square, including laying	$11.75	$8.85	$10.50	$5.75	$12.50	$21.00	$30.00
Repairs in 30 years	1.15	11.25	3.50	11.50	2.50	.....	.....
Interest, 6% on cost, 30 years	21.15	16.75	19.90	11.35	22.50	37.80	54.00
Interest, 6% repairs, 15 years	1.00	10.12	3.15	10.35	2.25	.....	.....
Totals	$35.05	$46.87	$37.05	$38.95	$39.75	$58.80	$84.00

In the item for repairs for the tin roof a life of 30 years has been assumed, providing it is painted every two years at a cost of 75c. a square. In some localities it need only be painted every three years. No allowance has been made,

however, for the soldering of the tin roof, usually necessary from time to time; resoldering constitutes all the repair necessary on zinc roofs. The differences are more marked than the figures indicate, for at the expiration of 30 years the tin roof, as well as that made of galvanized iron, must be renewed; the slate roof may, or may not be good for another 30 years, but the zinc roof will be good for 50 more at least. Furthermore, the sheet zinc has a direct value as old metal when it is necessary to replace it, or in case of demolition of the building, which is not possessed by tin plate, galvanized iron, slates or tiling. Another important advantage over slate and tile roofs is that in many buildings the roof framing can be made lighter.

Electro-galvanizing.—Three processes for the electro-deposition of zinc upon iron are now in use in Europe—the Cowper-Coles worked by Watson, Laidlaw & Co. at Glasgow, Alexander's worked by Seelhorst & Werner at Rothenfeld, Hanover, and Richter's which is worked at Witkowitz. These were discussed by O. Vogel in a paper read at a meeting of the Eisenhütte Düsseldorf.* The advantage of the electrolytic method of galvanizing as compared with that commonly used are summarized as follows: A more uniform and thinner coating, which is nevertheless sufficiently protective; no loss in hard zinc and dross; and applicability to complex forms. The chief disadvantages are the higher cost of labor and more space required, a works for treatment of 10,000 kgms. of sheet per diem needing 200 sq. meters of ground. Some difficulty has been experienced in rendering the galvanized surface quite clean. Seelhorst & Werner claim to be doing this, however, and state the cost of galvanizing at their works to be 5 pf. per sq. meter against 50 to 60 pf. by fusion method.

The electro-deposition of zinc is described fully by S. Cowper-Coles in an article (illustrated by drawings) in the *Journ. Soc. Chem. Ind.*, June, 1896, p. 414. He states that the electro-deposited zinc on iron or steel withstands the action of copper sulphate (the test for telegraph wire) as well, if not better, than the ordinary galvanized wire. The chief disadvantages of electro-zinking, in his opinion, are its slowness, trouble experienced in coating articles of irregular shape, and the unfavorable appearance of the coating laid on in this manner, which lacks brilliancy, having an "unhealthy non-metallic look."

Zinc Grey.—The production of zinc grey or zinc dust in 1896 in Upper Silesia, which is the chief supply of this substance, amounted to 1,500,000 kgms. This was partly consumed in Germany and the remainder exported to England, France, Spain, Italy, India, Japan, Scandinavia, and America, demand being active, especially during the latter part of the year. Zinc dust is used extensively as a reducing agent in industrial organic chemistry, and to less extent in preparation of a paint for coating iron, as a reagent in several electro-metallurgical processes, and to precipitate gold and silver from cyanide solutions.

Cadmium.

A great increase in the demand for cadmium made the production of this metal a matter of some importance in 1896. Up to the middle of March the price had been steady at the low figures of 6.00@6.75 marks per kgm., when suddenly a sharp demand sprang up through the increasing use of a new alumi-

* *Journ. Soc. Chem. Ind.*, 1896, p. 122.

num-cadmium alloy, and prices rose to 14@20 marks per kgm., depending upon the size of the order and the quality of the metal. This was a higher quotation for cadmium than in many years, and several works which had abandoned the production of the metal as unprofitable took it up again. A great increase, however, does not seem probable, since the supply of cadmium-bearing zinc ores is not large. The cadmium of commerce assays 99.50 to 99.75% Cd. Besides the new alloy referred to it is used in the manufacture of fusible metals, and solder for aluminum; its sulphide is employed extensively as a pigment; and a solution of its chloride is used in soldering zinc.

PRODUCTION OF CADMIUM IN UPPER SILESIA.

Year.	Kilograms.	Val. Per Kgm (a)	Totals.	Year.	Kilograms.	Val. Per Kgm (a)	Totals.
1881.....	3,000	$2.30	6,899	1889.....	5,138	$0.95	4,876
1882.....	4,000	2.17	8,684	1890.....	4,158	0.88	3,652
1883.....	2,500	2.15	5,384	1891.....	2,849	0.88	2,517
1884.....	2,750	2.10	5,789	1892.....	3,200	0.89	2,850
1885.....	3,190	1.97	6,275	1893.....	5,285	1.04	5,461
1886.....	4,960	1.85	9,150	1894.....	5,952	1.04	6,186
1887.....	7,321	1.66	12,124	1895.....	6,847	1.39	9,509
1888.....	4,796	1.19	5,714	1896.....	11,306	1.87	21,186

(a) Values calculated from marks at ratio of $1=4 marks. These values are reported for the metal at the works.

The New York Spelter Market in 1896.

During the past year the course of the spelter market has been subjected to violent fluctuations, more or less due to what may justly be called artificial circumstances. For a number of years past the position of the smelters of zinc ores has not been a desirable one; they have had to struggle hard in order to get meager profits. The competition of the different smelters when buying their ores has, since the growth of the industry, been so keen that only the best-managed works have been able to show some returns for the capital invested, which in any case were not what they might or ought to have been. Very often for long periods the metal contained in the ore has been worth more, after adding smelting, refining, and incidental charges, than the refined spelter could be marketed at, and, naturally, those interested have been considering for a long time past what steps, if any, could be taken to put the business on a better paying basis. Various attempts to this effect had been made in the past, but always failed, until in January last a number of smelters were united, as previously described.

At the beginning of the year large stocks had been accumulated from former production, which weighed heavily on the market. Besides, consumption was not brisk, and export, on account of the lower prices in Europe, not feasible; consequently the new combination determined on the policy of closing down a number of the newly acquired works and concentrating the smelting to others in order to reduce expenses. This policy would necessarily have had the effect of relieving the market (which would doubtless have righted itself before long) of the large stocks, but for this the new company would not wait, and at once exercised its power to raise prices, which procedure was not at all to the liking of the consumers, who from that time on became rather indifferent, since they could

always supply themselves, to a fair extent, from the independent smelters; besides, during the spring and summer months galvanizing trade fell off to an alarming extent, and consumption was less than it had been for a long time. Thus, in spite of the heavily reduced production, stocks did not decrease in due proportion, and this led to a considerable fall in prices.

While early in January 3½c. St. Louis was about the highest price obtainable, the market soon hardened when it became known that a new company was being formed, and about the end of the month 3¾c. St. Louis had been reached, which advance made further progress during February and March, when 4c. St. Louis and 4¼c. New York were established. By the end of April signs of weakness began to appear; more furnaces had to be closed down, and at the beginning of May 3¾c. St. Louis and 4c. New York were again the outside figures, and during that and the next two months a very irregular and flat tendency prevailed.

In the meantime the expansion of trade in Europe and the large demand for galvanized iron from the East had brought about a considerable advance in prices there. Before American spelter could be exported, however, prices here had come down to a still lower level. During July and August consumption was very poor; prices fell slowly, and at the beginning of September 3¼ to 3.3c. St. Louis was accepted, and then heavy sales were made for export, partly for immediate delivery and partly for delivery up to the end of the year. In all, from 6000 to 7000 tons were thus marketed abroad. Stocks were cleared out, prices rose, and by the beginning of October 3½c. St. Louis was freely paid.

During October home consumption became decidedly better, followed by such a large demand immediately after the general election that it was hardly possible to ship out the spelter quickly enough. Stocks were depleted, a great many furnaces were idle, and during the scramble for spelter, for immediate shipment, prices shot up very quickly, and 4c. St. Louis was again reached, a figure considerably above the parity of values abroad. Part of the spelter contracted for export, which had not been shipped, was held back; one after another of the furnaces was lighted, so that very soon the scarcity passed away, and with trade becoming very much quieter toward the end of November, the market suddenly became quite flat. At the end of that month 3⅞c. was accepted, followed in December by a further decline, and by the end of the month 3¾c. St. Louis and 4c. New York were the ruling prices.

AVERAGE MONTHLY PRICES OF SPELTER IN NEW YORK, IN CENTS PER POUND.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1892....	4.69	4.62	4.89	4.68	4.79	4.71	4.78	4.69	4.53	4.41	4.47	4.40	4.63
1893....	4.39	4.33	4.28	4.38	4.41	4.27	4.13	3.89	3.69	3.68	3.65	3.80	4.08
1894....	3.56	3.85	3.89	3.62	3.47	3.40	3.43	3.38	3.44	3.45	3.36	3.43	3.52
1895....	3.28	3.20	3.23	3.30	3.50	3.65	3.75	4.15	4.30	4.10	3.55	3.49	3.63
1896....	3.75	4.03	4.20	4.09	3.98	4.10	3.97	3.76	3.60	3.72	3.99	4.14	3.94

THE LONDON SPELTER MARKET IN 1896.

THE new year opened with spelter in a condition of lethargy, due partly to the stoppage of building operations, rolling, etc., common to the season, and partly to the general depression prevailing in the various metal markets at the

close of the old year, with the unsatisfactory political outlook referred to in our last report. Demand was practically dead, and prices continued to recede, until after the middle of the month, when £14 2s. 6d. was accepted for ordinaries—a fall of about 7s. 6d. from the opening value. The resumption of work at manufacturing centers and the low level of prices then began to make themselves felt, and a reaction set in, which soon carried the price up to the opening figure, £14 10s., and as very little spelter was forthcoming—producers everywhere holding back—values continued to stiffen during the greater part of the month of February. At first there was most activity, and a higher range of values on the continent, but the London market soon followed suit, and some large lines were done with consumers in the midlands and other parts of the provinces, galvanizers and yellow metal people being very busy. When £15 2s. 6d. had been reached, about the middle of February, the buying slackened off, all the more so as people had to a certain extent covered their wants; and the tone then became less buoyant, so that the month closed quiet at £14 17s. 6d.

March began with a continuance of the depression, but the market quickly threw this off, the absence of anything like large quantities for sale tending to give a tone of stability, and when, toward the middle of the month, a better consumptive demand set in, prices at once rose and finally closed firm at £15 10s. for ordinaries. An important element of strength at this time and during the greater part of the year was found in the very large scale upon which the shipments of galvanized iron were proceeding; those for February exceeded by about 9000 tons those of the same month in the previous year, which of course meant a correspondingly larger consumption of spelter.

When April opened the smallness of stocks still remained a feature, and there was no pressure to sell, but, on the other hand, consumers were holding aloof, and though at the Birmingham quarterly meeting there was a little more buying, prices were to a considerable extent spoiled by second-hand lots. On the continent, however, demand was active and much higher prices were paid there than in England; in fact, up to the parity of £16 10s. ex-ship was recorded, the value of ordinaries delivered at Birmingham being simultaneously about £16 7s. 6d. The exhaustion of the second-hand parcels which had been pressing upon the market enabled values here to reassert themselves, and the scarcity of spot stuff being accentuated, values were enhanced, the month closing in very firm tendency at £16 7s. 6d. to £16 10s. for ordinaries, with nearly all the English makers out of the market, and likewise the Germans, while the Belgian makers were only sellers at very full prices. About the middle of the month, the returns of galvanized iron shipments were published and showed 62,645 tons against 41,553 tons for the first quarter of 1895.

The advance made further progress throughout May, early in which month there was a good demand, especially for May and June delivery, for which positions there was great scarcity. Many dealers and consumers who had hitherto abstained from buying, in the hope of seeing the market relapse, now began to show anxiety to cover themselves, and as the depletion of prompt stuff grew more pronounced, the value rose steadily, £17 10s. up to £18 10s. being successively paid for Birmingham delivery, and the month closed at about this price, with every symptom of strength; demand good, supplies scarce; and with gal-

vanized iron also in greater request, while the shipments of these goods continued to show a large increase over the corresponding periods of last year. At the beginning of June £18 15s. ex-ship London was paid for an isolated parcel, while very little stuff was obtainable earlier than September. This price, however, proved to be the highest point of the year; the high range of values rendered possible the sale of American spelter, which ere long began to press on the market, relieving the squeeze. The market accordingly closed quiet at £18 to £18 2s. 6d. ex-ship London for ordinaries.

July was a month of drooping values, the depressing effect of the sales of American spelter being increasingly felt. Consumers bought only sparingly and ere the month terminated the value of ex-ship London spelter had fallen to £17 11s. 3d. The conditions during August were equally unfavorable, and about the middle of the month prices were much depressed by the sale of second-hand parcels, as low as £16 15s. being recorded. When these were absorbed the aspect of things grew somewhat brighter, and, with a little more life in home demand, added to a certain amount of influential buying in this country, we witnessed a recovery to £17 5s. ex-ship, the month closing rather below that figure, in steady tone, with a moderate demand.

In September the withdrawal of the support above alluded to plus a falling off in the shipments of galvanized iron, had the natural effect upon the market, which declined rapidly until £16 had been accepted. The lower range of values, together with the general perception of the fact that supplies of early spelter were growing scarce again, induced a rally to £17 2s. 6d., the month closing rather lower than that figure.

Scarcity of spot spelter was the opening note of the market in October, and fancy prices were paid for that position, while forward was neglected. Even for early delivery, however, the demand was not extensive, and the quarterly meeting of the trade in Birmingham on October 8, revealed a stagnant market, with general complaints of unsatisfactory trade, while prices of spelter had declined until, in the second week, £16 5s. ex-ship was taken. Rather later in the month, things brightened up considerably, and the ensuing demand, contrasted with the paucity of supplies, and with the absence from the market of the sellers of American spelter, brought about an advance to £17 2s. 6d. per ton ex-ship. November brought a development of the firmer feeling in sympathy with the general strength of the metal markets after the Presidential election in the States, and the strength of the market was increased by the facts that producers were mostly sold out and that American spelter was being bought for America, which had ceased to be a seller. The buying by consumers and speculators sent the value up to £18 2s. 6d. in the third week, but the subsidence of the demand was attended by a relapse to £17 15s. ex-ship, the closing value of ordinaries, while special brands are quoted at the usual rate of about 2s. 6d. higher.

During December the market showed hardly any change, although the tendency was somewhat firmer at the end of the year with prices ruling at about £18. While production in the States seemed to hold its own, buying for American account ceased entirely, as manufacturers over there, especially galvanizers, were rather disappointed regarding the amount of business they expected to do after election.

IMPORTATIONS OF ZINC INTO ENGLAND AND PRICES AT LONDON. (*a*) (IN LONG TONS.)

Sheet Zinc.	1887.	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.
Importations	Tons. 19,555	Tons. 18,262	Tons. 19,261	Tons. 17,542	Tons. 20,157	Tons. 19,006	Tons. 18,842	Tons. 18,452	Tons. 19,363	Tons. 21,003
Highest (*b*)	£ s. d. 19 5 0	£ s. d. 20 10 0	£ s. d. 20 2 6	£ s. d. 27 0 0	£ s. d. 26 0 0	£ s. d. 25 0 0	£ s. d. 20 10 0	£ s. d. 18 10 0	£ s. d. 18 0 0	£ s. d. 20 5 0
Lowest (*b*)	16 0 0	16 17 6	18 15 0	23 5 0	25 0 0	21 10 0	19 10 0	17 7 6	16 2 6	17 5 0

Spelter.	1887.	1888.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.
Importations	Tons. 56,187	Tons. 61,045	Tons. 56,842	Tons. 56,205	Tons. 58,483	Tons. 52,793	Tons. 56,926	Tons. 52,897	Tons. 62,523	Tons. 76,663
Highest (*c*)	£ s. d. 21 10 0	£ s. d. 22 0 0	£ s. d. 24 5 0	£ s. d. 25 10 0	£ s. d. 24 0 0	£ s. d. 22 15 0	£ s. d. 18 2 6	£ s. d. 16 15 0	£ s. d. 15 10 0	£ s. d. 18 10 0
Lowest (*c*)	14 0 0	14 10 0	16 10 0	19 15 0	22 5 0	17 15 0	16 15 0	14 0 0	13 10 0	14 10 0

(*a*) From W. T. Sargant & Co.'s Annual Circular. (*b*) Prices realized for London rolled zinc sold in public sales. (*c*) Quotations are for Silesian spelter.

THE BRESLAU SPELTER MARKET IN 1896.

THE price of spelter f.o.b. Breslau in 1896 ranged as follows: January, 14.25—14.15—14.40 marks per 50 kgms.; February, 14.75—15.00—14.80; March, 15.00—15.20; April, 15.90—16.00; May, 17.50—17.30—17.60; June, 17.80—18.00; July, 17.30—17.60—17.50; August, 17.00—17.20—16.90; September, 16.40—16.25—16.50—17.00; October, 17.00—16.50—16.85; November, 17.00—17.75—17.60; December, 17.40—17.80. The quarterly averages reported by the royal mines office at Breslau are as follows:

	1892.	1893.	1894.	1895.	1896.
	Marks.	Marks.	Marks.	Marks.	Marks.
First quarter	20.00	16.00	14.50	12.50	14.74
Second quarter	21.00	16.00	14.00	13.00	15.12
Third quarter	19.00	16.00	14.00	13.50	17.01
Fourth quarter	17.00	15.50	13.00	13.50	17.28

Zinc Refining.

By Bruno Kerl.

The crude zinc obtained in the reduction of zinc ore, which process has been fully described by Mr. W. R. Ingalls in The Mineral Industry, Vol. II., pp. 635–669, contains more or less foreign components, according to the purity of the ores, and both former and more recent investigations* have shown that they frequently exert an injurious influence on the quality of the zinc. Results vary widely, however, as regards the extent of their influence, and still further experiments are needed.

Lead, the impurity most frequently found, and the one which exerts a special influence upon the sheet-rolling, is often present in crude zinc in quantities of 0.3 to 3%. A tenor of 1½% will permit the rolling of the zinc without cracking the sheets, but will unfit them for many purposes. With 3% lead the metal is not rendered entirely useless for rolling, but the sheets become softer. For casting zinc, however, the presence of lead is less hurtful, if it separates uniformly on cooling, but zinc with a noticeable quantity of lead is not suitable for the manufacture of brass of a good quality, such as is required for cartridge shells and for graphical purposes.† While sheets of pure zinc are smooth and glossy, those containing lead are gray and rough, and are attacked by acid with greater difficulty and more unevenly, the lead separating as a sponge. To ascertain whether zinc is suitable for graphical purposes, Augurer recommends brightening a small spot with a finely-ground scraper and examining the fresh surface with a magnifying glass. The zinc appears as ashy, star-like scales, while lead is indicated by the presence of bluish flecks. Behrens‡ enters into explanatory statements on the microscopical behavior of pure zinc and of that containing lead. For graphical purposes a pure zinc must be used, or, if zinc contaminated with lead is the only metal obtainable, it should be permitted to remain undisturbed for some time in the refining furnace before being ladled out. Ordinary zinc sheets, plain or corrugated, are used especially for roofing, architectural purposes, water tanks, etc.

Iron is an impurity also frequently found in zinc; it is said that 1% of it makes the latter less fluid, harder, more brittle, and impairs its strength. The percentage of iron in zinc, however, seldom exceeds 0.2%; and in refining can be diminished to 0.03 or 0.02%. While, according to Jensch,§ spelter obtained from blende and calamine rarely contains more than 0.02% iron, the zinc produced from the dust obtained from the Upper Silesian iron blast furnaces (which contains 17 to 22% zinc and 23 to 25.5% ferrous oxide) has a tenor of 0.71% iron. That zinc can hold a still greater percentage of iron is shown by the hard zinc

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* *Karsten's Archiv für Mineralogie*, etc., 2 Reihe, Vol. XVI., Part II., p. 597 (Karsten); *Berg- u. Huttenm. Ztg.*, 1851, p. 198 (Eliot and Storer); 1873, p. 288 (Thum); *Ann. d. Chem. u. Pharmac.*, Vol. XX., p. 179 (Wittstein); *Dingler's Polyt. Journ.*, Vol. CXVI., p. 248.

† Bertha zinc from Pulaski County, Va. (The Mineral Industry, Vol. II., p. 669) contains, besides 0.019% iron, only traces of lead; zinc from the Passaic Works, N. J., 0.027% lead and 0.020% iron; Missouri zinc, 0.006% lead. The zinc of the Vieille-Montagne Company, made from zinc silicate, and also several qualities of Spanish and electrolytic zinc, are extremely free from lead, some containing not more than 0.00002%.

‡ H. Behrens, *Das mikroscopische Gefüge der Metalle u. Legirungen*, Hamburg and Leipsic (Verlag von L. Voss), 1894, p. 56.

§ *Zeitschr. f. Angew. Chemie*, 1890, p. 13.

obtained in the refining of the crude zinc, which segregates between the lead and zinc, while in the galvanizing of iron an alloy of zinc with 4% iron separates in the kettles. Herapath found as much as 7% iron in zinc taken directly from the condensers of the distilling furnace. The greater the iron percentage the more readily the zinc is dissolved in acids.

The following are some of the iron percentages found in spelter. United States: Glendale, Mo., 0.0095; Cherokee, Kan., 0.0315; Nevada, Mo., 0.0546; Pittsburg, Kan., 0.0523; La Salle, Ill., 0.0475; Lehigh, N. J., 0.041; Passaic, N. J., 0.020; Bethlehem, Pa., 0.0405. Upper Silesia, 0.0238 to 0.0280. Belgium: Bleiberg, 0.101. Austria: Cilli, 0.0253; Sagor, 0.010 to 0.032; Johannisthal, Krain, 0.018. Rheinland: Birkengang, near Stollberg, 0.022.

Cadmium, if present in large quantity, causes brittleness and a fine-grained fracture, although zinc with 15% cadmium might still be rolled into sheets 2 mm. thick without cracking. The trifling quantities of cadmium present in commercial zinc, however, have no injurious influence upon its physical properties. Zinc from Johannisthal, in Krain, contained 0.069% cadmium; from Lehigh, N. J., 0.078; Glendale, Mo., 0.0056; Nevada, Mo., 0.0011; Georgshütte, Upper Silesia, 0.0245; Sagorhütte, Austria, 0.054%. Zinc white may be colored yellowish either by a percentage of cadmium oxide or sulphide, as well as by ferric oxide.

Arsenic and antimony in large quantity render the zinc brittle, and, if used for the development of hydrogen, make an impure gas. Commonly no more than traces of arsenic and antimony are found in zinc, which do not have an injurious influence upon its physical qualities. According to Pack, Missouri zinc contained 0.0590 to 0.0603% arsenic, and 0.0249% antimony; Upper Silesian, however, only contained traces of antimony; zinc produced from the pure red-zinc ores of New Jersey and Pennsylvania, or zinc silicates (Vieille Montagne) is free from arsenic.

Copper is said to make zinc harder and more brittle, even if only 0.5% is present, so that in rolling it will crack at the edge, and the sheets cannot be folded without breaking. Missouri zinc, according to Pack, contained 0.0013 to 0.1123%, Lehigh 0.530%, Upper Silesian, from Reckehütte and Georgshütte, 0.0002%.

Tin is rarely ever found. Karsten says that zinc containing 1% tin, when exposed to a temperature in which it would otherwise become pliant, is rendered so brittle that when rolled at a still lower temperature it cracks at the edges of the sheets. Freiberg, Saxony, zinc contains up to 0.07%, and zinc resulting from the smelting of scraps, roof gutters, etc., may contain more, due to the solder.*

Silver, according to Pufahl, was found in Upper Silesian zinc (Georgshütte and Reckehütte) in quantities of 0.0007 to 0.0017%, and in zinc from Brixleg, in Tyrol, 0.0016%, probably due to the lead ores occurring with the zinc ores. Zinc from the Vieille Montagne works and from Missouri ores was free from silver.

Manganese is seldom present in zinc except in traces.

Bismuth.—Zinc from Peru and Reckehütte, Upper Silesia, contained traces of bismuth.

* *Berg- u. Hüttenm. Ztg.*, 1888, p. 28 (Föhr).

Thallium.—Kosmann* found in a specimen of crude zinc smelted from scrap 7.15% lead, 0.99% cadmium, 1.40% thallium, 7.19% arsenic, 2% iron, and some oxide of zinc.

Indium.—The Freiberg zinc contains 0.0393 to 0.0524% of indium.†

Magnesium and aluminum were found in zinc by Wittstein, of the former 0.46%, and of the latter 0.17%.

Sulphur is present in many zincs made from blende, and may deteriorate their qualities. Pack found in Missouri zinc 0.0035 to 0.0741%; in zinc from Georgshütte, Upper Silesia, 0.0020%, and from Peru 0.0601%. According to Funk,‡ the sulphur does not appear to be chemically bound to zinc, but occurs as zinc sulphide carried over in the distilling. According to Funk's method, sulphur can be detected when present in the proportion of 1 : 10,000,000. In a lot of bar zinc, said to be chemically pure, he found 0.004% sulphur.

Carbon also occurs in zinc, but according to Funk it is not chemically combined, but carried over mechanically in the distillation. The odor of the gases generated by the action of acids on zinc is principally due to hydric sulphide, not to hydrocarbons. Funk says that carbon may be detected in zinc when present to the amount of 1:1,000,000. Zinc from Southwestern Missouri is said to contain a carbon percentage of 0.0016 to 0.1775.

Silicon has been found in many sorts of zinc, for instance, in that from Bethlehem, Pa., 0.239%, and from the Southwestern Missouri works, 0.0346 to 0.1374%.

Chlorine to the amount of 0.2 to 0.3% was found by Künzel,§ together with lead and iron, in spelter made from the crusts from a Belgian works. To the eye it was good metal, but proved to be quite unrollable. Jensch also calls attention to the presence of chlorine in zinc.

Zinc oxide may commingle with the zinc either in the condensers of the distilling furnace or in the resmelting, and when present in larger amounts may produce a pasty metal (burned zinc), which gives castings without sharp edges, brittle, and difficult to work with chisel and file. Such burned zinc is produced when the distillation is carried too far, whereby the atmosphere of carbonic acid in the muffle or retort is expelled by air penetrating the walls, and the remaining zinc vapor is oxidized, the oxide going over into the condenser. This can only be prevented by checking the distillation before completion, though involving loss of metal thereby. By incorporating sal-ammoniac with the reducing agents, it has been sought to reduce the zinc oxide. Hydrogen is generated by the decomposition of the sal-ammoniac, which at red heat reduces the zinc oxide ($Zn + 2\,NH_4Cl = ZnCl_2 + 2\,NH_3 + 2\,H$); zinc oxide is also reduced by fumes of zinc oxychloride. In the refining of the crude zinc the formation of burned metal need not be feared, the process being carried out in a reducing atmosphere.

Besides the foreign admixtures, the temperature also largely influences the characteristics of the zinc. For instance, satinized sheets, which must possess luster and hardness, for the smoothing of paper, are manufactured from metal heated higher than usual, and left to settle thoroughly, and the sheets are rolled

* *Chem. Ztg.*, 1886, No. 50.

† *Muspratt's Techn. Chem.*, 4 Aufl., Vol. IV., p. 676.

‡ *Zeitschr. f. Anorgan. Chem.*, 1896, p. 49.

§ *Berg- u. Hüttenm. Ztg.*, 1874, p. 6.

oftener and more slowly at a temperature lower than usual; on the other hand, tanners' sheets, which serve as underlay in the working of superior skins, are made of a specially soft and gradually cooled zinc.

Many sorts of crude zinc are so pure that they can be utilized without refining—for instance, the above-mentioned zincs from Pulaski, Passaic, Spain, etc. But, generally speaking, the crude article contains too much lead and iron to be sufficiently ductile, or to yield faultless products—for instance, the spelter from Upper Silesia—and requires to be refined by resmelting in reverberatory furnaces, whereby the largest part of the lead, by reason of its greater specific gravity (11.4), separates from the lighter zinc (sp. gr. 6.8 to 7.3) and settles to the bottom, while between the lead and the zinc an iron alloy (bottom or hard zinc) forms, and mechanical admixtures collect on the surface as dross. Kraut* asserts that the percentage of lead in the plates from the same pouring from the same furnace varies, and remains irregularly distributed even after the remelting, so that a correct test sample (a ladle sample is best) is hardly possible, and the purchaser, in order to protect himself, must stipulate that in the remelting of the zinc no portion rich in lead shall settle to the bottom. How difficult it is to separate lead from zinc is shown by an experiment by Ross.† He permitted a pipe 1.3 m. high, filled with molten zinc at red heat, to cool gradually during 8 days, when the upper part of the bar still contained 0.5% lead; experiments in centrifugal separation gave no better results.

The refining of zinc in the zinc-smelting works of Upper Silesia is highly perfected in connection with the rolling mills; the ores (calamine and blende) utilized by them contain large quantities of lead. For the resmelting of the zinc there are employed reverberatory furnaces 5 to 6 m. long, 2.75 to 3.2 m. broad in front, 2.75 to 3.2 m. at the back, 1.8 to 1.9 m. high,‡ with two grate-firings on one of the narrow sides and a hearth-sole of tamped clay inclining toward the ladle-door, in front of which a sump is hollowed in the hearth. Between the two fireplaces is the charge opening, from which a sloping, longitudinally corrugated iron plate leads into the hearth. The ladling-pot lying opposite is separated from the furnace-chamber by a thick division (kapelle) of refractory material, the bottom of which resting upon firebrick in the hearth dips into the metal bath, whereby the outlet is sealed against the air and the impurities floating on the surface of the metal are kept away from the ladling-pot. The gases of combustion pass from the fireplaces over the hearth and through flues in the arch before the kapelle into a canal over the arch, and thence into a chamber which conducts them to a chimney 20 to 25 m. high. The zinc plates are pre-heated upon the corrugated iron plate, and are gradually pushed into the zinc bath. A full charge is from 20 to 30 tons zinc, which is fed in during 2 or 3 days; the temperature is not raised much beyond the fusing point of the zinc.§ A large part of the lead will thereby separate and settle; over it comes a layer of zinc containing lead and iron (hard or bottom zinc), and above this the refined zinc,

* *Zeitschr. f. Analyt. Chem.*, 1886, p. 269. † *Berg- u. Hüttenm. Ztg.*, 1894, p. 320.

‡ *Ibid.*, 1873, p. 290, Taf. 8, Figs. 5–7; 1877, p. 100, Taf. 3, Fig. 5 (Older furnaces).

§ According to older statements by Daniell zinc fuses at 412°, but more recent experiments place it at 433.3° of the mercurial thermometer, and at 415° of the air, and Le Chatelier's thermo-electric pyrometer (*Berg- u. Hüttenm. Ztg.*, 1895, p. 419). The boiling point of zinc is, according to Becquerel, 891°; Deville and Troost place it at 1040°, and Violle at 929.6° C.

upon the surface of which dross or zinc ash collects, protecting the metal from oxidation. About every 12 hours the dross is raked upon the sloping iron plate at the charge-hole, mixed with sal-ammoniac in the proportion of about 0.5 kgms. to 180 or 200 kgms., to free it from adhering particles of zinc and reduce zinc oxide, the residue being finally withdrawn through the charge-hole.

For emptying the furnace it was the custom formerly, after the dross had been raked away, to first ladle out the zinc and next the lead, after which the furnace was again charged with a fresh lot; this caused a great expenditure of fuel and loss of time. The present custom is to draw off the lead from underneath the zinc about once every month, without interrupting the work, and to recharge continuously with fresh zinc. For this purpose an endless-screw pump* is inserted through one of the openings on the long side of the furnace sloping to the hearth; the pump is a cast-iron pipe 1.3 to 1.4 m. long, 0.12 m. diameter, in which works an endless screw having about 12 turns and 90 to 100 mm. pitch. A still simpler contrivance† consists in placing a large iron or clay pipe, closed at the lower end with a clay plug, upon the hearth-sole, in the same way as the screw pump; the clay plug is then knocked out and the lead rises in the pipe, whence it is ladled out. It is the custom in Freiberg, after 600 cwts. of zinc have been refined and ladled out, to tap the lead from the hearth into an iron mold, to ladle off the metal from the top, which is likely to contain a little zinc, and to pole the pure lead remaining with green wood staves, to be used for assay-lead very poor in silver.

At Hohenlohehütte,‡ in Upper Silesia, 5000 kgms. of zinc are ladled out every 12 hours, and the same quantity of crude zinc is added; the lead settled out is removed every 8 days. When the zinc is to be drawn it is left to stand for about one hour, so that it may assume a uniform temperature; it is then poured with galvanized cast-iron ladles into cast-iron molds, and the crusts of zinc-iron forming on the sides are knocked off from time to time. A screw pump§ is more rarely used.

It is advisable to have the molds standing upon a revolving frame, or placed in a circle and filled from a movable clay-lined pot. After the skimming of the still fluid zinc in the molds, it is covered and left to cool, whereby the metal becomes more fibrous and pliant; the castings are still hot when sent to the rolling mill. Zinc is most pliable at 150° C.

For removing the layer of hard zinc, a work generally done during an idle time of the furnace, for instance when repairing, the zinc is first ladled out, the lead then pumped out, and finally the pasty mass of ferruginous metal is ladled out with perforated pots, whereby the entrained lead may drip off. From 9 to 10 tons zinc are refined in one furnace in about 24 hours, using about 900 kgms. stone coal.

The refined products are: 1. Zinc.‖ In Upper Silesia the lead percentage,

* *Berg- u. Hüttenm. Ztg.*, 1883, Taf. 1, Fig. 9; 1888, p. 422; 1890, p. 130.

† *Ibid.*, 1890, p. 130. ‡ *Ibid.* § *Ibid.*, 1883, p. 14.

‖ Analyses of zinc, THE MINERAL INDUSTRY, Vol. II., p. 669 (Upper Silesia, Austria); Vol. IV., p. 484 (United States, Peru); Schnabel, *Handb. d. Mettallhüttenk.*, Vol. II. (Southwestern Missouri, New Jersey, Pennsylvania, Austria). Methods of analysis, *Fresenius Quant. Chem. Analyse*, 6 Aufl., Vol. II., p. 373; *Zeitschr. f. Analyt. Chem.*, 1881, p. 503 (Günther); Dammer, ill., *Lexicon d. Verfölschunger*, etc., Leipsic, 1887, p. 988 (v. Knorre).

averaging 2% in crude zinc, is in refining reduced to from 1 to 1.7%, and the iron to from 0.02 to 0.025%. Spelter with 3 to 4% lead produces, after refining, zinc with 1.7% lead, 0.02% iron, and 0.04% sulphur at Hohenlohehütte.*

2. Liquated lead, with 1 to 2% residual zinc, used for desilverizing crude lead, for accumulators, etc.

3. Hard or bottom zinc, for which different treatments have been proposed, such as repeated remeltings in an iron kettle, and gradual cooling analogous to the Pattinson process, whereby more iron separates constantly and purer zinc results, re-distilling, electrolyzing, heating with $\frac{1}{40}$ of its weight of sal-ammoniac in a reverberatory furnace, etc. This ferrous zinc is at present used for compounding hard brass for delta, tombac, muntz, sterro metal, etc.

4. Dross (zinc ash), which is reduced by sifting, comminuting the coarser pieces and sifting again, and adding to the metal in the smelting furnace.

OTHER METHODS FOR REFINING ZINC.

Numerous methods have been proposed for refining zinc in the dry way, and several are employed on a small scale. Among others, Richards adds aluminum to the liquid zinc and permits the mass to stand for some time, whereby the impurities, except lead, separate and rise to the surface. L'Hôte, in order to expel the arsenic, stirs into the molten metal 1 to 5% anhydrous magnesium chloride to volatilize arsenic chloride. Harnickel distils the zinc under strong pressure and condenses the vapor in vacuo, whereby the more volatile constituents go off. Lescolur first smelts the zinc with saltpeter, next with zinc chloride, driving off antimony, arsenic, sulphur, and phosphorus; but iron, lead, and copper cannot be expelled. Selmi, in order to remove the arsenic, dips a piece of sal-ammoniac, fastened to an iron wire, once or twice into the molten zinc, holding it firmly against the bottom; zinc oxide is also reduced thereby. Gumming smelts granulated zinc with a mixture of sulphur and soda. Stolba mixes three parts calcined gypsum with one part pulverized sulphur and water into a thick paste, forms this mass into balls of about 5 cm. diameter, dries them on a wooden stick, and dips them into the molten zinc, when sulphur and water vapors escape with a violent commotion of the metal bath; after a while the outer crust of the balls is taken off and the operation repeated several times, after which zinc free from arsenic remains, though only traces of iron and lead are removed. Merton for removing copper smelts zinc together with sulphur, rakes off the ensuing sulphide metals, next distils the zinc under pressure, and pours it into molds cooled by water. According to Funk, sulphur and carbon may be removed by repeated meltings of the zinc and filtering through an asbestos funnel. If molten zinc be stirred with a greased wooden rod, and an addition of stick sulphur, sulphide metals will separate and collect on the surface; the addition of the sulphur is continued until a sample shows no arsenic with a Marsh apparatus. Ende thinks, however, that traces of fat may by this process remain in the zinc, which in the reduction of the ferric oxide salts with zinc and subsequent titration of ferrous oxide with potassic permanganate may cause disturbance.

*Refer also to THE MINERAL INDUSTRY, Vol. II., 1893, p. 668.

THE ELECTROLYTICAL PRODUCTION OF ZINC.

A large number* of methods of using various kinds of raw materials (ores, alloys, salts, etc.) as soluble and insoluble anodes have been proposed, but few of these propositions have had a practical success. The conditions of the electrolytical production of zinc are more unfavorable than, for instance, that of copper and the precious metals, on account of the inferior value of the zinc and the indifferent conductivity of its ores, so that they are less adapted for soluble anodes; beside this, the zinc tenor of the ores must generally be converted in a chemical way into a water-soluble condition by an oxidizing or chlorinating calcination, whereby salvage suffers in the lixiviation, since zinc salts will remain in the residues. The former difficulty that the zinc, in place of being cohering, was deposited in a spongy state, has now been remedied.† For the present this industry is still based on the recovery of zinc from raw materials, which were not adapted to economical treatment by former metallurgical methods, *e. g.*, zinc-lead ore from Broken Hill, New South Wales,‡ and iron pyrites cinders§ in Meggen, Westphalia.

The Broken Hill ores are now smelted with favorable results by Ashcroft's process,‖ at Gray, in Essex, England,¶ where they are roasted in a reverberatory furnace, the resulting zinc oxide lixiviated by a solution of ferric chloride ($3\ ZnO + Fe_2Cl_6 + 3\ H_2O = 3\ ZnCl_2 + Fe_2(HO)_6$, the zinc chloride solution electrolyzed with anodes of iron or carbon and cathodes of sheet zinc, and the iron-rich residue, containing lead and silver, smelted in the lead blast furnace. For precipitating 1 part zinc, from 3 to 4 parts stone coal are used for steam; the greatest expense is due, however, to the consumption of iron, which amounts to 67 parts iron for 100 parts zinc. The advantages of the process are: A zinc-poor lead ore for smelting, the development of an important amount of electrical energy in the circuit, and the production of a compact zinc, while at the same time the solvent is regenerated.

The pyrites of Meggen, with an average of 8% zinc, after the roasting for the manufacture of sulphuric acid, yield residues (cinders) which, with a zinc percentage ranging up to 15%, cannot be utilized as iron ore; this becomes possible, however, after the larger part of the zinc has been extracted from them by chlorinating roasting and lixiviation. The resulting zinc liquors furnish, after purification from iron, the material for the electrolytical production of zinc in a plant at Duisburg** on the Rhine, but the process is not known. The metal enters commerce as plates 1 in. thick, with crystalline fracture, and can be used directly for alloys, but must be resmelted if it is to be employed in the manufacture of sheets. The electrolytical plants at Fuhrfurt a. d. Lahn, Weidenau, and Nomberg on the Rhine are said to also use pyrites cinders as raw material. Aaron

* *Berg- u. Hüttenm. Ztg.*, 1895, p. 431 (Hasse); Borchers, *Electro-metallurgie*, Braunschweig, 1896; Abtheilung, Vol. II., p. 266; Dürre, *Ziele u. Grezen d. Electro-metallurgie*, Leipsic, 1896, p. 213; Vogel and Röffing, *Handb. d. Electro-chem. u. Electro-met.*, Stuttgart, 1891, p. 187; Schnabel, *Handb. d. Metallhüttenk.*, 1896, Vol. II., p. 209; THE MINERAL INDUSTRY, Vol. IV., p. 807 (Borchers).

† *Berg- u. Hüttenm. Ztg.*, 1895, p. 441.

‡ *Ibid.*, 1894, pp. 931, 1069; 1895, p. 354; 1896, p. 284.

§ *Ibid.*, 1880, p. 463; 1883, p. 517; 1884, p. 475; 1887, p. 144; 1894, p. 1; 1895, p. 351.

‖ *Ibid.*, 1894. p. 93; 1895, p. 427; 1896, p. 310.

¶ *Chemik. Ztg.*, 1896, p. 595; *Engineering and Mining Journal*, Vol. LX., 26, 146.

** *Berg- u. Hüttenm. Ztg.*, 1895, p. 169.

Hirsch manufactures, at Ilsenburg in the Harz, a refined zinc from zinc containing lead according to a process unknown to the writer.

The experiments of Rösing,* at Friedrichshütte, in Upper Silesia, and Rösler and Edelmann† at Hoboken, near Antwerp, to treat electrolytically the silver-zinc-lead alloy resulting from the desilverizing of lead bullion, have not proved to be profitable as compared with the usual methods. Electrolytical zinc with 99.92 to 99.94% zinc, up to 0.002% silver, and at most 0.05% lead, 0.02% copper, and 0.0044 to 0.0099% iron, was obtained at Hoboken, together with a cathode sediment containing about 75% silver and 12% lead, from which copper, zinc, and iron were separated with dilute sulphuric acid, while the chloride of silver present was decomposed with iron filings.

The electrolytical kinds of zinc mostly contain up to 0.05% lead (according to a statement in the *Oesterr. Ztsch. f. Berg- u. Hüttenwesen*, 1895, p. 123, Duisburg zinc contains 99.98% zinc and only 0.01 to 0.03% lead), but are almost twice as dear as other commercial zincs, and consequently are employed to a limited extent only—for instance, in chemical laboratories, for the development of hydrogen, for composing very ductile brass, etc. Sheets made from it are not suited for engraving on account of their softness. A zinc just as pure, if not purer, can be obtained from pure zinc ores in the ordinary way by distillation, although it is also quite expensive. For instance, as already stated, Bertha zinc‡ contains only traces of lead, and only 0.019% iron; Passaic zinc, 0.027% lead and 0.002% iron.

The purest zinc, free from lead, iron, and cadmium, was produced by Mylius and Fromm§ on a small scale by repeatedly electrolyzing a solution of basic zinc sulphate, and distilling the zinc in vacuo; it was purposely produced in a spongy condition. It contains in 100,000 parts at most 1 part foreign substances, which could not be detected chemically, and was only made known by the spectroscope.

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* *Berg- u. Hüttenm. Ztg.*, 1886, p. 463; 1895, p. 432; *Dingler's Polyt. Journ.*, 1887, p. 93.

† *Chemik. Ztg.*, 1892, No. 84, p. 1564; *Berg- u. Hüttenm. Ztg.*, 1894, p. 109; 1895, p. 433.

‡ *Engineering and Mining Journal*, 1893, Vol. LVI., No. 22 (Moxham); *Berg- u. Hüttenm. Ztg.*, 1894, p. 107.

§ *Zeitschr. f. Anorg. Chem.*, 1895, p. 144.

The Treatment of Zinc-Lead Sulphide Ores.

By F. L. Bartlett.

Next to iron pyrite, zinc sulphide is the most common constituent in all ores of copper, lead, and iron, as well as of the precious metals. Almost every mining camp of the Rocky Mountain mining districts is afflicted with it, sometimes found on the surface mixed with lead, copper, and silver ores, but more often coming into the vein as depth is attained. The Leadville, Colo., and the Broken Hill, New South Wales, deposits are common examples of the manner in which zinc blende displaces the soft surface deposits of lead and iron carbonates as the mines become deeper.

It has ever been a problem with the miner how to get rid of zinc without loss of other more valuable metals. The smelters do not want it, and charge extra for smelting any ore which contains more than 12 or 15% of blende. The patent offices in this country and in Europe abound with devices and schemes for separating the zinc from the lead in ore. Indeed, the writer is in possession of nearly 150 descriptions of patented processes for treating zincy ores, not one of which is in practical use to-day.

METHODS OF SEPARATION.

It would take too much space to discuss the many methods devised, and little practical information would be gained. The feasible methods may be divided into the following classes:

1. Separation by concentration.
2. Separation by electricity.
3. Separation by chemical methods.
4. Separation by smelting.

These will be discussed briefly in turn.

1. *Separation by Concentration.*—This method is practiced largely at Leadville, Kokomo, Georgetown, Creede, and many other places in Colorado, as well as at Park City, Utah, and some points in Montana, but its success depends entirely on the character of the ore. Pure zinc blende rarely carries any silver or gold; the lead, copper, and iron which are mixed with it carry the values. Cleaned, separate crystals of blende will invariably be almost free from silver, and when the blende is coarsely crystallized careful crushing and concentrating will easily make a good separation of the values. Oftentimes, however, the zinc is finely crystallized, or, in fact, is blended with lead, iron, and copper; in such cases no separation or concentration can be made. For instance, in a mixture of blende, galena, and iron and copper sulphides, if coarsely crystallized, the galena can be easily taken out by any good form of wet concentration; but the copper, zinc, and iron sulphides, owing to their nearly equal specific gravities, will come off together. A second concentration can be made of the last three and the iron increased somewhat, but this is not of much utility. The average specific gravities of the ores commonly mixed with blende are as follows:

Galena, sp. gr.....7.40	Chalcopyrite, sp. gr.....4.04
Pyrite, sp. gr.....4.85	Blende, sp. gr.....4.00

Owing to the value of iron in the great smelting centers for fluxing, attempts have been made to roast such mixtures to form a magnetic iron, which can be separated by magnets. Some fair results have been obtained on Leadville tailings in this way, but it has not yet proved practical on a large scale. As blende is harder than iron pyrite, another method, now on trial, is to crush the tailings in a ball crusher, expecting the iron pyrite to crush first to a fine powder, and by screening to separate it from the less friable blende, but at best this can only make a partial separation. In short, it may be said that the separation of galena from blende will be successful only when the mixed ore is well crystallized, and even then the loss will be heavy. A fair example of the mixed Leadville zincy ores will run about as follows:

Zinc blende	28%	Silver	12 ozs.
Iron pyrite	36%	Gold	0.05 oz.
Galenite	16%		
Rock	20%		

Such an ore will give concentrates containing:

Galena	40 to 50%	Silver	20 ozs.
Blende	15 to 20%	Gold	0.10 oz.
Pyrite	30 to 50%		

The tailings will contain 35 to 50% blende, 4 to 6% galena, balance pyrite and rock, and there will be from 7 to 10 ozs. silver left in the tailings. At Georgetown the tailings run about 35% metallic zinc, or about 50 to 55% blende, with 5 to 8% galena, containing 15 ozs. silver and 0.1 oz. gold. At Park City, Utah, the tailings from zincy ores run about 15 ozs. silver and 6% lead, balance blende and pyrite. As a rule, it may be said that silver in zinc-lead ores is mainly contained in the galena, and that the values left in the tailings will depend on the amount of the lead left in them, which will carry the same proportion of silver that the galena does in the original ore. Judging from the assays of many thousand tons of zinc tailings purchased at the American Zinc-Lead Works, Cañon City, from many different concentrators, the best work done has been to reduce the lead to 2% in the tailings, while they often run 8, 10 and 12% lead, with an average of about 6%, and a saving of 50 to 70% of the gold and silver is as much as can be expected in zincy ores by concentration. The lead may be saved in the proportion of from 60 to 80%, according to the class of ore.

2. *Separation by Electricity.*—One of the latest suggestions for the separation of zinc from sulphide ores is rapid fusion by electricity. The blende being practically infusible, it is expected that the more fusible pyrite and galena will run away from the blende. On a small scale this works fairly well, the lead, copper, and iron making an easily fusible matte which leaves a scoria of infusible zinc blende. There are, however, serious difficulties in the way of operating on a commercial scale by this process.

Other electric processes contemplate the production of electrolytic zinc after the zinc of the ore is put in solution by chemical means. This comes more properly under the head of "wet methods."

3. *Separation by Chemical Methods.*—Mixed zincy ores have offered a field to the "wet process" inventor which has been worked without intermission or suc-

cess for many years. It would occupy too much space to even mention the names of the processes; none of them have been successful, so far as the writer knows.

It seems very easy to roast zincy ores to form zinc sulphate and leach that out, producing a fairly clean solution of that salt, practically free from silver and gold, leaving all the lead, iron, and rock behind. Then it seems quite easy to precipitate the zinc and render it available as oxide or carbonate, but like many other things it looks much easier than it is. It is extremely easy to roast part of the blende into soluble sulphate, but very difficult to convert all of it. It is still more difficult to prevent a large share of the iron and copper from going into solution, and the result is found to be a nasty solution of great bulk, from which lime, soda or magnesia may be used to precipitate the zinc, but the result is an enormously bulky precipitate, very difficult to wash and handle, and of little value when obtained. The residue, supposed to contain the iron oxide, lead sulphate, gold and silver, is left in a finely divided condition, undesirable for most smelters.

Electricity has been used to precipitate pure zinc from solutions of roasted zincy ores, and vast sums of money have been spent on this process with occasional fairly good results, but the writer does not know of any which have as yet proved profitable to the investors, and for this reason detailed descriptions of the methods tried will be omitted.

The obstacles to be overcome are:

1. Excessive cost of making a clean, satisfactory solution of the zinc from blende ores.
2. The great electromotive force, or power, required to produce metallic zinc.
3. The difficulty of preventing polarization.

These difficulties may be overcome, but at best the process is bound to be expensive. The writer has followed up nearly all the experiments in this line without being convinced that any one of them would prove successful in a commercial way.

4. *Separation by Smelting.*—In discussing this, as well as the other methods of separation, space will not permit a detailed description of the failures made. In the West, where the mines are located within a reasonable distance of the large smelters, all classes of zincy ores command a ready sale if they contain a fair percentage of lead, copper, gold or silver, or mixtures of these, and the treatment charge is based on the treatment of other ores plus an extra charge when the zinc exceeds a certain percentage. A few years ago 8% was the zinc limit, now 12% is allowed. The regular price is charged for treating ore containing up to 12% zinc; if more than that is present 50c. for each additional per cent. of zinc is added to the amount charged. The zinc smelters of Kansas and Missouri will purchase pure blende provided it contains not more than 8% of iron, nor more than 2% of lead, but they will not pay for the silver, gold, or copper, if any, in the ore. The lead smelters work off zinc by the "drowning" process, *i.e.*, by mixing in all the zinc the charge will stand. These ores must be well roasted, and the limit of the amount of zinc that can be worked off by mixing may be from 8% to 12% in the charge.

The objections to the zinc are based mainly on three factors:

1. Greater loss of silver in roasting.

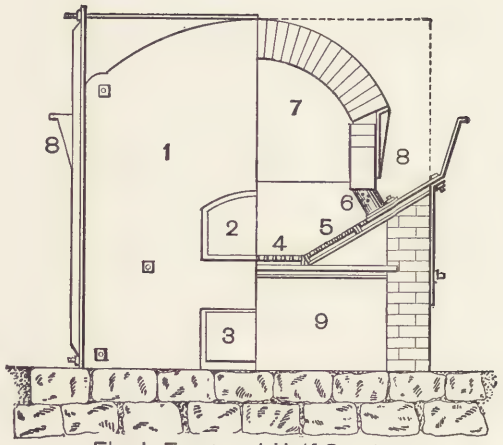


Fig. 1 Front and Half Section

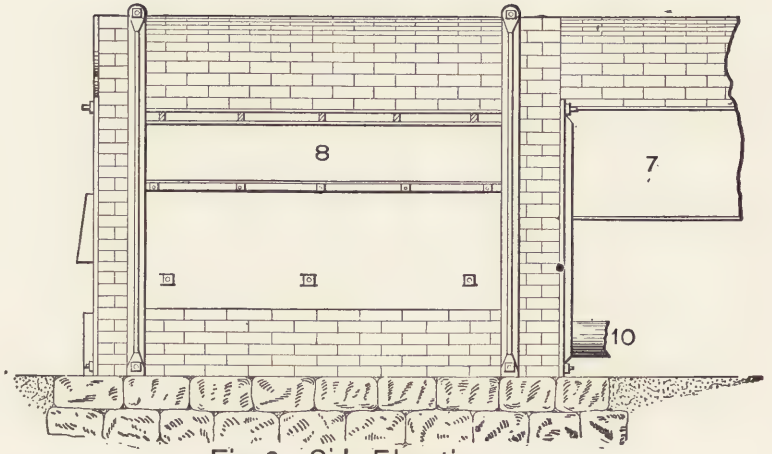


Fig. 2 Side Elevation

BARTLETT'S BLOWING-UP FURNACE FOR ZINCY ORES.

2. Greater loss of silver in smelting.
3. Retardation, or slower smelting.

The amount of loss in roasting zincy ores is not inconsiderable; with 20% of zinc it may reach 1½ ozs. silver to the ton of ore. In the blast furnace the loss is increased according to the amount of zinc in the charge. It causes a great volume of fume, which carries off mechanically a great deal of finely divided ore. It forms, if any sulphur is present, a zinc sulphide which is practically infusible. This zinc sulphide will carry with it both gold and silver; some of it will get into the slag and cause loss, and if present in large quantities it forms bad accretions in the furnace, and it also retards smelting, making slower running and consequently increasing the cost of the operation. The ordinary lead smelters, therefore, are justified in making an extra charge when zinc is present to any considerable extent in the ore. While many plans have been devised to save zinc in ordinary lead smelting, no plan has yet proved of any value.

SMELTING AT THE CAÑON CITY PLANT.

This plant was devised by the writer for the purpose of utilizing the zinc and lead, as well as the gold, silver, and copper in the ore. The process is based on the results of many years' work with complex zincy ores. The original and fundamental idea is to drive off the zinc and lead in a mixture which can be refined into a white pigment used as a white lead substitute, and without loss of silver or gold. The writer found that a fractional distillation can be made of sulphide ores, driving off the lead and most of the zinc without undue loss of silver, provided there is present an excess of sulphur, or sulphide of iron or copper, sufficient to make a small amount of matte. The obstacles to be overcome were to prevent the formation of acid compounds destructive to the collecting apparatus, and to make a merchantable pigment when using ordinary bituminous slack coal and sulphide ores, the subsequent smelting of the cinder after eliminating the zinc being an exceedingly simple matter.

The process at Cañon City, Colo., is in its simplest form as follows: All ores containing 20% or more of zinc are crushed to pea size, mixed with 10 to 25% of pea and dust coal, and blown up in a specially constructed furnace (Figs. 1 and 2), using a blast of 4 to 8 oz. pressure. It requires from 20 to 40 minutes only to drive off the lead and the greater part of the zinc and sulphur. A cinder is formed containing the silver, gold, and copper, in a matte mixed with more or less slag. This cinder is smelted in an inclined low blast furnace (Figs. 2 and 3), mixed with other suitable ores and fluxes to produce a high-grade matte. The zinc in this charge may reach from 15 to 20%, and is mostly driven off as fume. The fume from the "blowing-up" furnaces and from the blast furnaces is caught in bags, mixed and refined in a suitable furnace (Figs. 4 and 5), whereby carbon, arsenic, sulphur, and other impurities are eliminated, and a pure white pigment produced, suitable, after being ground in oil, to be used as a white lead substitute.

DESCRIPTION OF THE FURNACES.

As the processes used at Cañon City are patented, and the manipulations are peculiar to this particular branch of work, it will be uninteresting to enter into

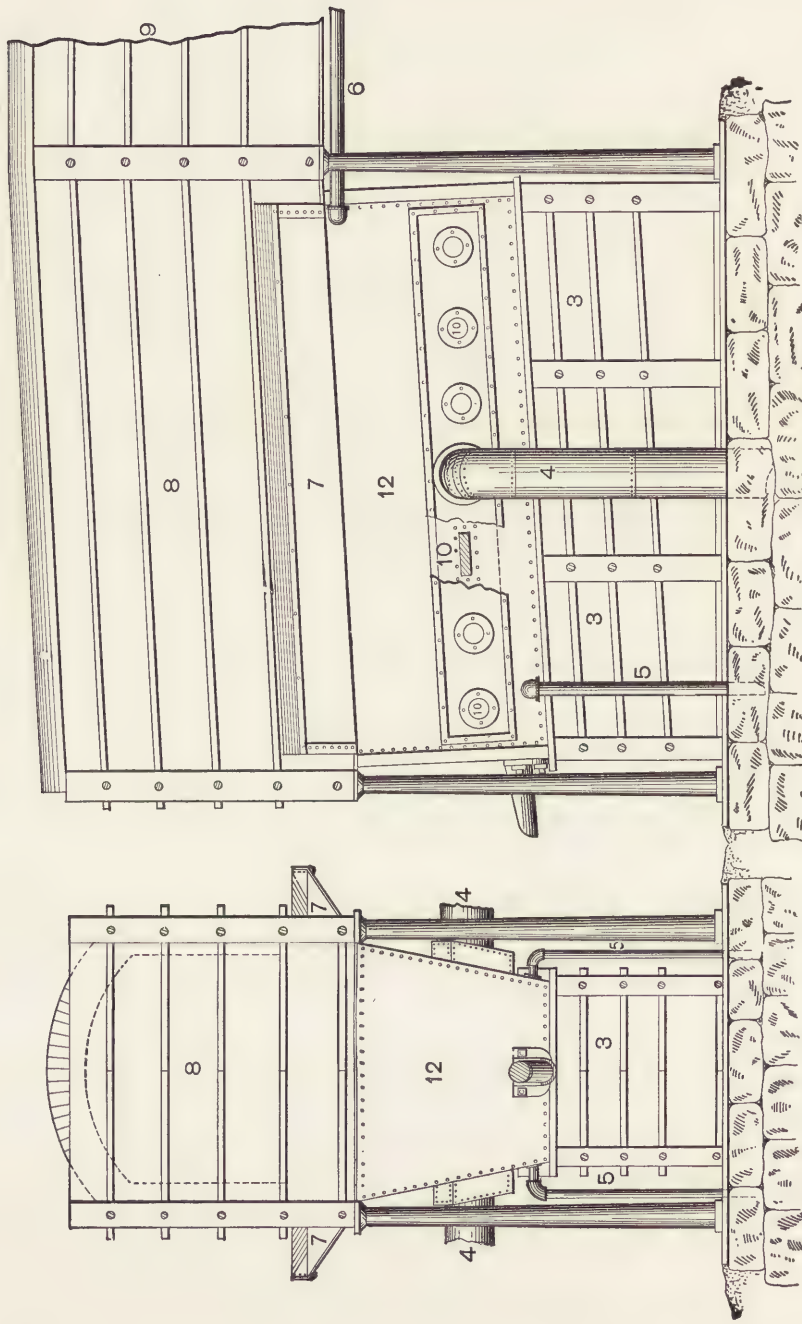


Fig. 4 Side Elevation

Fig. 3 End Elevation

BARTLETT'S BLAST FURNACE FOR ZINCY ORES.

minute details. Some ideas have been worked out, and some old theories have been exploded, as will be fully detailed further on.

Of the furnaces used at Cañon City, Figs. 1 and 2 represent the latest form or type used for "blowing up."* This furnace is used for ores containing 20% or over of zinc. The ore crushed to pea size is mixed with 15 to 20% of slack coal, charged into the pockets (8), and burned on the grates (4) and (5). The fume passes over the bridge wall and into flue (7), thence on through a cooling system to the bag rooms. Blast is applied through the pipe (10), and passes up through the grates (4) and (5) and the tuyères (6). The ore sinters or slags as soon as the zinc and lead are sufficiently driven off, and the cinder is drawn out of the front door; fresh ore dropping down out of the pockets (8) keeps the operation continuous. The sulphur can all be driven off, or any desirable amount can be left, at the option of the furnaceman. No lead is left in the cinder. The operation is rapid and cheap. Almost any class of ore can be sintered or slagged, and in this condition is well adapted to the subsequent blast-furnace process. The capacity of the furnace is from 3 to 6 tons daily, according to the class of ore; it is used also for burning slack coal and dust for steam purposes, or for puddling or roasting furnaces. It produces a clear white flame of high heating capacity, absolutely free from smoke or unconsumed gases, as the heat passing through the side walls is sufficient to drive off the gases before the coal reaches the combustion chamber. The gases passing downward over the coal are ignited by the hot-air jets from the tuyères (6), and are thus burned in advance of the partially coked coal which slides down on the grates. The grates and the tuyères last well, but no firebrick have been found for the arches which will stand the intense heat for any great length of time; the arches are, however, easily and cheaply renewed.

Figs. 3 and 4 represent the blast furnace† constructed of steel jackets (12), the end jackets being removable. The jackets are 40 in. high, 108 in. long, and 36 in. wide between the tuyères (10), which are 14 in. from the bottom. The tuyères (10) are longitudinal slots 8 in. long by ¼ in. wide, controlled by adjustable blocks. The charge is fed on both sides into the pockets (7), which rest on top of the jackets. The ore sliding down the jackets makes an angle of about 45°, being very low in the center. To facilitate removal of melted material the furnace is inclined 1 in. to the foot; it is run with a hot top and the fumes pass out through the flue (9), thence into the bag rooms. A hot blast is obtained by taking up the waste heat from the flues. Such a furnace handles from 30 to 70 tons daily, according to the quality of the charge, which may contain as high as 20% of zinc. The furnace is run on 30-day campaigns, when it becomes necessary to stop and clean out the flues. Very little flue dust is made, but what is made sinters to the flue walls, and has to be removed when the area of the flues gets too small. Large percentages of raw sulphide ores can be run in the charge owing to the intense oxidizing action.

Figs. 5 and 6 represent the pigment refining furnace.‡ This consists of a helical screw (2) running in a cast-iron retort (1), driven by a reduced cog gear (3), and is heated through the furnace (13) and flues (14). The raw pigment

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*U. S. Patents Nos. 420,030, 512,531 and 539,021.

†U. S. Patents Nos. 406,896. 406,870, 406,871, 415,185 and 420,029.

‡U. S. Patents Nos. 406,898 and 415,186.

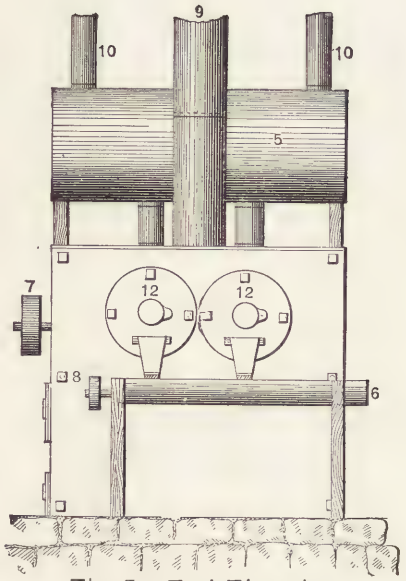


Fig. 5. End Elevation

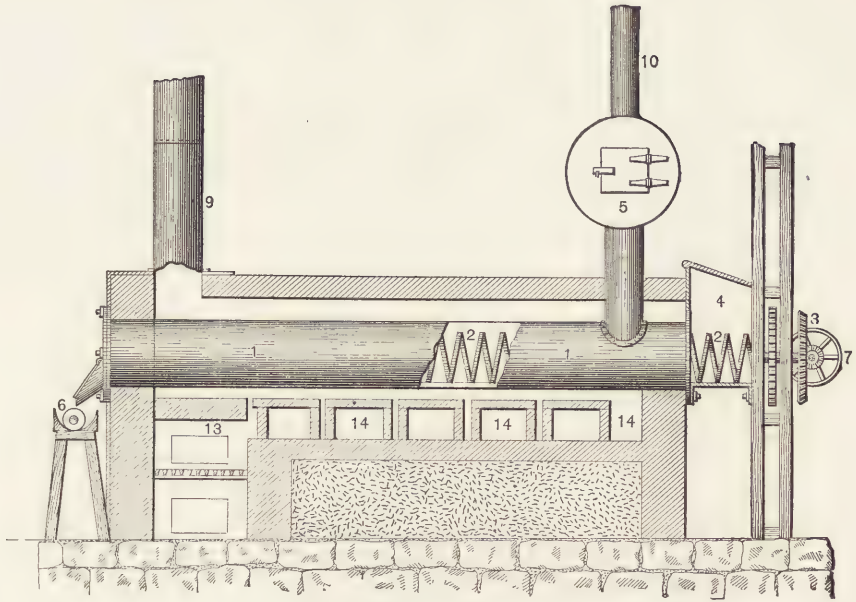


Fig. 6 Side Elevation

BARTLETT'S REFINING FURNACE FOR MAKING ZINC-LEAD PIGMENT.

from the furnaces described is fed into the hopper (4) and pushed slowly forward through the retort. The air supply is regulated by the disks (12). The gases and expelled products pass into the collecting drum (5), and the finished product falls into the conveyor (6), where it is cooled and conveyed into bins.

The object of this process is twofold; first, to drive off arsenic, cadmium, and other impurities, and to reduce the sulphur and carbon; second, to condense and solidify the fume, which passes over and under the heavy screws, and is ground and condensed under heat. The pigment formed in this way is not only very dense, and free from impurities, but is so altered that it in no way resembles the ordinary sublimed pigments. It is fireproof, white, dense, and free from all loosely combined elements. The heat required is about 1500° F., which is quite sufficient to produce a fixed and stable compound. The pigment is almost chemically free from anything but zinc and lead, but much experience is required to refine it properly, the adjustment of air, heat, and speed being a matter only gained by experience and knowledge of the proper requisites of a good pigment.

Considerable quantities of mercury, cadmium, and selenium, and in fact nearly all the rare elements, are collected in the drum (5).

PRODUCTION AND COLLECTION OF ZINC AND LEAD FUME.

When mixed lead and zinc sulphide ores are exposed to an air blast in the presence of carbon, the lead is first driven off in the form of sulphate, sulphite and oxide; zinc follows, and goes off as mixed oxide, sulphite, and sulphide; sulphur passes off at the same time as sulphurous gas, together with more or less finely divided ore and carbon carried mechanically. This fume, if allowed proper expansion and cooling, will drop all the fine ore and coal as flue dust, while the fume itself will never settle, and will travel any distance in the air. A close distinction must be made between fume and flue dust. The former is smoke, consisting of extremely finely divided materials chemically combined, as zinc oxide, lead oxide, lead and zinc sulphides, while flue dust is fine ore and coal carried off by the action of the blast, mechanically mixed with the fume. The fume, when well cleared of the mechanically carried dust, will contain very little silver, while the flue dust will always contain about the same proportion of silver as the original ore mixture.

A common notion has been to settle the dust from fume by conveying it through long flues, but this is of little utility, even enlarged chambers having but little effect on hot fume. If, however, the fume is cooled, then passed through large chambers and made to impinge against numerous projections, it becomes broken up, and the dust will settle. This provides very well for the collection of dust, but the fume proper will keep on, and the only rational method of collecting it is by straining through cloth or some similar medium. Water, steam, and spray have no effect on fume, as it is about as hard to wet as a duck's back. Loosely woven woolen or cotton cloth will stop it and allow the gases to pass through; one square yard of such cloth can be depended on to collect one pound of fume daily if twice cleaned or shaken, and it is doubtful if any cheaper or better device can be found. For woolen cloth the fume should be cooled to 250° F.; for cotton, to 200° F. Woolen will last from 6 to 10 years; cotton will last 18 months to 2 years. Woolen costs about six times as much as cotton, and will last about six times as long.

LOSS OF SILVER IN FUME.

At the Cañon City works it has been found that there is a certain fixed loss of silver in the fume regardless of the amount of silver in the ore mixture. In the well settled fume from the "blowing-up" process it amounts to about 5 ozs. per ton of fume, while in the blast-furnace process it amounts to 8 or 9 ozs. per ton of fume. Arsenical and antimonial ores lose more silver than others; straight zinc ores, nearly free from lead, lose the least silver. Copper and iron sulphides in the mixtures prevent silver loss; lime increases it. Fully oxidized zinc and lead ores, when blown up with carbon, lose silver very heavily.

The fundamental basis of the patents covering the Cañon City process is built on the fact that the ores are all treated in the sulphide form, *i. e.*, an excess of sulphide of iron and copper is depended on to hold back the precious metals during the expulsion of the zinc and lead fume, and an oxidizing action rather than reducing is aimed at, by which means the loss in silver and gold is kept down to the minimum, and is but little greater than that of the ordinary lead smelter.

At Cañon City copper is depended on to collect and hold the gold and silver, and when possible is used in both the "blowing-up" process and in the final blast-furnace smelting. The greater the amount of copper in the charge the less the loss of silver, both in the fume and in the slag; 3% of copper is sufficient to keep the loss down to a reasonable figure; less than that will cause an increased and proportional loss, which may be given as follows:

With 4% copper in the ore mixture the loss in treatment is ¾ oz. per ton; with 3% copper the loss in treatment is 1 oz.; with 2% copper the loss in treatment is 2 ozs.; and with 1% copper in the ore mixture the loss is 3 ozs. of silver. The use of lime in the charge up to 5% causes little effect; with larger percentages the loss of silver in slags becomes lighter, but the loss in the fume increases. Gold is not lost to any appreciable extent; in fact, there is a constant small gain. The amount of lead obtained in the fume is always in excess of the amount paid for in the ore, probably owing to defects in the customary methods of assaying, which show less than the true amount in the ores. The loss of zinc amounts to about 5%, and is fully accounted for in the slags.

NOTES ON THE BLAST-FURNACE PROCESS.

At Cañon City the blast furnace used is radically different from any other, as will be seen by the previous description. The ore charge carried is but 12 in. higher than the tuyères, which are adjustable slots ¼ in. by 8 in. The blast is carried at high pressure, and the top is run exceedingly hot to facilitate the expulsion of the zinc and to prevent incrustations. The furnace is inclined toward the slag and metal spout. The slag and matte are allowed to run continuously, and are separated in outside wells. The cinder from the "blowing-up" process usually contains, as a fair average:

	1.	2.	3.	4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
FeO	36.4	45.2	50.5	30.5
SiO_2	30.2	27.5	28.6	30.6
S	10.6	7.7	8.6	5.6
C	3.2	3.5	3.2	3.0
Zn	19.5	15.5	10.2	14.7
Cu	1.0	Trace.	Trace.	2.0

The zinc left in the cinder is always in the form of sulphide, together with a little iron and copper sulphide. Very little additional fuel or flux is required to smelt the cinder, but it is customary to add raw zincy ores, silicious ores, and some lead and copper ores, so that the charge will contain about 17 to 20% of zinc, 2½ to 4% copper, 3 to 10% lead, 15 to 20% sulphur, and 10% of burned lime, with the usual proportion of silica and iron to make a rather high silicious slag.

Owing to the low charge, very little reducing action is produced in the furnace, the aim being to oxidize and expel the lead and zinc and form a clean slag and a high-grade matte. The amount of fuel used is in proportion to the amount of zinc in the charge, and varies from 6% to 15%.

The rapidity of the smelting is governed by raising or lowering the silica. If advisable to run the zinc very low in the slag, the fuel is increased and the silica raised; this causes slower smelting and higher-grade matte with less zinc in the slag and more in the fume.

The writer's experience in smelting zincy ores does not agree with that of some other metallurgists in regard to the form in which zinc enters the slag; he finds that zinc is simply absorbed as oxide, that it does not form any special combination, and does not to any extent replace basic elements. Silicate of zinc is not formed, but oxide of zinc may be absorbed to a large extent, often to 20 or 25%, especially in basic irony slags, while with acid slags the zinc diminishes in direct ratio to the silica percentage. Lime, up to a certain proportion, aids in the absorption of zinc, but beyond 10 or 15% it reacts and expels it.

The following slag analyses, illustrating the above remarks very fully, are taken at random from the books of the American Zinc-Lead Smelter. The aim there is to reduce the zinc to as low a percentage in the slags as is consistent with good work and profitable results, and the average of yearly runs has been kept down to 5 or 6% metallic zinc, or 7 to 8% oxide. With lead smelters the aim would be to force zinc into the slags, and under proper conditions this can safely be done up to 20% oxide.

ZINCY SLAG ANALYSES.

	Straight Iron Slags			Iron and Manganese Slags.						Iron and Lime Slags.						
	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.	14.	15.	16.
	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
SiO_2	32.4	36.5	33.6	36.2	38.0	38.5	39.0	40.5	44.2	38.0	39.7	36.5	32.5	37.0	38.2	37.5
FeO	50.5	49.3	47.2	34.7	34.6	33.6	32.5	31.9	36.3	30.1	39.5	30.3	31.7	30.1	34.3	34.2
ZnO	15.2	12.5	14.2	13.0	14.2	12.5	11.8	11.2	7.2	11.4	10.1	9.3	9.3	8.2	3.7	3 1
CaO	.....	.....	2.2	4.0	2.5	4.6	4.2	4.0	2.0	15.4	7.8	18.5	18.5	19.4	20.4	19.2
MnO	.....	.....	.....	7.8	8.1	7.0	8.0	8.9	7.4	3.1	.....	2.0	4.2	1.3	3.0	2.1
Silver, ozs.	2.5	1.75	2.0	1.5	1.0	.75	$\frac{3}{16}$	¼	¼	¼	¼	½	¾	¼	¼	¼

A study of the above develops the fact that zinc absorption is higher when the silica is low and iron high; also that silver loss is then the greatest, while high silica, low iron, and high lime give the lowest percentage of zinc and silver. With low silica zinc is taken up largely, even in presence of a large proportion of lime, but with low silica and high iron it is taken up still more easily.

No evidence has been found of the formation of a zinc silicate in the slags, nor of any combination of the zinc with any other element than oxygen. Both sili-

cate and sulphide of zinc are infusible in the furnace, and tend to form accretions, and no amount of fluxing will dislodge these. The only effective way to discharge them from the furnace is to run down low and use a large quantity of fuel and a heavy blast. This decomposes both zinc sulphide and silicate, forming oxides which will enter the slag or go off as fume. Fully oxidized zinc ores, if run in a charge free from sulphur, are easily slagged off, but zinc will take up sulphur in the furnace from other ores and form artificial blende, its avidity for sulphur being next to that of copper.

Another fact not well known is that zinc is much less volatile than lead. Some experiments made to determine the fractional volatility of the metals shows that with ores in the sulphide form, exposed to the action of incandescent carbon and an air blast, the metals disappeared in the following order: Mercury, cadmium, antimony, lead, zinc. Under expert manipulation lead can be completely separated from zinc by fractional volatilization, but in a commercial way it is extremely hard to control this operation.

NOTES ON THE ZINC-LEAD PIGMENT.

The refined pigment produced at the Cañon City works consists of an intimate mixture of zinc and lead in very stable form.* Chemists differ in their analysis of the refined product; broadly it may be said to consist of a compound of zinc oxide and lead sulphate containing an excess of oxygen. The metallic constituents from an ordinary sample are as follows:

	Per Cent.
Zinc, metallic	47.33
Lead, metallic	24.92
Sulphur	2.96
Oxide of iron, etc	0.45
Oxygen	24.34

No combination of these can be made from a chemical standpoint without showing an excess of oxygen. The raw fume before refining contains zinc and lead as sulphides, sulphites, and sulphates; these are reduced by the refining process to more stable compounds; sulphur is eliminated and oxygen taken up. The material driven off and collected in the refining process is an interesting mixture, which varies with the class of ore used. An analysis made by J. F. Sleeper of material collected from refining fume made from Leadville sulphides gave:

	Per Cent.		Per Cent.
Sulphur	0.06	Bismuth	0.08
Silica	7.50	Antimony	0.04
Selenium	0.08	Copper	0.12
Organic matter	4.35	Lead	2.84
Water	4.60	Ammonia	5.76
Cadmium	0.32	Fluorine	40.70
Zinc	9.75	Chlorine	2.13
Mercury	2.61	Sulphuric acid	4.39
Aluminum	8.56	Hydrofluosilicic acid	2.92
Iron	4.48	Traces of indium, thallium and gallium.	

Other samples contain much more cadmium and selenium, but all samples contain a large amount of fluorine, fluoride of alumina, and ammonia being in excess. There is a peculiar waxy organic matter, and also ammonia, which doubtless comes from the fuel used, and mercury is nearly always present in small

* U. S. Patent No. 477,488.

quantity. The amount of this dust collected from the treatment of one ton of ore does not exceed one pound; consequently the concentration is very great, enabling the determination of many elements which could not otherwise be discovered in the ore itself.

This process has been in use in a commercial way at Cañon City for 10 years, and the pigment made has secured a regular market demand. This may not be the best or the most economical method of handling zincy ores, but it has been successful, and the zinc and lead produced in pigment bring as much in the market as they would were they reduced to metal, and the demand is fully as large. It should be remembered that the demand for zinc, either as metal or as pigment, is limited, and those who have spent years in trying to utilize metallic zinc from ordinary smelting may not have considered that if they succeeded in making it there is not a large market for it. This is one great difficulty in attempting to utilize the zinc from the Broken Hill ores of Australia. If all the zinc from these mines were saved and put on the market, in any form, however cheap, it would simply stagnate the trade. By far the greater use is found in the white lead substitute which the writer has attempted to produce.

MODIFICATIONS OF THE CAÑON CITY PROCESS.

Zincy ores do not always contain value enough in gold and silver to ship to smelting centers. The result is that there are vast quantities on the dumps and in the mines which carry less than $8 or $10 value in gold and silver, with from 25 to 35% of zinc and 5 to 10% of lead. Owing to the distance from fuel, and high costs generally, such works as those at Cañon City cannot be carried to the mines, and it is doubtful if such ores will ever be worked. On the other hand, when the value in silver, gold, and copper is large enough so that the zinc and lead can be thrown away and still leave a profit, then a modified process can be applied to good advantage. Especially is this the case when fuel and labor are reasonably cheap, and other ores can be had for mixtures. Even heap roasting and treatment in the Cañon City blast furnace for the production of coarse matte is often admissible and more profitable than long shipments of the raw ore to the markets. The commendable features of the Cañon City process are cheapness of treatment and a fair saving of values.

An ore not uncommon to many of the isolated mining camps of old Mexico contains 25% zinc, 20% lead, and from 15 to 25 ozs. of silver. The lead in such an ore contains too much value to be discarded. Such an ore, after a moderate roasting, just sufficient to change the sulphide of lead into a sulphate, can be fused on a hearth furnace and considerable of the lead be recovered in the form of metal. The residues can then be smelted with other ores in the Bartlett blast furnace and a fair recovery be made of the silver as well as of the lead.

Another modification of the process, when sulphuric acid can be obtained reasonably cheap, is to work the ore by the regular process, but instead of refining the fume into pigment, it is matted with weak sulphuric acid, which dissolves the zinc completely, leaving the lead, in the form of sulphate, easily reduced to chemically pure pig lead. The zinc can be recovered from the sulphuric acid solution in several ways: for example, by precipitation with magnesite, forming a soluble salt of magnesia and production of zinc oxide, which can be reduced to

metal on the spot or shipped to the nearest spelter works. The location of the ore, and the character of it, together with fuel supply and market conditions, must determine what modification of the process is more suitable to adopt.

The blast furnace is especially well adapted to the treatment of impure copper ores, since it expels arsenic and antimony almost completely. Lead can also be expelled from mattes, and no speiss is ever formed even with the most impure ores. Another point should not be overlooked in reference to the enrichment of the matte. It is possible with this furnace to bring up low-grade mattes to a high concentration without previous roasting. This is due to the scorification action of the heavy blast entering a low charge. Mattes of from 15 to 20% copper can easily be brought up to 45 or 50% in one run, and a second run with proper fluxes will bring it up to 60 or 70%. Other modifications will suggest themselves to the intelligent metallurgist.

MINE LABOR AS REGULATED BY LAW IN THE UNITED STATES.

By William Franklin Willoughby.

The conditions under which mining operations must be conducted offer such peculiar dangers that most nations have found it desirable to enact special laws determining the manner in which this industry shall be prosecuted. The present article is an attempt to show in a rapid sketch how this obligation has been interpreted by the different commonwealths of the United States. In other words, it is desired to show to what extent the mining of coal in the United States is considered an industry requiring special regulation, and what is the character of this regulation as it exists at the present time.

Coal is not mined in any considerable quantities except in a few of the States. We therefore find that of the 45 States and three organized Territories, 24, or one-half, do not possess any special laws relating to coal-mining. Four other States have but a few scattering laws relating to particular features of mining. Thus in Utah the constitution directs the legislature to pass laws prohibiting the employment of children under 14 years of age, and of women of any age, in underground mines, though as yet no such law has been enacted, and there are laws requiring mine-shafts to be fenced, and two outlets to be provided in the case of large mines.* Idaho and South Dakota have mine inspectors, but merely give to them the general duty of looking after the security of miners, without specifying the particular conditions under which mining operations shall be carried on. New Jersey has provided that the inspectors of factories shall also inspect mines, and prohibits the employment of children under 12 years of age in mines.

The remaining 20 States, however, Alabama, Arkansas, California, Colorado, Illinois, Indiana, Iowa, Kansas, Kentucky, Maryland, Missouri, Montana, New Mexico, New York, Ohio, Pennsylvania, Tennessee, Washington, West Virginia, and Wyoming, have elaborated more or less detailed codes of mining regulations. A study of the extent to which coal-mining is subjected to special legal regulation therefore involves only the consideration of the legislation of these 20 States.

An examination of these various laws shows that a very general agreement has been reached by the different legislatures in regard to the character of the regulation that should be provided. The same provisions and even the same phraseology are found repeated in the statutes of State after State. The differences that exist are mainly in the extent to which regulation is attempted, and the

* As this article goes to the printer information is received that Utah has recently passed a law regulating mine labor in some detail.

efficiency of the system that is provided for its enforcement. It is quite feasible, therefore, to study the legislation of all of the States as a whole.

If these mining codes be examined analytically, it will be seen that their purposes can be grouped in six distinct classes: (1) The regulation of the employment of women and children; (2) the formulation of a set of rules or regulations setting forth more or less specifically the manner in which the operations of mining must be conducted; (3) the securing of competent men for responsible positions largely through a system of State examination and the granting of certificates of competency; (4) the requirement that all fatal or serious accidents be reported and investigated; (5) the protection of the rights of miners through regulating the manner of weighing or measuring the quantity of coal mined, and the frequency and character of wage payments; and (6) the provision of an inspection service for the purpose of ensuring that the laws relating to mining are duly enforced. By taking up each of these points in turn we will be able to obtain a clearer idea of the extent to which these various objects have been provided for by the different States.

Women and Children's Work.—First, in regard to the extent to which the employment of women and children has been specifically prohibited. Of the 20 States to which we have accredited mining codes, ten, Alabama, Arkansas, Colorado, Illinois, Indiana, Missouri, Pennsylvania, Washington, West Virginia, and Wyoming, absolutely forbid the employment of women either in or about mines (clerical work in offices sometimes excepted). As regards the employment of children, however, 18 out of the 20 prohibit the employment of both sexes under a certain minimum age. Of these, Alabama has the least rigid system of exclusion; her law simply specifying that children under 10 years of age cannot be employed in or about mines. The three States, Iowa, Tennessee, and West Virginia come next with a minimum age of 12 years. Following them are Colorado, Kansas, Missouri, New Jersey, and Ohio, all excluding children under 12 years of age, and in addition those children under 14 in Missouri, those under 15 in New Jersey, and those under 16 in the other States unless they are able to read and write. Nine States, Arkansas, Idaho, Illinois, Indiana, Montana, Pennsylvania, South Dakota, Washington, and Wyoming, have fixed the minimum employment age at 14, and Arkansas has in addition required boys under 16 to be able to read and write as a condition precedent to employment. The United States statute on this subject simply provides that children under 12 years of age cannot be employed in any mine in the Territories. These prohibitions apply to work above as well as under ground, with the exception of Pennsylvania, where the employment of children under 14 in mines is prohibited, but boys between the ages of 12 and 14 are allowed to work about mines.

Length of the Working Day.—It is interesting to note that but two States, apparently, have attempted to regulate the hours of labor in this particular industry. Wyoming has declared eight hours to constitute the maximum length of the working day that can be required of any miner unless extra efforts are necessary to save life. This, however, does not prevent mines from being operated any number of hours by the employment of several shifts of workers. In Ohio a similar law limiting work by employees of a mine or railroad to ten

hours a day was recently declared to be unconstitutional, as interfering with the liberty of contract and the property rights of the employer. A number of States, however, have declared eight or ten hours to constitute a day's work in the absence of any contract to the contrary.

Mine Regulations Proper.—Much the greater portion of all mining laws is devoted to setting forth in greater or less detail the various regulations that must be observed in the working of mines. The development of these has been in almost all cases one of gradual evolution. The earlier laws simply provided that proper precautions should be taken to secure the safety of miners. From year to year additional legislation was enacted specifying particular conditions that must be observed. In time these provisions were in cases gathered together and re-enacted as a single law, thus constituting what might be called a mining code.

It is manifestly impracticable to attempt here to describe the exact character of the legislation in each State individually. Fortunately even the desirability of doing this does not exist. The same provisions are found reproduced with but little change in the laws of almost all of the States; the only difference being the extent to which the formulation of mining regulations has been carried. The following recapitulation of the essential provisions of this legislation gives all the material points covered by any of the mining laws. Some of the laws notably those of Pennsylvania, cover practically all of the points here enumerated, while others merely include the most important.

The mining code of an American State in its most developed form, therefore, provides (1) that every owner, operator, or superintendent of a mine employing over a certain number of persons, usually ten, shall cause to be prepared an accurate map or plan of such mine on a scale of 100 or 200 ft. to the inch, showing all the workings of the mine; that this map shall be revised at least once in six months in order to show new workings; that when a mine is abandoned a final accurate map must be made of it; and that copies of these maps must be furnished to the mine inspectors and other copies be kept where they can be readily inspected at the mines; (2) that in mines where 20, sometimes ten, persons are employed there must be at least two escapement-openings to the surface from each seam, separated from each other by natural strata of a certain thickness, 100 or 150 ft.; (3) that mines must be so ventilated by artificial means that there will be furnished a minimum of 100 cu. ft. of air per minute for each person employed; (4) that doors used to direct or control ventilation be so hung that they will close of their own accord, and that doorkeepers be provided for the more important passages; (5) that an adequate supply of timber for props be constantly available; (6) that suitable means be provided for raising and lowering workingmen in mines, and, to secure this, that the cage used for this purpose have a top or bonnet of metal to protect the passengers from falling articles or rock, that no single cable be used, that the cage be equipped with a safety-catch, and that the cable-drum be provided with flanges and a brake; (7) that all passageways through which cars pass have shelter-holes in the sides not less than 15 or 30 feet apart into which workingmen may retreat to avoid passing cars; (8) that the mines be kept well drained; (9) that there be a metal speaking-tube or other means of vocal communication between the bottom and top of all shafts; (10) that

a certain code of signals, usually as specified in the act, be employed to regulate the movement of the cages up and down the shafts; (11) that only authorized persons be allowed to ride on loaded cars or cages; (12) that no coal be hoisted while men are ascending or descending the shafts; (13) that all machinery be properly guarded; (14) that abandoned passages be closed; (15) that shaft-openings be fenced; (16) that steam-boilers be inspected at certain intervals of time; (17) that only a certain quality of vegetable or animal oil be used for lighting; (18) that precautions be taken to prevent injury from falling coal or rock; (19) for the regulation of blasting operations; (20) that copies of the mining rules be conspicuously posted. For mines generating fire-damp especial precautions must be taken, as (21) that they must be examined every morning with a safety-lamp before the miners go to work; (22) that all safety-lamps be owned by the mine-owner; and (23) that bore-holes of a certain depth be kept in advance of the workings of any passage where approaching old workings.

The above are the usual provisions of a mining code. In a few cases conditions are given which are not included in this list. Thus Pennsylvania requires all stables in mines to be built in the solid strata without the use of wood; Pennsylvania and Montana, that stretchers be provided for removing injured workingmen; and Kansas, that all blasts shall be fired by special shot-firers.

Examination of Mine and Fire Bosses.—Rules, however, can never replace the personal element. The best of rules are of but little avail unless competent men can be secured to supervise their application. The most significant and important feature of the whole system of mine regulation, therefore, is that by which a number of States have sought through a system of examinations to insure that those in charge of the actual operations of mining shall be competent men. The positions thus specially provided for are those of mine foreman or boss, fire boss, and occasionally that of hoisting engineer.

The majority of the mining States, including California, Colorado, Kansas, Maryland, New Mexico, Tennessee, and West Virginia, simply provide that the underground operations shall be in charge of a competent superintendent or mining boss, whose especial duties are to see that a proper amount of ventilation is provided, that the walls and roof are properly timbered, etc., and that in the case of all mines generating fire-damp there shall be employed a "fire boss" with the duty of examining all working places for gas every morning before the miners go to work.

The more important mining States, however, have gone much further. They have treated the positions of mine boss and fire boss as of such responsibility that no one should be allowed to fill them until duly certified by the State to possess the required competency and experience. These positions have been consequently put into the same category of licensed occupations as piloting and plumbing. States have thus on the one hand required every mine to be under the supervision of such officers, and on the other that these officers must be in possession of certificates of competency granted after satisfactory examination.

The Pennsylvania law, for example, provides that on the petition of any mine inspector the court of common pleas in any county in the district shall appoint a board of examiners, to consist of a mine inspector, a miner who has received a certificate of competency, and an operator or superintendent, whose duty it shall be

to examine all applicants for the positions of mine or fire boss. To secure a certificate of competency applicants must be 23 years of age or over, have had five years' experience as miners in bituminous coal-mines of the State, after they were 15 years of age, be citizens of the State, of good moral character, and pass an examination as to their knowledge of mining. Certificates of two grades are granted; that of the first grade, to those who have had experience in and are therefore qualified to serve in mines producing gas; and of the second grade, to those who have not had this experience and therefore cannot be employed in such mines. In order to protect those serving as mine or fire boss when the act was passed it was provided that "certificates of service" should be granted to those who had been employed with the same company during the year preceding, with the proviso, however, that before they could take service with another concern they should obtain a certificate of competency.

Illinois, Alabama, Montana, and Wyoming have followed the lead of Pennsylvania and have enacted almost identical provisions, Illinois in addition requiring that all hoisting engineers shall also be provided with certificates of competency.

Reporting of Accidents.—As the prevention of accidents constitutes the most important purpose of mine regulation, it is evidently very desirable that accurate information should be obtained concerning the frequency and causes of accidents in order to determine the responsibility for such occurrences and whether any progress in their prevention is being made. Every one of the 20 States possessing factory codes, with possibly one or two exceptions, requires that the mine owner or superintendent shall report to the inspector of mines every accident resulting in death or serious injury to an employee. It is further provided that in case of fatal accidents the coroner shall be notified and an inquiry held to determine the person at fault for its occurrence. The mine inspectors are also required to embody in their regular reports statements of all accidents occurring during the year. The information thus afforded is of great value; but unfortunately in no case does it approach anything like desirable detail. The term "serious injury" which occurs in all the acts is altogether too vague and uncertain. In order to be of the maximum value the report of each accident should show the cause of the accident, the extent of the injury caused, whether resulting in death, total or partial permanent incapacity for labor, or temporary disability and the length of time so disabled, the age of the person injured, and whether the accident was due to the fault of the person injured, to another, or to an unavoidable cause. Whatever the information obtained, however, it is to be regretted that greater uniformity does not exist in the mode of reporting accidents in the different States. Improvement in the collection of statistics of accidents to miners is largely dependent upon the mine inspectors of the different States adopting the same form of report for accidents.

Inspection of Mines.—We now turn to what in a way may be called the keynote to the whole system of the State regulation of mines—that of the appointment of State officers or mine inspectors with the duty of personally supervising or controlling certain features in mining and seeing that the conditions required by the States are complied with. It has been the universal experience that labor legislation is of little utility unless some system of government supervision is at

the same time provided. In a way, therefore, the measure of the efficiency of mine regulation is that of the efficiency of mine inspection.

Most of the mining States have recognized this necessity. Twenty-one States have mine inspectors. In the majority of these, as Alabama, Arkansas, Idaho, Kansas, Kentucky, Maryland, Maine, Missouri, Montana, New Jersey, New York, South Dakota, Tennessee, and Wyoming, but a single inspector has been provided for. In Tennessee the inspector is the Commissioner of the Bureau of Labor and Mining Statistics. In New Jersey, where there are no coal-mines, the offices of inspector of factories and mines are combined. Maine, which also has no coal-mines, has an inspector of factories, workshops, and mines, but his duties seem to be of a purely statistical character. The United States statute provides that an inspector of mines shall be appointed by the President for each Territory producing 1000 tons of coal yearly. Indiana and Colorado have each two inspectors of mines. Iowa and West Virginia have three inspectors; and Washington provides for a State geologist to act as mine inspector with two assistant inspectors. Illinois has seven inspectors; Ohio, one chief inspector and seven district inspectors; and Pennsylvania provides that the State shall be divided into districts each to contain not less than 60 or more than 80 mines, and an inspector to be appointed for each district.

The duties of the inspectors are generally stated to be to inspect all mines with specified frequency, yearly, semi-annually, or quarterly, and to see that all the requirements of the laws relating to mining are strictly complied with. In addition to these general duties, however, it is usual to specify that they shall keep an exact record of all inspections, and that they shall report annually or biennially, showing particularly the condition of the mining industry and the number of accidents. These reports, therefore, serve the double purpose of showing the results accomplished by inspection and of giving general statistics and other information concerning mines.

For the enforcement of mining laws with their technical provisions it is evident that not only is a considerable technical knowledge required of the inspectors, but that no small degree of discretion must be left to them in the enforcement of the obligations that they impose. It is highly desirable, therefore, that competent and specially trained men should be secured for these positions. As the most certain way of accomplishing this, the practice is now becoming general for these officers to be selected only upon satisfying certain requirements and passing a wholly or partly competitive examination.

The Pennsylvania system thus provides for a board of examiners composed of two mining engineers and three other persons who have passed examinations as mine inspectors or mine foremen to examine candidates for the position of inspector of mines. The examination must be in writing, with an oral examination concerning explosive gases and safety-lamps. Candidates must be citizens of the State, of temperate habits, 30 years of age or over, have had at least five years' experience in bituminous-coal mining in the State, and an experience with mines generating fire-damp. The names of successful candidates are certified to the governor, who makes the appointments.

Illinois, Indiana, California, Iowa, and Washington have similar boards of

examiners. The other States which do not possess examining boards usually specify that inspectors must be of a certain age, possess both a theoretical and a practical knowledge of mining, and have had an experience in practical mine-work for a certain number of years.

Payment of Wages.—The last class of mining laws are in their nature quite distinct from those that we have been considering. They have for their object the regulation of the relations between the mine operators and their employees. The economic dependence of the miners has undoubtedly in some cases been taken advantage of in the past, and the miners defrauded, or at least unjustly treated, in a number of ways. The two greatest grievances of the miners have been that the employers have not given them credit for all the coal mined by them; and, secondly, that they have been compelled to trade at stores created and run by the mine-owners.

In itself the creation by the companies of stores to supply the wants of their employees possesses nothing detrimental to the rights or liberties of the workingmen. On the contrary there is no reason why they should not serve a very useful purpose. Unfortunately there can be little doubt that in many instances these stores have been used by the companies as means of oppression. Miners were required to trade at the store run by their employers, and to insure their doing so they were frequently paid in scrip orders on the store instead of in money. A system of credit was at the same time practiced which kept the workingmen constantly in the debt of the company; and, as the wages were withheld to meet this indebtedness, employees would frequently go for long periods of time without receiving any money that they could dispose of as they saw fit.

For a long time this "truck system," as it is called, constituted one of the greatest sources of friction that existed between the mine-owners and their employees. The miners themselves did not possess sufficiently strong organizations to offer a successful resistance. Great pressure was therefore brought to bear upon the legislature of all of the mining States to prohibit by law the system of company stores; and most of the States passed laws to this effect. This prohibition has taken the two forms of either directly forbidding mining companies to own or control stores, or, more usually, requiring all wages to be paid in money, or if in scrip, that this scrip be redeemable upon demand in money. At the same time the payment of wages as often as once every one or two weeks would be made obligatory.

The constitutionality of these laws has been attacked in a good many instances, and to a large extent they have been declared unconstitutional. In Pennsylvania, Illinois, Missouri, and Tennessee they were declared void as violating the liberty of contract guaranteed by the constitution. In Tennessee the curious ground was taken that such a law indirectly provided for imprisonment for debt which was prohibited by the constitution. In Indiana, however, the law was upheld; and New York avoided any constitutional objection by limiting the scope of the law to corporations.

In spite of the fact that laws regulating the payment of wages have been declared void in so many States, the results aimed at have, in great part, been accomplished. This has been due, on the one hand, to the arousing of public

opinion on the subject, and, on the other, to the fact that employers are beginning to recognize more fully their obligations toward their employees.

As regards the second complaint, that concerning the manner of determining the amount of coal mined by each miner, practically all of the 20 States under consideration have enacted laws the purpose of which is to insure that the coal be honestly weighed. There is little difference between the legislation of the different States. The typical method of regulation is to provide that at all mines there shall be provided suitable and accurate scales for weighing coal; that these scales shall be inspected periodically by the mine inspectors or the miners themselves; that the weighman must make oath honestly to perform his duties and keep an accurate record of the amount of coal weighed; that these records shall be open to inspection; and finally, as an additional precaution, that the miners shall have the right to employ a "check-weighman" who shall be permitted by the company to superintend the weighing of all coal and thus control the work of the company's employee. In case such a check-weighman is employed, he must also be sworn, and must keep an accurate record of the coal weighed. Some States, as Pennsylvania and West Virginia, also require that all cars shall be numbered, and their weight or capacity plainly marked on each one.

A more direct intervention of the State is that where a few States, notably Illinois, Indiana, West Virginia, Kansas, Iowa, Missouri, and Washington, have made it compulsory upon all mine operators to weigh coal before it is screened. This law has been resisted on the part of the mine operators, and in Illinois at least has been declared unconstitutional as being special legislation and depriving persons of the liberty of making their own contracts.

We have now passed in review the various ways in which the operation of mining has been subjected to special regulation by the States. Experience has amply demonstrated that this interference on the part of government, and the formulation of regulations setting forth in detail the various precautions that must be taken, have been absolutely necessary for the protection of miners against accidents. The present degree of regulation has been the result of a gradual growth, and the goal has as yet been by no means reached. Pennsylvania, Illinois, Ohio, West Virginia, and several other important coal-mining States are now in possession of quite complete mining codes, but the majority of the States have far from reached their standard. The latter have, however, the more complete legislation of the former States as models, and not a year passes without additions and improvements to the mining laws of some of them.

To one looking over the whole field, it would seem that the most important step that can be taken is that of the extension of the system already practiced by a number of States of insuring through examination and the granting of certificates that responsible positions are filled by competent men. Mr. James Bryce, with his accustomed perspicuity, has said that good men can make any political system work tolerably, but that no system however perfect will give satisfactory results unless in the hands of honest and capable persons. What is true of political machinery is equally true of industrial organizations. Certainly it is desirable to have a good code of mining regulations, but it is more important still that capable men be secured to direct their application.

THE MINING STOCK EXCHANGES IN 1896.

We give in the following pages reports from all the leading exchanges in which there are dealings in American mining stocks. In this country the financial conditions have not been favorable to speculation and the general condition of affairs on the exchanges has been one of quietude and depression rather than of activity. The partial exception found has been in the stocks of the copper mining companies. It is to be noted that while there has been great activity in gold mining, the shares of gold mining companies have attracted little attention. The interest shown has been in mining work rather than in speculation.

The Boston Stock Market in 1896.

The market for mining stocks in Boston for 1896 was fairly active, especially in the copper stocks, which are a Boston specialty. The leading stock in point of activity and wide fluctuations in price is that of the Boston & Montana Copper Company, the dealings in which aggregated more than all the others together. The stock is widely distributed and is a favorite with speculators, who are enabled to manipulate prices to suit their purposes. The company paid $10 per share in dividends for the year, and the stock sold at a range from $65 to $96.50 per share during the year. Its lowest point was touched on January 7, and the highest November 16. Its closing price was about $92.

Next in point of activity we note Butte & Boston, a stock which was supposed to have considerable merit, and sold at one time last year at $28.50 per share. Owing to some legal complications the mine was closed down early in the year, and the stock after selling at $13.25 dropped to $1.25 per share. A large block of stock owned by the Davis estate was purchased by a syndicate, and steps were taken to reorganize the company and assess the stockholders $10 per share. The company has been under the same management as the Boston & Montana, and is believed to have a good property. Late sales of the stock were at $7.37½ per share.

The Old Dominion Copper Company has a mine which belongs to the same class as the foregoing, and has many friends. The stock was quite largely dealt in, and was put on the unlisted department of the exchange in December, 1895, at $17 per share. It sold at $23.25 during the year, and as low as $10.25. Its closing price was about $16.

The Lake Superior mines had a fair share of business during the year, with the inevitable ups and downs characterizing this class of stocks. Foremost of all is the old reliable Calumet & Hecla, with its famous record. Its lowest price for the year was $295, and the highest touched was $335 for a single share. It paid $25 per share in dividends and at the close sold at about $325, with a dividend recently paid.

The Tamarack had hard luck and was obliged to reduce its dividends from $8 per share last year to $6. Its market record is from $120 to $65; recently the mine has improved in its outlook and the closing price was about $92 per share. Since its organization it has paid $830,000 in dividends.

Quincy at one time sold up to $134 per share; its lowest record was $104 ex-dividend in August. The company paid $20 in dividends for the year, and at its closing price, $120, is considered a safe investment. The scrip, on which $18.75 has been paid, sold at $63 and $97 respectively. A further and final installment of $6.25 per share is due in April next, when new stock will be issued which will partake of future dividends.

Osceola is a stock of which many good things are said. It sold during the year at $20.50, the lowest, and $31.50, the highest figure. The closing price was $28.50. The dividends for the year were $2.50 per share against $3 for 1895.

Kearsarge is quite a favorite speculative stock, and was largely dealt in this year. The price ranged from $8.75 to $17.75; at the close it was selling at $15.50. The company did not pay a dividend during the year.

Franklin, one of the oldest mines on the list, was not very active. The stock sold at $8 and $15; closing price about $11. The original mine is supposed to be nearly exhausted, and work on the new property called the Franklin, Jr., was pushed the past season with favorable results.

Atlantic sold at $15 in August, which was the lowest for the year. In November it touched $23.50, the closing price being about $21.50. The stockholders have not received a dividend for several years, but it is confidently expected that dividends will be resumed in 1897.

The Tamarack, Jr., mine is being rapidly developed and the outlook is considered encouraging. Its market price has ruled from $8 to $17.50 per share, at closing $13.50.

Wolverine is another mine of which great things are expected. It sold in January at $5.87½, and later touched $9.37½, which was the highest for the year, closing at $8.50.

Centennial has been reorganized and assessed $2 per share. It sold at one time at 10c. per share, assessment unpaid, and later at $2.50, assessment paid.

Arnold was also assessed $1 per share. Before the assessment was levied it sold at $1.75 per share, and also at 25c. The latest sale was at 30c., assessment unpaid.

Tecumseh is a mine which is being developed, and is said to have the Calumet & Hecla vein. The latest reports state that as depth is attained the rock improves in appearance. The stock is well held and sold at $1.75 at one time during the year. Later it touched $4.50. The last sales were at $3. National sold at 75c., the only price for the year. Allouez sold as high as $1 and as low as 25c. per share.

FLUCTUATIONS OF MINING STOCKS AT BOSTON DURING 1896.

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Copper:													
Allouez, Mich	$25	1.00	.30	1.00	.....	1.00	.....	1.25	1.00	1.00	.....	.25	.....
Arnold, Mich	25	1.57	1.00	1.50	1.13	1.38	1.00	.....	.....	1.63	1.50	.....	.....
Atlantic, Mich	25	17.50	15.00	20.25	17.00	20.00	16.00	18.25	17.25	21.75	17.00	22.25	17.75
Boston & Montana, Mont	25	69.00	65.00	81.00	72.00	79.75	72.00	77.13	73.75	88.75	74.00	91.50	82.13
Butte & Boston, Mont	25	13.25	11.00	11.00	2.88	3.00	1.00	3.25	1.50	3.00	2.25	2.63	2.00
Calumet & Hecla, Mich	25	298.00	290.00	305.00	300.00	310.00	305.00	309.00	305.00	311.00	300.00	310.00	303.00
Centennial, Mich	25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.10	.....
Franklin, Mich	25	12.25	10.50	15.00	13.50	15.00	11.00	12.50	11.00	14.25	10.38	11.50	8.50
Kearsarge, Mich	25	13.50	10.50	13.50	12.13	17.75	10.38	11.75	9.00	14.50	7.88	14.25	10.50
Old Dominion, Ariz	25	19.00	16.38	19.50	17.25	18.50	15.00	17.00	13.50	22.50	15.00	22.00	13.38
Osceola, Mich	25	27.25	22.13	29.00	23.50	28.35	24.50	26.00	24.88	31.50	24.25	29.50	25.00
Quincy, Mich	25	125.00	120.00	134.00	122.50	130.00	124.00	125.00	115.00	125.00	112.00	120.00	112.00
Scrip	.....	70.00	65.00	75.00	69.00	81.25	71.00	81.25	78.00	85.00	77.50	83.50	80.00
Tamarack, Mich	25	120.00	111.00	117.50	103.00	112.00	86.00	100.00	85.00	104.00	88.00	95.00	82.00
Scrip	.....	90.00	80.00	88.00	82.00	85.00	.....	.....	.....	.....	.....	82.00	.....
Tamarack, Jr., Mich	25	17.50	12.50	17.50	14.00	16.25	14.50	15.00	14.50	14.50	12.25	13.50	10.00
Tecumseh, Mich	25	3.25	2.00	4.50	3.25	4.00	3.00	3.00	.....	3.25	2.50	3.10	1.75
Wolverine, Mich	25	7.00	5.75	8.00	6.50	8.00	6.63	7.25	6.50	8.00	6.38	7.75	6.50
Gold:													
Boston & C. C., Colo	1	.50	.35	.43	.30	.38	.30	.28	.....	.....	.....	.....	.....
Gold Coin, Colo	1	1.00	.90	1.00	.80	.90	.60	.80	.60	.70	.50	.60	.48
Merced, Cal	15	34.75	28.25	34.50	27.50	33.00	19.00	21.00	11.25	14.13	9.50	14.38	6.20
Pioneer, Cal	10	4.88	4.00	7.25	4.00	10.88	7.50	10.00	7.75	9.13	5.75	6.13	3.00
Santa Rosa, Cal	10	1.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Santa Ysabel, Cal	5	11.00	9.00	14.75	11.00	16.38	13.00	14.50	12.50	13.00	12.00	13.00	10.00
Miscellaneous:													
Dominion Coal, N. S	100	11.00	9.38	14.25	10.50	15.00	13.25	15.00	12.75	16.00	13.50	15.50	8.50
Dominion Coal, pref	100	90.00	83.00	92.00	88.00	94.00	92.00	92.00	89.00	93.00	92.00	93.00	91.25
Illinois Steel, Ill	100	64.50	62.00	70.00	65.00	67.00	61.00	75.00	67.00	72.50	68.00	65.00	59.00
Lake Superior Iron, Mich	25	29.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	28.50	.....
Napa Cons. Quicks., Cal	7	7.50	7.00	7.00	6.88	10.00	7.00	.....	.....	7.00	.....	7.00	.....
Westinghouse E. & M., Pa.	50	31.50	29.50	31.50	28.75	31.00	28.00	30.50	29.00	29.00	.....	29.00	.....
Preferred	50	54.00	52.00	53.50	52.38	53.00	52.00	54.00	52.75	53.00	52.00	54.00	50.75

Name and Location of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Copper:													
Allouez, Mich	.50	.....	.....	.....	.50	.40	.50	.....	1.00	.75	.....	.....	2,620
Arnold, Mich	.....	.....	1.25	.....	1.38	.....	1.25	1.00	1.25	.25	.50	.25	14,665
Atlantic, Mich	19.00	16.00	16.00	.....	18.50	15.50	18.00	17.00	24.00	18.75	22.00	20.50	13,371
Boston & Montana, Mont	89.00	75.00	78.00	72.25	91.25	76.75	90.25	80.50	96.50	88.25	93 00	88.50	817,326
Butte & Boston, Mont	2.00	1.63	2.00	1.50	3.50	1.50	3.00	1.50	6.25	2.13	7.38	5.00	260,542
Calumet & Hecla, Mich	303.00	.....	310.00	300.00	315.00	300.50	320.00	310.00	335.00	315.00	325.00	320.00	3,087
Centennial, Mich	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3.13	2.00	9,733
Franklin, Mich	11.00	8.50	10.00	8.00	10.00	8.00	9.00	.....	12.50	10.50	12.00	11.00	6,859
Kearsarge, Mich	12.50	9.50	10.00	9.00	14.00	9.50	13.50	9.75	17.75	13.00	17.00	15.38	51,792
Old Dominion, Ariz	16.38	11.00	14.50	10.25	17.75	14.50	17.25	14.75	19.75	16.00	18.00	15.88	153,625
Osceola, Mich	26.00	21.00	23.75	20.50	28.00	24.00	27.50	24.75	32.00	27.00	30.75	29.00	28,860
Quincy, Mich	115.00	107.00	108.00	104.00	111.00	.....	114.00	110.00	130.00	116.00	120.00	118.00	5,667
Scrip	84.00	75.00	78.00	76.00	88.00	77.00	85.00	80.00	97.00	88.00	95.00	94.50	4,709
Tamarack, Mich	84.00	72.00	75.00	65.00	90.00	70.00	88.00	81.00	102.00	85.00	95.00	91.00	12,557
Scrip	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	436
Tamarack, Jr., Mich	10.75	9.00	9.00	8.00	13.50	12.00	12.00	11.75	15.50	11.50	16.00	13.00	17,689
Tecumseh, Mich	2.00	.....	.....	.....	4.00	2.00	4.00	3.25	4.00	3.50	3.25	3.00	11,605
Wolverine, Mich	7.50	6.25	6.50	6.00	7.50	6.00	7.50	6.75	9.38	7.25	9.50	8.25	35,026
Gold:													
Boston & C. C., Colo	.....	.....	.....	.....	.20	.....	.....	.....	.20	.10	.10	.....	23,280
Gold Coin, Colo	.50	.45	.55	.50	2.50	.....	3.00	2.35	3.20	2.75	3.23	3.05	138,851
Merced, Cal	7.50	3.75	5.75	3.75	8.75	7.50	8.00	5.75	10.00	7.00	7.50	7.00	103,816
Pioneer, Cal	5.50	3.13	4.25	3.25	6.75	4.00	6.50	4.88	8.00	4.50	5.63	4.38	334,532
Santa Rosa, Cal	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	650
Santa Ysabel, Cal	9.00	6.75	9.00	7.00	10.00	9.00	10.00	8.00	11.88	9.25	11.75	10.50	37,449
Miscellaneous:													
Dominion Coal, N. S	10.25	7.88	8.00	6.25	8.50	7.50	8.75	7.00	12.00	9.38	11.50	10.00	183,357
Dominion Coal, pref	86.00	80.00	81.00	80.00	81.00	.....	84.00	80.00	90.00	85.00	92.50	91.00	3,471
Illinois Steel, Ill	60.00	49.50	45.50	36.00	48.50	37.00	47.00	39.75	57.00	47.00	48.25	43.00	34,425
Lake Superior Iron, Mich	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	260
Napa Cons. Quicks., Cal	.....	.....	.....	.....	7.00	6.50	7.00	6.88	6.75	.....	.....	.....	1,545
Westinghouse E. & M., Pa.	27.88	24.00	24.50	23.00	24.00	23.25	25.00	23.00	28.00	26.00	.....	.....	7,862
Preferred	50.00	48.75	49.50	48.50	51.00	49.50	50.00	48.75	51.75	49.50	52.00	50.00	4,816

NOTE.—Total sales for the year, 1,324,483 shares.

There have been a few sales of Lake Superior Iron Company stock at $28.50 @$29 per share, but very little of it comes on this market.

The gold and silver mines were more or less active during the year, but the speculation has been confined to two or three specialties. The Merced showed the widest fluctuations. When first dealt in it sold up to $35 per share, which was considered at the time much above its real value. Later it was assessed $2 per share and sold at $3.50; since it has gradually worked up to $8.50. Its closing price was about $6.75.

Pioneer was the favorite gold stock and was quite active. Last winter it was pushed up to $10.87½ per share; since then it has sold down to $3.25, and up again to $9.12½. It closed about $5.

Santa Ysabel was also subject to wide fluctuations; its lowest price was $7; highest, $16.37½, and closing about $10.

Gold Coin mines was originally put on the market with 1,000,000 shares of a par value of $1. Last spring it was converted to 200,000 shares, of a par value of $5. It paid three dividends of 1½c. per share on 1,000,000 shares, and under the new deal two dividends of 10c. per share. Its market price was 47½c. under the former, and $3.22½ under the latter. It closed at $3.05. The Boston & Cripple Creek stock early in the year sold at 47½c.; later its market price was 10c.

The Napa Quicksilver Company paid 80c. in dividends for the year. Its stock is largely held for its dividends, and is not often on the market. It sold at $6.50@$7.50 per share during the year. The Ætna Quicksilver Company's stock also is rarely quoted. Its price is about $4 per share.

While the year did not fulfill all the expectations of the friends of mining industries in Boston, there was much to encourage the belief that 1897 would be a year of unusual prosperity.

Cleveland Mining Stock Market in 1896.

The business depression that marked the year 1896 had an ill effect on the mining stock market of Cleveland. The sales that were recorded were, as a rule, quite light. Early in the season it became apparent to the dealers in mining stocks that the Presidential campaign, with the agitation that would follow as a consequence, would interfere with business for some months at least. Investors took that view of the situation, and the ultimate result of the closing of some of the mines and the general feeling of uncertainty was that very few of the stocks were quoted at a higher rate than they were at this time in 1895. One company's stock has declined considerably during the twelve months. Those which have held their own against the depression considered themselves very fortunate.

Republic stock advanced from $12 to $16 during the year, an increase of 33⅓%. Lake Superior Iron Company stock fell from $32 to $25; Chandler dropped from $38 to $30; Cleveland-Cliffs Iron Company stock rose by easy stages from $40 to $45; Pittsburg, Lake Angeline, and Jackson were steady at $75. Minnesota began the year with a quotation of $70 and closed with $66. No other stocks are dealt in to any extent in this market.

The Mining Stock Exchanges of Colorado in 1896.

The speculative excitement in Colorado in 1895 attendant upon the Cripple Creek discoveries, increasing output, larger development, heavy investments in mining ground, and the formation of many new companies, abated considerably during 1896. The Cripple Creek boom had passed its culminating point; yet on the whole trading was fairly active throughout the year, and if compared with the years 1892–94, may be said to have been satisfactory to commission brokers, notwithstanding the fact that there were so many more in the business than there had been before the boom.

Interest was pretty closely confined to the gold mining stocks, as, ever since the great decline in the price of silver in 1893, not only mining operations, but also investment and speculation have been mainly directed toward gold. Of the gold camps Cripple Creek naturally attracted the most attention, there being comparatively few transactions in Gilpin County shares, which latter, in fact, are for the most part not on the market.

The relative volume of business as compared with that of 1895 may be thus stated: From May 15, 1895, to the close of that year the Colorado Springs Mining Stock Association handled 18,721,122 shares of listed stocks, valued at $3,653,150, and 36,357,997 shares of unlisted stocks, a total of 54,979,119 shares. In the same seven months the Colorado Springs Board of Trade handled 75,991,070 shares, valued at $3,402,126, while there were considerable sales in the Consolidated Exchange in the same city. There were then three exchanges at Colorado Springs, but in the early part of 1896 the Board of Trade and the Consolidated united. For the year the whole business in listed stocks at Colorado Springs amounted to 42,867,424 shares, for which, taken altogether, much lower prices were obtained than ruled in 1895. The most active stocks at Colorado Springs were: Isabella, 5,113,198 shares sold at quotations ranging from 38½c. to 66¼c.; Pharmacist, 3,530,000 shares, 5¼c. to 20¼c.; Cripple Creek Consolidated, 2,797,512 shares, 8½c. to 22½c.; Creede & Cripple Creek, 2,610,150 shares, 2c. to 9c.; Work, 2,222,150 shares, 6c. to 26c.; Union, 1,969,500 shares, 23½c. to 60c.; Fanny R., 1,227,550 shares, 2⅝c. to 12⅝c.; Garfield-Grouse, 1,220,206 shares, 3⅛c. to 10¼c.; Alamo, 1,203,017 shares, 3c. to 10½c., and Jack Pot, 1,000,650 shares, 2½c. to 15c. Portland, which may be taken as a representative Cripple Creek property, opened at $2 in January, sold down to $1.07½ in August, and closed at $1.28 to $1.36, the total sales for the year being 638,474 shares. Of the Aspen stocks, Mollie Gibson sold at 54c. in January, and was fairly steady throughout the year, the lowest quotation being 40c., while Argentum-Juniata, which touched 70c. in March, went to 41c. in August, and in December stood at 44@50c.

The Colorado Mining Exchange at Denver is the oldest board of the kind in the State, having been opened in July, 1889, since which date it has had various vicissitudes. Business at this exchange was very dull in 1892–93, but revived when the Cripple Creek boom began and the Leadville gold belt came to the front. In 1896 the total sales were 142,837,824 shares. The largest week's business was in the week ending January 18, when 8,271,750 shares changed hands; the smallest, in the week ending May 9. The heaviest trading was

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS DURING 1896.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Ajax	$1.00	.15	.10	.15	.09	.....	.....	.10	.....	.10	.....	.....	.....
Alamo	1.00	.10½	.08	.08½	.06	.07½	.05⅝	.08	.03	.06½	.04	.06	.03
American Coal	1.00	.06⅜	.04⅛	.05	.03	.04	.02	.03½	.01¾	.02½	.01⅛	.02	.01¼
Anaconda	5.00	.72¾	.64	.67	.54¾	.70¼	.52½	.70	.50	.70	.55	.65	.57
Anchoria-Leland	1.00	3.50	2.50	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Aola	1.00	.05¾	.03¼	.04	.02⅞	.03½	.01½	.03¾	.01	.02½	.01	.02	.01
Argentum-Juniata	2.00	.62	.43	.69	.60	.70	.50	.64	.50	.58	.47½	.59	.49¾
Bankers	1.00	.22½	.17¾	.20⅛	.16¼	.20	.14	.16	.09	.14¾	.11	.14	.11½
Banner	1.00	.05¾	.04¼	.06	.03¾	.06	.03	.05¾	.04	.05½	.02¾	.05⅛	.02
Ben Hur	1.00	.11⅜	.07¼	.08½	.06	.07⅛	.05¼	.07	.03¾	.06¼	.04	.06½	.03⅛
Blue Bell	1.00	.14	.08½	.10⅝	.07½	.10	.05¾	.08½	.04⅛	.08	.04	.06	.02⅞
Bob Lee	1.00	.04	.02¼	.03	.01⅞	.03	.01	.02	.008	.01¾	.01	.01½	.00¾
Boston & Cripple Creek	1.00	.02¾	.01⅞	.02¼	.01½	.02	.01	.02½	.00⅞	.01¼	.005	.01	.005
Buckhorn	1.00	.09	.06¾	.10	.06	.07⅛	.04¾	.05	.02½	.03¾	.02½	.03⅞	.02¾
Colorado C. & M	1.00	.08⅞	.06⅞	.07¾	.05	.06½	.04¾	.05⅞	.03⅜	.04	.03	.04⅝	.03¼
Columbine	1.00	.06¼	.04⅛	.04¼	.02⅝	.03½	.02⅛	.03¾	.01½	.03¼	.01¼	.03	.01
Copper Mountain	1.00	.03½	.01⅞	.02½	.01½	.02¼	.01¼	.02	.008	.01½	.005	.01⅛	.005
Creede & Cripple Creek	1.00	.07¼	.06½	.07¾	.05⅝	.08	.04½	.05¼	.02	.04⅛	.03	.04⅞	.03
Cripple Creek Cons	1.00	.22½	.18¾	.22	.16¼	.19½	.18½	.17¼	.12½	.14¾	.15¼	.15¾	.13⅝
Cripple Creek Exploration	1.00	.17¼	.17	.18	.17	.18	.16	.17	.12½	.13	.12½	.12½	.12
Crœsus	1.00	.03½	.02⅞	.03¼	.02	.03	.02	.02½	.01½	.02	.01	.01½	.01
Dante	1.00	.13	.11¼	.12⅞	.09	.12	.08	.11¾	.06½	.10½	.07	.10	.06½
Des Moines	1.00	.08⅛	.07½	.07¾	.06½	.08	.04	.08	.02	.05	.02	.03½	.02
Enterprise	1.00	.27	.19½	.23	.17	.20	.11	.13	.09⅝	.10½	.07	.10	.07
Fanny R	1.00	.11	.07¾	.12	.07⅛	.12⅝	.09	.12	.08	.10½	.07½	.10	.07¼
Favorite	1.00	.16	.10	.11¾	.09	.09⅞	.06	.08	.04½	.08	.04	.08½	.05½
Franklin	1.00	.02¾	.02⅛	.02⅝	.01	.02	.01	.02¼	.00⅝	.01⅛	.008	.014	.005
Garfield-Grouse	1.00	.08	.05¼	.06⅞	.05⅜	.06⅛	.04⅛	.05¼	.03½	.04⅛	.03⅛	.04⅝	.03¼
Golden Age	1.00	.04⅝	.03	.03½	.01¾	.03	.01⅛	.02⅛	.01	.01⅞	.01	.03	.01
Golden Eagle	1.00	.05¾	.04½	.05	.03	.05	.02	.03½	.01½	.03½	.01	.02⅛	.01
Golden Fleece	1.00	1.75	1.50	1.68	1.50	1.68	1.40	1.75	1.60	1.71	1.64	1.70	1.40
Gold & Globe	1.00	.28	.20	.28½	.17½	.24½	.19	.24	.20	.24	.19½	.22	.18
Gold King	1.00	.66	.30	.61	.45	.60	.42	.60	.45	.53	.27½	.50	.25
Gold Standard	1.00	.12	.09	.10¾	.08	.10⅝	.08⅜	.10	.04½	.10	.07¼	.09	.05
Gould	1.00	.19⅛	.11⅜	.16	.12	.14½	.10	.14	.06	.08¼	.05	.09	.05
Grotte	1.00	.04	.02¼	.02⅝	.01¾	.02¼	.01⅞	.01¾	.01⅛	.....	.....	.01⅛	.009
Henrietta	1.00	.04	.03	.03⅜	.02¼	.02⅝	.01⅜	.02	.00⅛	.01⅛	.008	.01¼	.007
Humboldt	1.00	.05¾	.04	.05	.03	.04⅛	.02½	.03¼	.01	.05	.01	.03	.01½
Ida May	1.00	.17	.14½	.16	.14½	.20	.14	.16	.10	.15	.13	.12¼	.....
Ingham C	.50	.26	.25	.25½	.22	.26	.24	.26	.10	.25	.21	.25	.18
Iron Clad	1.00	.12	.08	.10	.06½	.08⅞	.05½	.06⅛	.04½	.05¼	.04	.05¼	.02
Isabella	1.00	.60½	.55	.57⅞	.49¾	.58¾	.49½	.58¼	.51⅞	.59¼	.52¼	.66¼	.59¼
Isabella Stamp	1.00	.66	.52	.54	.48¾	.58½	.48½	.57¾	.51½	.58¾	.52⅜	.66	.58½
Jack Pot	1.00	.15	.12	.12½	.08¾	.12	.08	.10	.05	.07	.05	.05¾	.04¼
Jefferson	1.00	.26	.19	.22⅞	.18	.26	.16	.22	.15	.18¾	.14	.20	.15
Keystone	1.00	.10¼	.07⅛	.08	.05¾	.08½	.05	.06	.03½	.05	.03	.04⅞	.02⅞
Ladessa	1.00	.....	.....	.02	.01¼	.01¾	.01¼	.....	.....	.....	.....	.005	.004½
Lincoln B	1.00	.08½	.07	.08	.06¾	.....	.....	.....	.....	.....	.....	.02½	.008
Lottie Gibson	1.00	.04	.02⅞	.03¼	.01¼	.02½	.01	.02½	.00⅞	.02	.00⅞	.01½	.006½
Marion	1.00	.02	.01	.01½	.01⅛	.01¾	.01¼	.01¼	.008	.01¼	.006	.01	.002½
Matoa	5.00	.30½	.20	.27	.20	.25	.10	.20	.10	.14	.08¼	.13	.09
Mollie Gibson	1.00	.54	.40	.51	.45	.84	.47	.75	.50	.80	.50	.85	.61
Monarch	1.00	.13½	.06	.07½	.05	.07	.01¼	.06	.03	.06	.02½	.06	.02
Mt. Rosa	1.00	.21	.14½	.18	.15	.17¼	.10½	.15	.08⅛	.11	.08	.15	.09
Mutual	1.00	.08½	.07½	.08⅜	.07	.08	.04¼	.06	.02¾	.04	.03	.04	.01
Nugget	1.00	.15	.10	.15	.10	.15	.06	.15	.03	.08	.04	.12	.06
Ophir	1.00	.12½	.10	.12	.09½	.12	.08	.10	.05	.10	.05	.10	.03
Oriole	1.00	.05¾	.04⅞	.05½	.04	.04⅝	.03	.03⅛	.02½	.03½	.02½	.03⅜	.01⅞
Orphan Boy	1.00	.14	.10	.13½	.09	.15	.08	.12	.07	.14	.07	.11¾	.06
Pappoose	1.00	.09½	.06½	.08¾	.06½	.07	.05	.06	.03¼	.04	.02⅜	.04¾	.03
Pharmacist	1.00	.18⅞	.13	.17	.14¼	.15	.11⅛	.11⅛	.07⅞	.08½	.05¾	.10⅞	.06¾
Portland	1.00	2.00	1.65	1.85	1.41	1.79	1.35	1.52	1.33	1.86	1.43	1.96	1.72
Princess	1.00	.11	.09½	.11	.08	.14	.11	.12	.10	.....	.....	.07½	.02
Reno	1.00	.09⅝	.06	.08	.04½	.06	.03¼	.05¾	.02	.06	.03¼	.05	.01
Sacramento	1.00	.10	.08	.10½	.06½	.09	.07	.09	.05	.06½	.04½	.05	.01⅛
Silver Street	1.00	.02½	.02	.02¾	.01½	.02	.01	.02	.00⅞	.01½	.005	.01½	.005
Specimen	1.00	.17	.15	.15	.13	.13¼	.10	.10½	.08	.10	.07	.10	.07
Squaw Mt. Tunnel	1.00	.12	.09¾	.11	.07⅞	.08	.05	.06	.03⅝	.04	.01⅞	.03⅜	.01¼
Temonj	1.00	.16	.10	.14	.08	.13	.06	.09	.04	.08	.04	.06¼	.04¾
Trachyte	1.00	.04¾	.02	.05¼	.04	.05½	.03⅛	.05	.02	.04½	.02	.04½	.02½
Union	1.00	.45¼	.38	.60	.40⅛	.54⅜	.42	.47½	.35	.41	.33½	.38	.32
Virginia M	1.00	.03¾	.02½	.03⅛	.02	.02⅝	.01⅛	.03	.008	.02	.008	.02	.009
Work	1.00	.26	.19¾	.20¼	.16½	.17¼	.13⅝	.13¾	.09¾	.12⅜	.10¼	.12⅜	.10⅛

during January, February, and the beginning of March, and again it was fair in October to December. The highest business was during the summer months. The course of prices was the same as at Colorado Springs, and at both

FLUCTUATIONS OF MINING STOCKS AT COLORADO SPRINGS DURING 1896—*Continued.*

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Ajax	.....	.....	.10	.....	.....	.....	.....	.....	.....	.....	.....	.....	55,450
Alamo	.06	.03½	.04	.03	.05½	.03½	.05½	.04½	.07⅞	.05⅜	.08⅛	.05¼	1,203,017
American Coal	.02¼	.01⅜	.01¾	.00⅞	.02⅝	.01	.02⅞	.01⅞	.03⅞	.02¾	.05⅜	.04	620,876
Anaconda	.62	.50	.56	.31	.65	.40	.86¾	.49½	1.40	.84	1.20	1.01	687,230
Anchoria-Leland	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,015
Aola	.01⅝	.01⅛	.01⅛	.01	.....	.....	.....	.....	.....	.....	.....	.....	122,375
Argentum-Juniata	.57½	.49¾	.52	.41	.60	.42½	.58½	.49	.55	.45½	.50	.44	765,745
Bankers	.13	.09	.10½	.07	.10	.08¼	.....	.....	.14	.13⅝	.....	.....	290,700
Banner	.05	.01	.04	.01	.04	.01	.04⅝	.01⅛	.04	.02¾	.05½	.01¼	180,100
Ben Hur	.06	.02	.05	.04	.05	.04½	.05¾	.05½	.07	.06	.06⅜	.06⅛	258,000
Blue Bell	.06	.02¾	.05	.02½	.08¾	.04	.10	.05¾	.....	.....	.....	.....	96,950
Bob Lee	.01¼	.005	.00⅛	.00½	.01½	.007	.01⅜	.01	.02⅛	.01⅜	.01½	.01¼	631,530
Boston & Cripple Creek	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	93,500
Buckhorn	.03¼	.02	.02¼	.01⅛	.02⅜	.01⅞	.03	.02¼	.04⅛	.03⅜	.04¼	.03⅝	742,850
Colorado C. & M	.04⅛	.03⅛	.03¾	.02⅞	.04⅛	.03⅜	.04¼	.03⅝	.04¾	.04¼	.....	.....	722,500
Columbine	.02	.01⅜	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	78,000
Copper Mountain	.01¼	.00⅝	.01¼	.009	.....	.....	.....	.....	.....	.....	.....	.....	279,300
Creede & Cripple Creek	.05¼	.03½	.09⅛	.04⅛	.09⅛	.07⅞	.08⅜	.06	.09	.06¾	.07¼	.06	2,610,150
Cripple Creek Cons	.15	.10½	.12⅝	.08⅜	.14⅞	.09⅞	.14⅞	.12	.20⅛	.14¾	.16½	.14⅜	2,797,512
Cripple Creek Explo	.12¼	.....	.12¼	.12	.13	.12¼	.12½	.03½	.10¾	.10	.10	.....	299,000
Crœsus	.01⅞	.007	.03	.006	.01⅝	.00⅞	.01½	.009	.01⅜	.01¼	.01¼	.01	622,550
Dante	.10½	.05	.08¾	.05	.....	.....	.08½	.....	.....	.....	.08¾	.....	123,450
Des Moines	.03⅛	.02	.04½	.01	.02¾	.01⅞	.03¼	.02¾	.04⅛	.01⅛	.04	.03	354,500
Enterprise	.09½	.05	.08	.04	.08	.03¾	.07⅞	.06¼	.08½	.07¼	.07⅝	.07⅜	155,825
Fanny R	.08¾	.05⅛	.08	.05	.08	.05¼	.08	.05¾	.07½	.06¼	.06⅞	.02⅜	1,227,550
Favorite	.08	.04	.06⅜	.05	.07½	.04	.08⅞	.07¼	.15½	.08⅞	.09½	.08	455,056
Franklin	.01⅛	.00⅜	.00⅞	.00⅘	.01	.006	.01⅛	.00⅘	.01½	.01	.01⅛	.....	625,550
Garfield-Grouse	.03⅞	.03⅛	.04¼	.03¼	.05⅛	.03½	.10¼	.05⅜	.10½	.09⅛	.09⅞	.08⅜	1,226,206
Golden Age	.02	.00⅞	.01¼	.008	.008	.....	.03¼	.02⅛	.03⅜	.02⅜	.....	.....	230,650
Golden Eagle	.02½	.009	.01⅜	.01	.....	.....	.03	.01⅜	.04	.03	.....	.....	114,150
Golden Fleece	1.50	.92½	1.05	.82	1.20	1.00	1.02	.99	1.09	.97	1.00	.96	168,990
Gold & Globe	.20	.16	.18	.14½	.17	.10	.14¾	.14	.17½	.14⅞	.15⅜	.13½	153,200
Gold King	.55	.40	.34	.....	.45	.35	.62	.55	.....	.....	.....	.....	15,300
Gold Standard	.08	.03¼	.04⅞	.03	.04½	.04	.05	.04¾	.05½	.05	.....	.....	233,100
Gould	.08	.05	.07	.04	.12	.06	.14⅛	.11¼	.....	.....	.....	.....	147,475
Grotte	.012	.007	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	208,000
Henrietta	.009	.00⅞	.007	.00⅝	.....	.....	.....	.....	.....	.....	.....	.....	340,010
Humboldt	.02¼	.01⅝	.05¾	.04½	.02	.01½	.....	.....	.03⅜	.03	.....	.....	245,200
Ida May	.....	.....	.15	.14	.12	.10	.....	.....	.....	.....	.....	.....	82,300
Ingham C	.23	.15	.20	.04	.12	.04	.10¼	.09½	.10⅞	.09¾	.11	.10	209,834
Iron Clad	.03⅝	.02⅛	.07	.03¾	.07⅛	.06⅛	.....	.....	.09¼	.08⅞	.....	.....	251,466
Isabella	.64¾	.53¼	.54	.40	.44½	.40¾	.46¾	.38½	.48	.41	.44¼	.42	5,118,198
Isabella Stamp	.64½	.51½	.53½	.38⅛	.47	.39¾	.46¼	.38	.46¼	.40¼	.43⅜	.41	873,050
Jack Pot	.05¾	.02½	.05	.02¾	.06	.04	.07⅛	.05⅝	.08	.05⅜	.08¾	.05	1,009,650
Jefferson	.20	.15	.....	.....	.....	.....	.12⅛	.12¼	.....	.....	.14	.12	287,500
Keystone	.03¾	.02	.02¾	.02¼	.05⅛	.02½	.05¼	.04½	.06	.05½	.05½	.04⅞	150,500
Ladessa	.006	.....	.00⅝	.003	.....	.....	.....	.....	.....	.....	.....	.....	28,000
Lincoln B	.008	.004	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	38,000
Lottie Gibson	.01	.005	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	163,300
Marion	.01	.002	.00⅝	.004	.00⅜	.004	.01⅛	.00⅛	.01⅛	.00⅛	.00⅛	.00⅞	852,300
Matoa	.11½	.04½	.09	.02	.12⅛	.06	.14¼	.11	.15½	.12	.11⅜	.10	201,100
Mollie Gibson	.80	.46	.51½	.40	.50	.41	.50	.41½	.68	.45	.63	.55	376,175
Monarch	.05	.04¼	.02½	.02	.02½	.009	.04⅜	.03⅜	.05½	.03¾	.05⅝	.05⅜	82,200
Mt. Rosa	.13¾	.11¾	.12	.09⅞	.18	.11¼	.22	.14⅛	.16¼	.14⅞	.14⅞	.12	869,730
Mutual	.03¼	.02	.03½	.02¾	.05	.02⅞	.04⅛	.03½	.06⅝	.....	.....	.....	228,750
Nugget	.13	.07	.....	.....	.06½	.04½	.10	.06¼	.....	.....	.....	.....	38,494
Ophir	.08	.04	.06¾	.05	.09⅞	.05¾	.08¼	.07¼	.09½	.08⅞	.09	.08⅜	164,850
Oriole	.03	.01	.02½	.01¾	.05	.02	.05	.02½	.03⅜	.02⅜	.02⅜	.02	676,000
Orphan Boy	.12	.08	.09	.07½	.12	.07¼	.11	.09¼	.12	.11⅝	.....	.....	138,200
Pappoose	.04½	.03¼	.03½	.02	.02¾	.02⅜	.04⅛	.02¾	.03⅞	.03¾	.03½	.02⅞	951,500
Pharmacist	.10⅝	.08	.08⅞	.07½	.11	.08⅛	.16⅛	.11½	.20¼	.15⅝	.17	.15	3,530,090
Portland	1.78	1.28½	1.32½	1.07½	1.38	1.18	1.55½	1.35	1.45	1.27	1.35	1.28	638,474
Princess	.07	.04	.07½	.02⅛	.04½	.02	.....	.....	.....	.....	.....	.....	11,000
Reno	.02¾	.01⅛	.02¼	.01⅝	.....	.....	.03¼	.....	.....	.....	.....	.....	50,000
Sacramento	.04⅛	.03¼	.03½	.02⅞	.03½	.03	.06⅛	.04⅞	.....	.....	.05¾	.05⅜	251,950
Silver Street	.01¼	.005	.....	.....	.01½	.01⅜	.....	.....	.....	.....	.....	.....	89,250
Specimen	.10½	.06	.10⅛	.06	.08½	.07½	.08¾	.07⅜	.09	.07⅞	.08¾	.08	410,150
Squaw Mt. Tunnel	.02¾	.01⅞	.02⅞	.01	.02⅝	.01¾	.02¾	.02¼	.02⅞	.02⅝	.....	.....	751,500
Temonj	.06	.03	.06	.03¾	.07	.04½	.09¼	.08	.....	.....	.....	.....	58,500
Trachyte	.03½	.02	.03	.01¼	.03¾	.02	.03⅛	.02¾	.04¾	.03	.03¾	.03⅛	795,000
Union	.36⅛	.31	.33¼	.23½	.33⅜	.25	.34	.27¾	.36¼	.29½	.32¾	.25	1,969,501
Virginia M	.01½	.009	.01⅛	.01¼	.02⅝	.02⅛	.....	.....	.....	.....	.....	.....	119,700
Work	.12⅜	.08¼	.08⅞	.06	.09⅛	.06	.09½	.07	.11½	.07½	.09	.08⅛	2,222,150

NOTE.—Total sales for the year, 42,867,424 shares.

places the disastrous fires at Cripple Creek, and the prolonged strike at Leadville, had a depressing influence.

Besides these there are the Gold Mining Stock Exchange of Cripple Creek, the exchange at Pueblo, and there have been others of less note in the State.

Colorado mining stocks are largely dealt in outside of the State, in New York City, London, Paris, and even at Shanghai, where Sheridan-Mendota is regularly quoted.

The Helena Mining Stock Market in 1896.

By S. K. Davis.

The accompanying tabulated statement shows the stocks sold during 1896 in this market. Butte is the only other center for mining stock transactions in Montana, although there is some business done in Missoula; the balance of the State, as a rule, making its transactions in Helena and Butte.

The characteristics of the transactions were hardly in any full sense speculative—the first two years after 1890 settled that. Not only in mining stocks, but other avenues of venture, the dearly-bought lesson has had exceedingly good results, for the reason that the resources and management of mining companies whose stocks are offered are closely looked into, consequently the purchases of stock are, as a rule, made as an investment for dividends rather than as a speculation for a rise, and although the amount realized may not always be as large as the investors desire, still as a fair interest on capital the average will compare more than favorably with the average investments of the country. The Montana and Idaho mines whose stocks are offered to the public, so far as most agencies are concerned, are well, conservatively, and honestly managed, and the chances of loss (there is no absolute guarantee against loss in any investment, mining or other) are reduced to a minimum, through conservative management, while the average of gain is oftentimes very large.

For instance, the Iron Mountain Company, whose stock is more largely sold at home than abroad, paid in 1896, $82,500, and since organization, to date, $492,500; it is true that $27,500 of this was in receivers' transfers to stockholders on the books of the First National Bank of Helena, and it is the opinion of those most familiar with its affairs that it will eventually pay 100 cents on the dollar. The average price of the stock for 1896 was approximately 45c. per share, which would yield 40% per annum on the investment. As the policy of the management is never to accumulate debt, but rather a substantial balance in the treasury, it seems very safe, particularly as the mine makes as good a showing as ever.

The American Development Company is a well-managed institution with large prospects of success. Bald Butte had large expenses for exploitation of the property, and the earnings were devoted, since the last declared dividend, June 23, 1896, to the expenses incurred in that direction. Dividends to that date 13c. per share equal $32,500 for 1896. The stock is held in few hands, and both the mine and management have the confidence of those most interested. The Helena & Frisco was sold to the Exploration Company of London on the report and advice of Hamilton Smith, and the stock will be listed in London on a new capitalization of 200,000 shares, par value £1. The Castle properties, with the advent of the railroad, have great promise, at present

regarded as speculative, but doubtless some of them will, like Yellowstone and Judge, be dividend payers and investment stocks during 1897.

FLUCTUATIONS OF IDAHO AND MONTANA MINING STOCKS DURING 1893.

Name of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
American Development Co.	$1	2.00	.....	1.90	.....	.....	.....	2.00	.....	1.90	.....	.....	.....
Bald Butte	1	3.15	.....	3.30	.....	.....	.....	.....	.....	2.75	.....	2.75	2.50
Bimetallic	5	.....	.....	.....	.....	.....	.....	.....	.....	3.00	.....	.....	.....
Combination	10	.70	.....	.70	.65	.70	.....	.60	.....	.....	.....	.52½	.....
Helena & Frisco	5	.....	.....	.....	.....	.....	.....	2.50	2.00	2.00	.....	.....	.....
Helena & Victor	5	.....	.....	.30	.25	.50	.45	.50	.45	.40	.35	.....	.....
High Ore	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Iron Mountain	10	.50	.....	.50	.45	.52½	.47½	.50	.47½	.50	.47½	.47½	.....
Judge	.....	.....	.....	.....	.....	.....	.....	.07½	.....	.12½	.....	.09	.08
Merrill (Gold)	1	.....	.....	.60	.....	.....	.....	.55	.....	.....	.....	.....	.....
Ontario	1	.....	.....	1.00	.90	.....	.....	.....	.....	.85	.75	.....	.....
Yellowstone	5	.....	.....	.10½	.....	.12½	.....	.15	.12½	.18½	.....	.20	.....

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
American Development Co.	1.75	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	6,600
Bald Butte	2.25	.....	2.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	3,046
Bimetallic	.....	.....	2.75	.....	.....	.....	.....	.....	.....	.....	.....	.....	700
Combination	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,300
Helena & Frisco	3.50	.....	2.25	2.00	2.00	.....	.....	.....	.....	.....	*5.00	.....	539,500
Helena & Victor	.30	.25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	20,900
High Ore	.....	.....	.....	.....	.....	.....	1.50	1.00	1.00	.75	.....	.....	28,100
Iron Mountain	.42	.41	.42	.41	.42½	.....	.42	.....	.....	.....	.....	.....	204,063
Judge	.10	.09	.10	.....	.39	.....	.09	.....	.....	.....	.....	.....	255,850
Merrill (Gold)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,500
Ontario	.60	.50	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	13,800
Yellowstone	.20	.16½	.20	.17½	.....	.....	.17	.....	.....	.....	.....	.....	248,500

* 500,000 shares sold in December (half cash, half stock) at £1 per share in London.

THE NEW YORK MINING STOCK MARKET IN 1896.

DURING the early months of 1896 the public apparently devoted more attention to investment in this class of securities, and as a result the highest prices of the year were realized at that time.

The failure of the attempts to organize a new mining exchange discouraged outside speculation for the remainder of the year. In addition to this the fires at Cripple Creek and the miner's strike at Leadville affected trading, especially in the Colorado stocks. Wheat speculation during the closing months of the year was also detrimental to the mining stock market, inasmuch as brokers took to this field, considering it more remunerative than their own for the time being.

Practically all dealings of any account which were recorded in New York in 1896 were made at the Consolidated Stock and Petroleum Exchange, and in only a few cases did the New York Stock Exchange show such transactions.

The first venture in the way of extending dealings in mining stocks this year was the establishment of the New York Mining Exchange, which opened its doors to the public in February, only to close them again after a brief career. This exchange was incorporated in December, 1895, with a capital stock of $10,000. The incorporators were Henry A. Mott, Edwin A. Beers, James H.

Kerr, and Edward H. Williams, all of New York. The directors for the first year were these gentlemen and Isham B. Porter, Howard Scrymser, Stephen B. French, W. Leslie Scrymser, and W. C. Nicol. Mr. Ogden B. Pell was secretary *pro tem*, and J. H. Kerr treasurer *pro tem*.

One of the foremost promoters of this exchange was Isham B. Porter, of Colorado, who was its president. In his opening address Mr. Porter made a statement of the object and purposes of the exchange, and was followed by Judge J. W. Rucker, of Denver, Colo.; J. W. Deane, of Aspen, Colo.; E. T. Colborn, Salt Lake City, Utah; H. A. Mott, consulting engineer of the exchange, and others. Representatives of various Western mining exchanges, and others interested in the industry, were present on this occasion. In the prospectus of the exchange it was stated that no undesirable property would be listed. The classification of securities was as follows: *A*. Those properties which pay regular dividends out of actual net earnings. *B*. Prospects which pay occasional dividends out of actual net earnings, and properties which are developed to a point which affords a reasonable prospect for the early payment of dividends. *C*. Prospects upon which actual development work has been done. No securities could be listed until after the properties represented by them were reported upon by expert and disinterested engineers. In Class *A* the following stocks were listed: Ajax, Bullion-Beck & Champion, Centennial-Eureka, Daly, Gemini, Horn Silver, Mammoth, Mercur, Ontario, Silver King, Eureka Hill and Utah, all in the State of Utah; Anchoria-Leland, Isabella, Mollie Gibson, Pharmacist, of Colorado. In Class *B*, the Colorado stocks were: Argentum-Juniata, Bradley, Pioneer, Blue Jay, Colorado Giant, Cripple Creek Consolidated, Enterprise, Gold & Globe, Insley, Lincoln Boy, Matoa, Trail Run, and Work; the Utah stocks were: Daly-West, Golden Gate, and Sunshine. Class *C* contained Colorado stocks as follows: Anna May, Apothecary, Atlantic & Pacific, Bill Jim, Cripple Creek Exploration, Constitution, Dictator, Eagle, Elsie, Gold Hill, Gladys A., Investors, Illinois, Justice, Little Carlisle, Mineral Farm, Mic Mac, Mount Rosa, Old Gold, Ophir, Pike's Peak, and Summit; Utah stocks were: Alliance, Anchor, Gesyer, and Marion. The only Idaho stock was Manhattan.

The exchange expected to have all the companies on its list appoint transfer agents in this city so as to facilitate business, in order that buyers in New York would have no difficulty in getting the stock transferred in their own names.

Unfortunately the exchange did not start right. The promoters were too anxious to receive newspaper notoriety, and they got the most injudicious kind of advertising. This had its effect afterward, and we had not long to wait before a rupture took place among the officers and directors. The latter expressed their willingness to resign whenever the members should call for it, and plans were then considered for reorganizing the exchange. The result was that in March a new board of directors was chosen, consisting of B. A. Ammon, president; C. Vey Holman, vice-president; William Brandreth, treasurer, and O. P. Pell, secretary. On May 4, however, the Supreme Court appointed Horatio W. Stocker receiver for the exchange, application having been made by Directors C. V. Holman, William Brandreth, Horatio W. Stocker, Ella Wooster, and W. L. Boyd. They recited the cause to be heavy

expenses, actions pending against the corporation, and other actions threatened. The liabilities were only $5697, of which $3541 were contested claims. The nominal assets were $5358, of which $3775 was doubtful.

A few weeks after the collapse of this exchange other parties formed the American Stock Exchange, to deal in industrial and mining shares. This concern opened its doors on June 15. The officers were: President, Stephen H. Emmens; vice-president, R. K. Warren; treasurer, J. B. Kellogg; secretary, Alexander Seaman. In his opening address President Emmens laid stress upon the fact that the new exchange was in no respect a reorganization of the defunct institution. The membership of this exchange was placed at $100 to $500, with "no other charges of any kind for the first year." The exchange was incorporated under the laws of the State of New York, with a capital stock of $10,000, divided into 100 shares of $100 each. All securities were allowed to be dealt in on the exchange without being either listed or called, providing that there was filed in the recording department in respect of each such security a statement of particulars for the information of the public, and provided, also, that the accuracy of such statement was certified by two members of the exchange. Transactions were either "for cash" or "for the account." Those for cash were cleared on the day following the record, while those for the account were cleared on the next "settling day," either on the second or fourth Wednesday of each month. This feature was thought would greatly simplify and facilitate the conduct of business, especially with country clients and distant exchanges, and by enlisting the honor and good faith of the members would afford a much better protection to the public than could possibly be provided by any system of official investigation and listing. Actual business on the exchange, however, did not prove satisfactory to its originators; in fact, no business of any account was done. The public simply ignored it, as might have been expected.

Finally a new mining exchange was formed in August. The directors are Stephen V. White and Emery N. Downs, Brooklyn; Irwin C. Stump, Benjamin T. Marten, James R. Branch, William C. Dornin, Moses E. Wooster, Louis Ross, and Robert A. Irving, of New York City; George L. Lancon, of Rutherford, N. J.; Malcolm W. Nevin, of Montclair, N. J., and Dr. William Brandreth, of Sing Sing (formerly treasurer of the first mining exchange). The officers are: President, W. C. Dornin; vice-president, Louis Ross; treasurer, William Brandreth; secretary, Moses E. Wooster; chairman, John Gray. This concern was modeled after the New York Stock Exchange, but did not begin business before the close of the year.

Business on the Consolidated Exchange.—Of the Colorado stocks the Cripple Creek group was the most active. The highest prices were realized in January, while the latter months of the year showed decreases; although the month of September recorded the largest output of ore in the history of the camp—amounting in value to $817.000.

The stock of the Victor Gold Mining Company is held principally by a French syndicate which has listed it on the Paris Bourse. The depression in that market in certain months of the year caused some holders to send their stock to this side of the Atlantic in order to realize better prices. As a result

more shares were sold than in 1895. The lowest price in 1896 was in January, $5.25, but it rose again in April and May to $8.50, with a later reaction in December to $6.50. The Victor is a regular dividend payer, and has paid to date over $700,000. The other stocks in the Cripple Creek group show fluctuations in prices, with many sales.

Speculation in the Leadville group was hampered to a great extent by the prolonged miners' strike already referred to. Leadville Consolidated ruled at 10@18c. Little Chief opened at 19c., made an advance of 4c., and closed at 15c. The most active stock in this group was Iron Silver, which advanced to 95c. in July—the highest price since 1891, when the stock sold for $1.45. The major portion of this company's stock is held by Detroit capitalists. Life was put into these apparently dead shares by the favorable termination of seven years' of litigation between the Iron Silver Mining Company and other claimants over the Sierra Nevada property. This ground was claimed by the Iron Silver Company under the company's titles, and in July the Supreme Court ordered that the property in dispute be restored to that company. Up to April, 1889, the company had worked so profitably as to distribute $2,500,000 in dividends, but it paid nothing to its stockholders during the litigation.

Small Hopes was rather active; the highest price attained was in June when the stock sold for $1.75. In August, however, it declined to 65c.

Application for listing was made in September by the Red Mountain Mining Company to the Consolidated Stock and Petroleum Exchange. The company's prospect is in the Red Mountain mining district of Ouray and San Juan Counties, Colorado. The company is capitalized at $200,000, and the par value of shares is $1. The committee on mining securities at the exchange after due consideration decided not to list any more prospects with little or no development.

The Comstocks are falling off in favor with the New York public, and are now regarded by many as old stocks. Comstock Tunnel stock ruled somewhat lower in price; it receded from 10c. in May to 5c. in December. Bonds to the amount of $8700 were sold at 7@9%. Chollar made a notable advance. In May it sold for $3.30—the highest price since 1891, when it was $3.85. The lowest point reached in 1896 was in March, 15c. The reported rich strikes in this mine account for the increased value of the stock. Gould & Curry made a fair showing, realizing $2.60 in July—the highest since 1891, when it sold for $3.80. The lowest figure in 1896 was 30c. during March. Hale & Norcross showed transactions in May at $3.30—an increase as compared with the last six years. In 1892 it sold for $3.40. In December, 1896, the stock sold as low as 83c. The advanced price early in the year was due to rich strikes reported in neighboring mines in the Comstock lode. Consolidated California & Virginia sold up to $3.60 in May. Two months before, however, it was selling at $1.55. At the close it sold for $1.10@$1.65. Savage advanced from 27c. in April to $2.05 in June, closing at 30c.

With few exceptions the California stocks were rather active, while prices ruled somewhat lower than in 1895. Brunswick Consolidated was the stock most dealt in; it sold as low as 6c. in March, and as high as 26c. in October, with heavy transactions. The deal to consolidate the mines of the Standard

Consolidated, the Bulwer Consolidated, the Bodie Consolidated and the Mono companies was agreed upon during the latter months of the year, and the new organization is now called the Standard Consolidated Mining Company of Bodie. Its stock capital is 200,000 shares, of which the Standard Consolidated stockholders get 95,000 shares, the Bulwer Consolidated, one share of the new stock for every three held (33,000 shares in all), Bodie Consolidated one shares for every two held, and Mono Consolidated one share for every six of its own. After the distribution of these shares the remaining stock in the treasury of the company will be used in the purchase of additional ground. This consolidation had been expected ever since the stocks of all the Bodie companies passed into the hands of one party some time ago. In Standard Consolidated stock the highest price realized was in March, $2.25, but in September when some dissatisfied stockholders placed their holdings on the market it fell to $1.00. In November, however, the stock rose again to $1.25 and closed at $1.20. On the other hand, Bulwer Consolidated and Mono Consolidated each showed advances from the opening of the year to the time of consolidation.

The Utah stocks were in more demand than in 1895, and prices were fairly steady. As these are silver-mining stocks principally, the speculation depends to a great extent on the market for silver and lead. From February until July, when the metal market showed steadiness and advanced prices, the market for the stocks was also active. This good feeling, however, did not continue long after the close of July. Centennial-Eureka, a high-priced stock, was offered sparingly, and the only sales recorded were made in February at $60.13@$60.60. The company pays regular dividends, and has paid to date nearly $2,000,000, which is another reason why stockholders are not selling. Ontario and Daly, two Park City properties, were dealt in to a limited extent only. Daly sold in February for $7.13—the only sale this year. Ontario advanced from $7.50 in January to $15 in April, an increase of $5 over 1895. Horn Silver ruled lower in price this year, owing to the passing of a few dividends. The stock sold at $2.30 in the first half of this year, but dropped to $1.50 at the close.

The South Dakota stocks show only moderate dealings, with lower prices in several cases.

There was more attention paid to Montana stocks this year. Stock of the Bedford Consolidated Mining and Smelting Company, placed on the list in June, sold at the opening for $2.75, but near the close of the year were sold at $7@$7.38 in consequence of a corner on the part of some stockholders who knew the shares were tightly held.

A call for certain stocks not on the regular list of the Consolidated Exchange early in the year induced its committee on mining securities to place the following stocks on the temporary list: Alamo Mining Company, Anaconda Gold Mining Company, Ajax Mining Company, Anchor Mining Company, Alliance Mining Company, Bullion-Beck & Champion Mining Company, Cripple Creek Consolidated Mining Company, Creede and Cripple Creek Mining and Milling Company, Crœsus Gold Mining and Tunnel Company, Centennial-Eureka Mining Company, Daly West Mining Company, Favorite Gold Mining Company, Gold &

FLUCTUATIONS OF MINING STOCKS AT NEW YORK DURING 1896.

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Adams (s. l. c.), Colo	$10	.....	....	.30	.....	.25	....	.....	.....	....	....	.....	....
Ajax, Utah	10	.....	....	.70	.60	....	....	1.10	.....	....	....	.....	....
Alamo (g.), Colo	1	.10	.09	.10	.08	.06	....	.....	.....	....	....	.....	....
American Flag (s.), Colo	10	.....	....	.04	.....	.04	.03	.....	.....	....	....	.03	....
Anaconda (g.), Colo	5	.71	.65	.63	.....	.64	....	.....	.....	....	....	.62	...
Anchor (g. s. l.), Utah	20	.....	....	2.25	.....	2.40	....	.....	.....	....	....	.....	....
Andes (s. g.), Nev	100	.....	....	.....	.....	....	....	.50	.....	....	....	.....	....
Argentum-Juniata (g. s. l.), Colo	2	.61	.56	.....	.....	....	....	.....	.....	....	....	.52	.40
Bedford Cons. (g.), Mont	....	.....	....	.....	.....	....	....	.....	.....	....	....	.....	....
Belcher (s. g.), Nev	100	.....	....	.....	.....	.30	....	.45	.40	.65	.58	.80	.58
Best & Belcher (g. s.), Nev	100	1.00	....	.82	.75	.70	.25	.65	.48	2.00	.53	2.20	1.00
Bodie Cons. (g.), Cal	100	.40	....	.....	.....	.33	.17	.45	.25	....	....	.68	....
Bullion-Beck & Champ. (g. s. l. c.), Utah	10	.....	....	7.25	6.00	....	....	.....	.....	....	....	.....	....
Bulwer (g.), Cal	100	.....	....	.25	.12	.17	.16	.36	.23	.40	.33	.69	.30
Breece (s. l.), Colo	25	.25	....	.....	.....	....	....	.15	.....	....	....	.15	....
Brunswick Cons. (g.), Cal	1	.10	.07	.09	.....	.10	.06	.15	.10	.15	.12	.21	.14
Caledonia (g.), So. Dak	100	.....	....	1.00	.....	....	....	.....	.....	....	....	.....	....
Centennial-Eureka (g. s. l. c.), Utah	50	.....	....	60.60	60.13	....	....	.....	.....	....	....	.....	....
Chollar (s. g.), Nev	100	.....	....	.65	.....	.50	.15	.40	.....	3.30	.70	4.25	2.80
Chrysolite (s. l.), Colo	100	.23	.18	.20	.....	.15	....	.15	.13	.15	.13	.....	....
Comstock Tunnel (s. g.), Nev	100	.08	.07	.08	.07	.08	.06	.06	.....	.10	.06	.09	.08
Comstock Tunnel Bonds	100	.....	....	.....	.....	....	....	.....	.....	.08	....	.09	.08
Cons. California & Virginia (s. g.), Nev	100	2.80	2.45	2.15	1.95	1.88	1.55	2.05	1.60	3.60	1.75	3.50	2.00
Cons. Imperial (g. s.), Nev	100	.04	....	.04	.....	.03	....	.06	.....	.08	.03	.04	....
Creede & Cripple Creek (g.), Colo	1	.20	.07	.08	.07	.08	.05	.06	.05	.05	....	.15	.05
Crescent (s.), Utah	25	.....	....	.05	.....	....	....	.....	.....	....	....	.....	....
Cripple Creek Cons. (g.), Colo	1	.22	.20	.20	.....	....	....	.....	.....	.14	....	.14	....
Crœsus (g.), Colo	1	.05	.04	.05	.04	.04	.03	.03	.....	.03	....	.03	....
Crown Point (g. s.), Nev	100	.30	....	.....	.....	.65	.34	.42	.40	.79	.40	.70	.50
Dalton (s.), Utah	25	.....	....	.12	.07	....	....	.....	.....	....	....	.....	....
Daly (s. l.), Utah	20	.....	....	7.13	.....	....	....	.....	.....	....	....	.....	....
Deadwood-Terra (g.), So. Dak	25	.....	....	.....	1.00	....	....	.....	.....	....	....	1.75	....
Dunkin (s. l.), Colo	25	.....	....	.....	.....	.12	.08	.....	.....	....	....	.....	....
Eureka Cons. (s. l. g.), Nev	20	.....	....	.....	.....	....	....	.....	.....	....	....	.....	....
Father de Smet (g.), So. Dak	100	.....	....	.....	.....	.17	...	.17	.13	.17	....	.....	....
Favorite (g.), Colo	1	.....	....	.11	.....	....	....	.....	.....	....	....	.....	....
Golden Fleece (g. s.), Colo	1	1.75	1.60	1.75	1.60	1.70	1.55	1.80	1.65	1.75	....	1.70	1.50
Gold & Globe (g.), Colo	1	.26	.23	.24	.....	....	....	.....	.....	....	....	.....	....
Gould & Curry (s. g.), Nev	100	.....	....	.....	.....	.35	.30	.50	.....	2.00	.78	2.65	1.30
Hale & Norcross (g. s.), Nev	100	1.45	....	1.25	.....	1.50	1.15	1.45	1.10	3.30	1.35	3.00	1.65
Homestake (g.), So. Dak	100	29.00	....	.....	.....	....	....	.....	.....	....	....	.....	....
Horn Silver (s. l.), Utah	25	2.30	2.25	2.10	2.00	2.10	2.00	.....	.....	2.30	2.20	2.25	2.20
Iron Silver (s. l.), Colo	20	.20	....	.....	.....	.25	.23	.24	.20	.24	.18	.19	.18
Isabella (g.), Colo	1	.63	.60	.58	.49	.60	.50	.56	.52	.58	.54	.65	.59
Julia (g. s.), Nev	100	.....	....	.....	.....	....	....	.....	.....	.56	....	.....	....
Kingston & Pembrooke (i.), Ont	10	.23	.20	.25	.....	25	....	.....	.....	.25	....	.....	....
La Crosse (g.), Colo	10	.10	.09	.09	.....	.10	....	.11	.09	.10	....	.....	....
Leadville (g.), Colo	10	.12	.11	.18	.12	.15	.14	.15	.13	.14	.12	.14	.13
Little Chief (s. l.), Colo	50	.....	....	.19	.....	.18	.16	.19	.18	.19	.16	.20	.18
Mercur (g.), Utah	25	.....	....	7.38	6.88	....	....	.....	.....	....	....	.....	....
Mexican (g. s.), Nev	100	.75	.65	.....	.....	.65	.36	.70	.35	1.20	.45	1.40	.84
Mollie Gibson (s.), Colo	5	.51	.45	.48	.....	.75	....	.75	.....	....	....	.75	....
Mono (g.), Cal	100	.....	....	.17	.16	....	....	.....	.....	....	....	.....	....
Moulton (s. g.), Mont	5	.....	....	.30	.....	....	....	.....	.....	.30	....	.....	....
Mt. Rosa (g.), Colo	1	.18	....	.17	.15	.16	.12	.13	.08	.10	.08	.16	.09
Occidental Cons. (g. s.), Nev	100	.....	....	1.00	.....	1.00	.85	.....	.....	....	....	1.50	1.25
Ontario (s. l.), Utah	100	11.00	7.50	11.25	11.00	....	....	15.00	13.50	....	....	.....	....
Ophir (g. s.), Nev	100	.....	....	1.20	.....	1.30	1.20	1.20	1.05	2.50	1.25	2.40	1.30
Overman, Nev	100	.....	....	.....	.....	....	....	.....	.....	....	....	.....	....
Pharmacist (g.), Colo	1	.20	.14	.18	.14	.16	.11	.11	.09	.10	.07	.09	.08
Phœnix Cons. (g.), Ariz	100	.12	.07	.08	.....	....	....	.....	.....	.03	....	.03	....
Plymouth Cons. (g.), Cal	10	.....	....	.....	.....	.20	....	.12	.....	....	....	.....	....
Portland (g.), Colo	1	1.90	1.80	1.85	.....	1.65	1.42	1.50	1.30	1.55	1.45	1.90	1.70
Potosi (g. s.), Nev	100	.65	.55	.55	.....	....	....	.30	.20	1.55	.34	1.60	1.50
Quicksilver (q.), Cal. (com.)	100	3.00	....	.....	.....	....	....	.....	.....	....	....	.....	....
Rover (g. s.), Utah	10	.....	....	.80	.....	....	....	.....	.....	....	....	.....	....
Savage (g. s.), Nev	100	.....	....	.45	.40	.36	.30	.27	.....	1.30	....	2.05	1.50
Sierra Nevada (g. s.), Nev	100	.65	..	.87	.40	.75	.60	.75	.60	1.45	.65	1.15	.70
Silver King (s.), Utah	20	.....	....	13.75	.....	....	....	.....	.....	....	....	.....	....
Small Hopes Cons. (s. l), Colo	20	.90	.85	.95	.....	1.05	.95	.90	.....	.90	....	1.75	....
Specimen (g.), Colo	1	.....	....	.....	.....	....	....	.....	.....	....	....	.08	....
Standard Cons. (g. s.), Cal	100	1.85	1.10	2 20	1.90	2.25	1.80	2.15	1.75	2.25	1.90	.....	....
Sunshine (g.), Utah	10	.....	....	3.50	.....	....	....	.....	.....	....	....	.....	....
Syndicate (g.), Cal	100	.....	....	.....	.....	....	....	.....	.....	....	....	.....	....
Tetro (s.), Utah	1	.....	....	.05	.....	....	....	.....	.....	....	....	.....	....
Union Cons. (g. s.), Nev	100	.50	.40	.40	.30	.76	.45	.65	.38	2.10	.50	1.20	.65
Union Gold (g.), Colo	1	.43	....	.....	.....	....	....	.44	.43	.90	.40	.....	....
Utah Cons. (s.), Nev	100	.....	....	.....	.....	....	....	.....	.....	....	....	.19	.16
Victor (g.), Colo	5	5.50	5.25	6.68	5.63	6.75	6.28	8.50	6.75	8.50	8.00	8.00	....
Work (g.), Colo	1	.....	....	.20	.....	.17	....	.15	.13	....	....	.....	....
Yellow Jacket (g. s.), Nev	100	.50	.45	.....	.....	.45	.38	.40	.32	.90	.40	.75	.55

FLUCTUATIONS OF MINING STOCKS AT NEW YORK DURING 1896—*Continued*

Name and Location of Company.	July.		August.		September		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Adams (s. l. c.), Colo	....	...	....	....	.....	....	.....	....	.....	.....	.....	.....	2,000
Ajax, Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	700
Alamo (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.05	.....	2,700
American Flag (s.), Colo	....	....	....	....	.....	....	.....	....	.02	.....	.....	.....	8,100
Anaconda (g.), Colo	....	....	.58	.51	.60	....	.67	.65	1.45	1.20	1.20	1.05	13,700
Anchor (g. s. l.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Andes (s. g.), Nev	....	....	....	....	.....	....	.....	....	.....	.....	.16	.....	300
Argentum-Juniata (g. s. l.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	1,050
Bedford Cons. (g.), Mont	5.00	2.75	7.00	....	7.38	....	.....	....	.....	.....	.....	.....	11,400
Belcher (s. g.), Nev	.50	.38	....	....	1.00	.60	.65	....	.52	.31	.....	.....	5,200
Best & Belcher (g. s.), Nev	1.75	.85	1.00	.93	1.25	1.00	1.10	1.00	.90	.....	.50	.....	12,500
Bodie Cons. (g.), Cal	.55	....	....	...	.70	....	.70	.64	.....	.....	.70	.....	3,400
Bullion-Beck & Champ. (g.s.l.c.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	400
Bulwer (g), Cal	.33	....	.34	.28	.46	....	.48	.45	.46	.42	.56	.43	16,400
Breece (s. l.), Colo	....	....	....	....	.....	....	.16	....	.15	.....	.....	.....	10,400
Brunswick Cons. (g.), Cal	.21	.18	.22	.19	.25	.21	.26	.15	.23	.17	.23	.17	173,100
Caledonia (g.), So. Dak	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	10
Centennial-Eureka (g. s. l. c.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	395
Chollar (s. g.), Nev	2.80	1.95	2.35	1.90	2.00	....	2.25	1.80	.....	.....	.....	.....	6,600
Chrysolite (s. l.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.11	.10	19,360
Comstock Tunnel (s. g.), Nev	.08	.07	.08	.07	.08	.06	.07	.06	.07	.06	.06	.05	100,200
Comstock Tunnel Bonds	....	....	....	....	.....	....	.07	....	.....	.....	.....	.....	8,700
Cons. California & Virginia (s.g.), Nev	2.15	1.65	1.95	1.65	2.05	1.70	1.85	.85	1.85	1.75	1.65	1.10	17,425
Cons. Imperial (g. s.), Nev	.04	....	.03	....	.....	....	.....	....	.....	.....	.....	.....	14,450
Creede & Cripple Creek (g.), Colo	.08	.04	.08	.06	.09	.07	.08	.07	.08	.07	.08	.06	109,450
Crescent (s.), Utah	....	....	....	....	.....	....	.....	....	.02	.....	.....	.....	1,500
Cripple Creek Cons. (g.), Colo	.15	.12	.12	.11	.15	.10	.13	.12	.18	.14	.15	.14	18,000
Crœsus (g.), Colo	.02	....	....	....	.02	.01	.02	....	.02	.....	.02	....	95,700
Crown Point (g. s.), Nev	.48	....	.35	.29	.36	....	.58	.50	.....	.....	.20	.19	8,650
Dalton (s.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	18,000
Daly (s. l.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	50
Deadwood-Terra (g.), So. Dak	....	....	....	....	.....	....	.....	....	1.15	.....	.....	.....	1,300
Dunkin (s. l.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	700
Eureka Cons. (s. l. g.), Nev	....	....	.28	.25	.26	....	.....	....	.35	.20	.. ..	.....	600
Father de Smet (g.), So. Dak	....	....	.21	.20	.....	....	.....	....	.....	.....	.17	.....	1,100
Favorite (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	2,000
Golden Fleece (g. s.), Colo	....	....	....	....	.....	....	....	....	.95	.....	.95	.....	28,000
Gold & Globe (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	1,000
Gould & Curry (s. g.), Nev	1.00	....	.70	.60	.53	....	.80	.70	.....	.....	.41	.31	4,470
Hale & Norcross (g. s.), Nev	1.35	1.25	1.60	1.15	1.60	1.40	1.60	....	1.20	.95	.97	.83	9,230
Homestake (g.), So. Dak	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	100
Horn Silver (s. l.), Utah	1.95	1.75	....	....	.....	....	1.50	....	1.60	1.55	.....	.....	4,050
Iron Silver (s. l.), Colo	.95	.25	.80	.20	.50	.32	.....	....	.45	.40	.....	.....	15,200
Isabella (g.), Colo	.64	.58	.46	....	.44	....	.45	.37	.39	.....	.....	.....	17,200
Julia (g. s.), Nev	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Kingston & Pembrooke (i.), Ont	.14	....	....	....	.15	....	.....	....	.....	.....	.34	.25	3,600
La Crosse (g.), Colo	.08	....	.09	.08	.....	....	.....	....	.....	.....	.....	.....	8,600
Leadville (g.), Colo	....	....	.12	....	.....	....	.11	.10	.....	.....	.....	.....	39,100
Litttle Chief (s. l.), Colo	.18	.16	.16	....	.....	....	.19	.18	.15	.....	.....	.....	14,320
Mercur (g.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	200
Mexican (g. s.). Nev	.75	.70	.65	....	.60	.45	.50	....	.66	.47	.50	.38	10,200
Mollie Gibson (s.), Colo	....	....	....	....	.....	....	.42	....	.....	.....	.....	.....	2,650
Mono (g.), Cal	....	....	....	....	.....	....	.25	....	.....	.....	.23	.....	1,700
Moulton (s. g.), Mont	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Mt. Rosa (g.), Colo	.14	.12	.13	.12	.19	.11	.20	.16	.17	.15	.15	.....	45,850
Occidental Cons. (g. s.), Nev	.80	....	....	....	.....	....	.....	....	.....	.....	.....	.....	2,100
Ontario (s. l.), Utah	....	....	....	....	10.00	....	.....	....	10.00	.....	10.50	.....	950
Ophir (g. s.), Nev	1.30	1.00	1.10	.95	1.20	.95	1.30	....	1.25	1.20	1.00	.80	7,700
Overman, Nev	....	....	....	....	.....	....	.....	....	.....	.....	.05	.....	100
Pharmacist (g.), Colo	.10	.08	.10	.08	.11	.04	.14	....	.20	.15	.15	.13	59,600
Phœnix Cons. (g.), Ariz	....	....	....	....	.06	.04	.07	.06	.20	.03	.....	.....	24,700
Plymouth Cons. (g.), Cal	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	100
Portland (g.), Colo	1.45	....	1.15	1.00	1.35	....	1.50	1.30	1.40	1.25	1.35	1.25	11,580
Potosi (g. s.), Nev	1.30	1.05	1.10	....	.90	....	.85	.60	.74	.....	.54	.....	7,100
Quicksilver (q.), Cal. (com.)	....	....	....	....	.....	....	.....	....	2.00	.....	.....	.....	300
Rover (g. s.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	100
Savage (g. s.), Nev	.70	.54	.70	....	.55	....	.50	.45	.70	.55	.....	.....	5,300
Sierra Nevada (g. s.), Nev	.60	.55	.70	.45	.60	.	.70	.60	.....	.....	.40	.38	16,865
Silver King (s.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	400
Small Hopes Cons. (s. l.), Colo	....	....	.70	.65	.....	....	.....	....	.....	.....	.....	.....	4,700
Specimen (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Standard Cons. (g. s.), Cal	1.45	1.30	....	....	1.00	....	.....	....	1.50	1.25	1.30	.....	10,000
Sunshine (g.), Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Syndicate (g.), Cal	....	....	....	....	.02	....	.04	.02	.04	.....	.....	.....	4,950
Tetro (s.) Utah	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	300
Union Cons. (g. s.), Nev	.53	.34	.50	.21	.44	.26	.52	.30	.60	.50	.....	.....	31,150
Union Gold (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	3,900
Utah Cons. (s.), Nev	....	....	....	....	.....	....	.....	....	.....	.....	.38	.....	3,000
Victor (g.), Colo	7.25	6.50	7.00	....	7.00	6.88	7.00	6.75	7.00	6.75	6.75	6.50	17,040
Work (g.), Colo	....	....	....	....	.....	....	.....	....	.....	.....	.....	.....	3,070
Yellow Jacket (g. s.), Nev	.47	.42	....	....	.35	.20	.65	.25	.52	.40	.43	.27	7,350

NOTE.—Total sales for the year, 1,117,815 shares.

PRICES OF INDUSTRIAL AND

Name of Company	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Acetylene L. H. & P.	$25	22.00	10.00	23.50	20.00	20.00	.....	.....	.....	.....	.....	.....	.....
Baltimore & Ohio	100	44.00	36.00	41.00	21.63	20.00	13.00	19.00	16.13	16.75	12.00	19.25	16.75
Cambria Iron	50	48.00	40.50	47.25	44.50	46.75	39.88	43.00	41.00	43.00	40.00	45.00	40.00
Chesapeake & Ohio	100	16.25	13.00	18.13	13.00	17.00	14.25	18.13	16.00	17.50	16.38	16.88	13.13
Choctaw & Gulf Certificates	50	12.75	8.75	9.75	7.50	8.13	6.38	9.00	5.00	8.50	6.00	7.75	5.25
Colorado Fuel and Iron	100	28.50	23.38	.....	.....	32.00	26.00	32.50	27.00	31.00	26.50	29.00	22.00
Columbus, Hoc. Val. & Toledo.	100	18.75	15.00	18.50	16.50	17.50	15.00	18.50	15.50	17.25	16.00	17.50	14.50
Cons. Gas, New York	100	.....	.....	161.88	154.00	.....	.....	.....	.....	.....	.....	.....	.....
Delaware & Hudson	100	129.13	119.75	130.00	126.50	128.25	125.50	128.13	126.00	128.00	125.38	127.75	124.25
Delaware, Lack. & Western	50	164.00	155.88	163.75	160.63	162.50	160.13	161.50	159.13	161.75	159.00	164.00	159.88
General Electric	100	27.88	22.00	33.50	27.50	30.50	21.75	38.75	36.25	36.88	33.25	34.63	27.00
H. & Broad Top	50	31.00	28.00	29.75	29.00	.....	.....	28.00	.....	.....	.....	.....	.....
H. & Broad Top, pref	50	53.00	51.00	52.50	51.50	52.00	51.75	53.25	51.00	52.50	52.00	52.50	52.00
Lake Erie & Western	100	21.88	18.00	22.50	20.50	20.50	17.50	21.00	17.50	19.75	16.63	19.75	17.25
Lake Erie & Western, pref	100	74.00	64.63	75.13	73.00	75.00	69.75	73.50	69.75	72.38	69.75	72.00	69.00
Lehigh Coal and Navigation	50	46.00	42.00	56.00	43.38	45.25	41.50	43.50	41.50	42.50	39.00	43.00	40.00
Lehigh Valley	50	38.50	35.00	38.25	36.00	36.50	34.00	36.00	34.25	35.00	33.75	34.25	30.50
Little Schuylkill	50	65.00	62.75	64.50	.....	.....	.....	65.50	63.50	63.50	63.00	63.50	.....
National Lead	100	26.88	23.00	28.50	25.00	24.88	22.50	28.63	24.00	26.75	24.00	26.50	23.13
National Lead, pref	100	88.00	82.50	90.25	86.25	87.75	85.50	92.00	86.50	92.00	90.00	90.50	85.00
New Jersey Central	100	108.50	94.75	109.00	104.50	108.50	104.00	109.25	105.88	107.50	103.75	108.00	102.25
N. Y., L. E. & Western	100	16.50	13.50	17.63	15.88	16.50	14.50	.....	.....	.....	.....	.....	.....
N. Y., L. E. & Western, pref	100	25.00	23.00	29.00	26.50	.....	.....	.....	.....	.....	.....	.....	.....
N. Y., Ontario & Western	100	15.75	12.50	15.88	15.00	15.25	14.00	15.25	14.13	15.13	14.25	14.88	13.00
N. Y., Sus. & Western	100	11.50	8.25	12.00	9.50	10.13	7.75	9.00	8.00	9.50	8.50	9.00	8.00
N. Y., Sus. & Western, pref	100	30.25	21.50	31.25	27.00	28.00	21.50	25.50	21.75	25.25	23.50	25.25	21.88
Norfolk & Western	50	2.38	1.50	3.25	2.63	2.50	1.63	1.50	1.13	2.25	1.13	4.75	4.00
Norfolk & Western, pref	50	8.25	6.13	9.25	8.00	7.50	6.00	6.13	4.50	8.50	3.75	10.75	9.75
Pennsylvania Railroad	50	52.25	50.00	.....	.....	.....	.....	.....	.....	.....	.....	53.38	52.25
Pennsylvania Salt Mfg. Co	50	.....	.....	.....	.....	103.00	.....	.....	.....	.....	.....	.....	.....
Pennsylvania Steel	50	35.00	25.00	35.00	30.00	33.00	25.00	30.00	26.00	26.50	25.50	25.00	.....
Pennsylvania Steel, pref	50	65.00	60.00	65.00	.....	55.00	.....	60.00	59.50	57.00	54.00	57.50	52.00
Philadelphia & Reading	50	15.00	2.38	15.00	11.50	12.25	9.75	13.13	10.25	13.00	9.88	16.00	7.75
Tennessee Coal and Iron	100	29.00	23.00	34.13	29.00	31.50	24.75	32.00	28.75	30.50	25.13	27.00	21.25
United Gas Imp	50	69.00	66.25	70.25	66.50	70.00	66.00	71.25	68.50	73.50	70.25	71.75	70.00
Welsbach, Can	5	2.75	2.50	2.75	2.50	2.50	2.00	3.25	2.13	2.75	2.13	5.00	2.00
Welsbach, com	100	40.00	38.00	42.00	38.00	39.00	38.00	38.00	35.00	30.00	.....	30.00	20.00
Welsbach, pref	100	74.88	73.88	77.00	74.00	72.00	68.88	69.00	67.50	68.00	65.00	64.00	.....
Welsbach Light	5	57.50	49.88	67.00	57.00	62.50	56.50	60.50	58.00	58.50	50.50	51.50	47.00
Westmoreland Coal	50	52.00	50.13	52.50	51.50	.....	.....	51.00	50.00	50.00	.....	.....	.....
Wheeling & Lake Erie	100	12.25	10.00	13.25	10.75	11.88	8.25	11.63	9.25	10.88	9.50	10.63	8.50
Wheeling & Lake Erie, pref	100	38.00	34.50	40.38	37.50	38.25	31.75	38.00	31.75	37.63	35.00	37.00	34.00

Globe Hill Mining Company, Mount Rosa Mining, Milling and Land Company, Mammoth Mining Company, Mercur Gold Mining and Milling Company, Pharmacist Mining Company, Rubicon Gold Mining Company, Silver King Mining Company, Sunshine Gold Mining and Milling Company, Specimen Gold Mining and Milling Company, Union Gold Mining Company, Work Mining and Milling Company. The major portion of these stocks belong to Colorado, principally to the Cripple Creek district. On August 25 the Fortuna Gold Mining and Milling Company was added to the permanent list, and on June 26 the Bedford Consolidated Mining and Smelting Company, of Montana, was appended to the regular list of the exchange. The following stocks were stricken from the temporary and regular lists during the year: Bogan Silver, Dalton Gold, Mercur Gold, Tetro, Rover Gold (all of Tintic Mining District, Utah), Argenta, Bechtel, Bradshaw, Consolidated Pacific, Carson River Dredging Company, Columbia Mining Company, Del Monte, Diana, Excelsior Gold and Silver Mining Company, Elko, Emmett Water and Mining Company, Freeland, Found Treasure, Grand Prize, Gold Stripe, Kossuth, Martin White, Monitor, Mutual Smelting and Milling Company, Mount Diablo, North Belle Isle, Proustite, Reward Mining and Smelting Company, Sutro Tunnel Company, Sutro Tunnel Trust Certificates, Swinam, San Sebastian, Sullivan, Scorpion, Tioga, United Copper. There were placed on the suspended list the fol-

COAL STOCKS DURING 1896.

Name of Company.	July.		August.		September.		October.		November.		December.		Sales.
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	
Acetylene, L. H. & P	.....	.....	.....	.....	30.00	.....	.....	.....	.....	.....	.....	.....	673
Baltimore & Ohio	18.50	15.50	16.00	12.25	14.50	10.50	15.50	13.50	18.00	15.75	17.00	12.50	360,157
Cambria Iron	40 00	35.00	44.00	34.00	44.00	35.50	41.00	39.00	47.75	39.75	44.00	42.50	12,793
Chesapeake & Ohio	14.88	12.13	13.50	11.00	15.75	13.00	16.13	13.38	18.50	16.13	17.13	15.00	204,471
Choctaw & Gulf Certifs	6.75	5.00	5.50	4.00	7.00	4.80	7.50	.....	8.38	7.25	8.00	7.13	85 135
Colorado Fuel and Iron	22.25	16.00	18.88	14.63	21.50	17.00	20.75	16.50	28.25	19.50	25.75	22.00	110,780
Columbus, H. V. & Toledo	17.25	15.00	15.00	12.50	15.88	13.00	17.00	15.00	20.50	16.25	19.75	16.50	84,547
Cons. Gas, New York	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	100
Delaware & Hudson	125.00	115.50	119.75	115.00	124.88	118.25	124.75	119.63	128.88	124.00	128.25	124.38	58,688
Delaware, Lack. & West	160.50	146.00	150.00	138.00	158.00	150.00	156.75	152.75	160.50	156.00	160.25	155.00	35,327
General Electric	27.63	20.00	24.63	21.13	29.50	23.50	29.50	24.50	35.50	29.25	33.88	29.00	754,556
H. & Broad Top	.....	.....	.....	.....	27.00	24.00	24.50	23.25	25.00	24.00	24.75	24.50	1,138
H. & Broad Top, pref	53.00	51.75	50.00	48.75	49.25	48.50	50.00	49.00	50.50	49.75	51.00	.....	1,679
Lake Erie & Western	18.25	13.25	14.88	12.13	16.75	14.75	18.00	14.75	20.75	18.00	19.25	18.00	23,834
Lake Erie & West., pref	70.38	60.00	63.75	55.50	68.75	63.00	68.75	58.50	73.75	69.25	73.00	69.50	26,691
Lehigh Coal and Nav	43.25	38.00	39.00	35.00	42.00	39.00	43.00	41.00	45.25	41.00	51.50	40.00	9,347
Lehigh Valley	33.00	30.00	31.00	28.00	30.13	29.00	30.25	28.00	34.25	30.13	33.00	30.00	68,251
Little Schuylkill	.....	.....	60.00	59.88	.....	.....	60.00	59.00	.....	.....	50.75	.....	471
National Lead	23.00	18.75	19.00	16.00	23.00	19.00	25.50	19.25	28.00	24.50	25.63	23.13	105,233
National Lead, pref	86.25	79.00	83.00	76.50	85.00	78.00	89.00	82.00	92.75	88.00	90.00	86.00	27,716
New Jersey Central	103.25	87.75	95.00	87.50	104.75	94.25	104.50	98.00	110.00	101.00	105.00	101.50	189,216
N. Y., L. E. & Western	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	16,259
N. Y., L. E. & W., pref	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,170
N. Y., Ontario & Western	14.00	12.13	12.63	11.13	14.38	12.25	14.50	13.00	16.88	14.38	16.00	14.00	148,693
N. Y., Sus. & Western	8.50	7.25	8.00	6.00	9.13	7.50	9.00	7.25	12.00	9.00	10.00	9.50	39,771
N. Y., Sus. & West., pref	22.50	17.00	18.25	12.00	23.88	19.00	24.75	19.50	30.00	24.75	27.00	24.00	153,838
Norfolk & Western	6.00	2.50	8.25	8.00	9.75	8.50	11.00	9.75	12.75	11.25	.....	.....	8,518
Norfolk & Western, pref	11.75	8.75	13.38	11.00	13.88	13.50	16.50	14.25	19.75	17.00	17.13	.....	7,921
Pennsylvania Railroad	53.00	50.38	51.00	49.00	52.00	50.13	52.50	51.25	53.00	51.88	52.25	51.75	92,774
Pennsylvania Salt Mfg. Co	.....	.....	.....	.....	.....	.....	101.00	99.00	.....	.....	.....	.....	64
Pennsylvania Steel	25.00	20.00	21.75	20.00	30.00	25.00	35.00	27.00	39.50	36.00	.....	.....	3,558
Pennsylvania Steel, pref	51.00	.....	40.00	.....	55.00	40.00	62.50	57.00	70.00	.....	.....	.....	860
Philadelphia & Reading	14.25	10.00	10.50	5.88	20.00	17.63	27.50	18.00	31.75	27.38	29.88	23.13	2,142,480
Tennessee Coal and Iron	21.75	13.00	17.63	14.00	23.25	17.25	25.13	19.88	32.50	25.00	29.63	24.00	884,917
United Gas Imp	68.00	62.50	65.00	58.00	68.50	63.00	69.00	66.75	74.50	69.50	73.88	71.50	43,064
Welsbach, Can	2.13	1.50	1.75	1.13	2.00	1.13	2.00	1.38	2.50	1.88	2.13	2.00	13,116
Welsbach, com	20.00	14.50	17.00	15.00	25.00	20.00	23.00	17.50	30.00	20.38	.....	.....	4,034
Welsbach, pref	59.00	50.00	60.00	53.00	60.75	59.75	65.00	60.00	70.00	65.00	66.00	62.00	4,191
Welsbach Light	45.00	34.50	40.00	35.00	44.50	39.00	45.00	40.00	57.00	44.50	48.50	39.00	26,449
Westmoreland Coal	.....	.....	50.00	48.00	48.00	.....	47.00	.....	50.00	.....	50.00	47.25	364
Wheeling & Lake Erie	9.50	6.63	7.63	5.25	6.75	5.88	6.50	5.38	9.25	6.50	8.63	7.00	77,685
Wheeling & L. E., pref	33.63	23.75	25.25	21.00	27.00	23.38	29.25	24.88	35.50	29.25	32.13	28.25	21,273

NOTE.—Total sales for the year, 5,852,202 shares.

lowing: Augusta Mining and Investment Company, stock and bonds, Colchis, Helena, Holyoke, Hestor, Lee's Basin Mining Company, Monte Cristo, Oriental & Miller, Plutus, Robinson Consolidated, Ruby Silver Mining Company, stock and bonds, Stormount, Silver Queen, Santiago, Shoshone, Trio, Tornado, Columbia & Beaver, and Cleveland Tin Mining Company.

The Consolidated Stock and Petroleum Exchange is making an effort to weed out all undesirable stocks, and to replace them by better known ones. Many of the stocks that are to be dropped from the list of the exchange have not been traded in for many months, and in some cases for years.

Year by year the reported sales of mining securities have grown smaller, and 1896 would doubtless have been as bad as its predecessors had not the Cripple Creek stocks been traded in. The transactions on the Consolidated Exchange for eight years are given in the table below:

Year.	Shares Sold.	Year.	Shares Sold.
1889	4,114,480	1893	*624.617
1890	3,925,926	1894	302.372
1891	2,522,660	1895	678,925
1892	1,527,371	1896	1,117,815

Much of the trading in these securities which was centered in New York years ago has gradually disappeared. In the majority of cases speculators prefer to deal direct with brokers who are at or near the mines in whose shares they want to invest capital.

Industrial and Coal Stocks.—In the foregoing table prices are given for the stocks of coal companies, coal carrying railroads, and industrial companies which are dealt in largely on the New York Stock Exchange. We have included in the list also a number which are quoted in Philadelphia but not in New York, so that the table covers the exchanges of both cities.

The Philadelphia Mining Stock Market in 1896.

There are many stock transactions in Philadelphia, but the shares of mining companies are not in favor, with the exception of the coal companies and a few iron and steel companies. The gold, silver, copper, and lead stocks are entirely ignored. In recent years also the oil companies have practically been dropped.

Generally the market follows New York quotations very closely. There are a few exceptions found in Pennsylvania stocks which are not found in the New York lists. Among these are Cambria Iron Company, which varied between $34 and $48, closing at $42.50; Pennsylvania Steel, whose limits were $20 and $39.50, closing at the highest price; Huntingdon & Broad Top, which touched $31 and $23.25, closing at $24.50; Lehigh Coal and Navigation, which ranged from $35 to $56, closing at $40; Little Schuylkill, from $50.75 to $65, closing at the lower figure; Westmoreland Coal, which sold at $52 in January, in December dropped to $47.25. A few sales of Pennsylvania Salt Manufacturing Company stock were recorded at $99 to $103, the par value being $50. The Welsbach light shares are especial favorites and are actively dealt in.

The Pittsburg Mining Stock Market in 1896.

Dealings in this market are confined to coal and natural gas stocks, those of local companies, with very few exceptions. The market was not at all active during the year and speculation was very limited.

FLUCTUATIONS OF PRICES IN MINING STOCKS AT PITTSBURG DURING 1896.

Company.	Par Value.	Opening.		Highest and Lowest During the Year.				Closing.	
				Bid.		Asked.			
		Bid.	Asked	Highest.	Lowest.	Highest	Lowest.	Bid.	Asked
Coal—									
Mansfield C. & C., Pa	$50	.....	37½	.....	.....	37½	.....	.....	.....
New York Gas Coal	50	.....	.....	40	37¾	40	.....	.....	.....
Lustre, Mex	10	13¾	14½	13¾	8	14½	9¾	7	.....
Natural Gas—									
Chartiers Valley	100	6½	7	6	4½	10	5	.....	6
Manufacturer's, Pa	50	.....	45	50½	45	54	45	.....	.....
People's Natural Gas, Pa	50	25	.....	31	25	32½	27½	.....	.....
People's Natural Gas and Pipeage, Pa	25	.....	14½	14¼	12⅜	15⅝	13	14	14¾
Pennsylvania, Pa	50	.....	.....	5	3	6⅝	5½	6	6½
Philadelphia, Pa	50	17⅛	18⅛	20½	16¼	20⅞	17⅞	17	18
Wheeling, W. Va	50	16⅛	18	.....	18½	19	17¾	18	18½

The only outside mining stock dealt in was Lustre, a Mexican company owned by Pittsburg people. This stock suffered somewhat during 1896 and closed at the lowest point of the year. The natural gas companies' shares were nearly all depressed.

The Salt Lake City Mining Stock Market in 1896.

By James A. Pollock.

Utah's mining share market during the year 1896 reflected practically all the conditions predicted a year before. More attention was given to the meritorious mining securities in this State by outside investors, both foreign and domestic, than for many years past, if not more than for any previous year. Outside investors turned largely to the dividend-paying stocks, the speculatives naturally receiving the most attention at the hands of local operators. As has been said many times, this market is more of an investment than a speculative one, and while quotations often fluctuate somewhat wildly, these violent alterations are more the result of a small, sensitive market than any real changes in the properties affected. In Utah there have been few examples of stock manipulation, and this sort of thing is most seriously frowned down by the genuine mining operators.

Almost without exception Utah mining companies are honestly and conservatively managed, and there is little or no waste of company funds or energy. The year 1896 has been a splendid one for Utah listed stocks in a dividend way, five new companies having entered the dividend list and two old-time payers having resumed payments. The total amount paid in dividends by the listed stocks during 1896 was $2,384,500, swelling the total to $32,368,500, with the total of many private payers not included in the list. In point of amount paid during 1896 the Centennial-Eureka led with $450,000 to its credit, closely followed by the Silver King with $412,500; then the Bullion-Beck with $295,000, the Eureka Hill with $250,000, the Mercur with $225,000, the Gemini with $200,000, and the Ontario with $180,000. The new payers were the Dalton & Lark, Sacramento, Swansea, South Swansea, and Utah Consolidated, while the Mammoth and Ontario resumed payments after several years' lapse. Daly also resumed dividends, but made only one distribution, that in August. Dividends paid by Utah mining companies producing silver, lead and copper are now the result of only the most economical management.

Several heavy deals in Utah mining properties reached a successful culmination during 1896, while a number of additional sales of great magnitude are now pending. Among those successfully closed were the sale of a large portion of the capital stock of the Sevier Gold Mining and Milling Company, the sale of a half-interest in the capital stock of the Sacramento Gold Mining and Milling Company, and the transfer of the Highland Boy group at Bingham.

Conditions surrounding Ajax improved very materially and the stock made decided gains during the year, with the tendency still upward. Regular and heavy shipments of high-grade copper, silver, and gold ore are coming from the properties.

Alliance was dead. There were practically no buyers in the field, and the

stock closed at the lowest point in its history. Some development work is being done by the company, but heavy litigation has interfered with the operations. No new assessments were levied.

With all work suspended at both mines and mill, Anchor ended the year in not very encouraging condition, but the stockholders have great faith in the company and sellers were comparatively few. From $2.25 at the first of the year the stock declined to $1.00.

Bogan had an uphill course. No work is now being done at the properties and more assessments will be necessary before development work can be resumed. It seems not at all unlikely that the control will change within the next few months. The stock was up and down several times, but closed at a very low ebb.

With dividend payments aggregating $295,000 for the year, Bullion-Beck closed a splendid record. On the average market value of the stock the company paid nearly 50% in dividends in 1896, which has been its record since 1893. During the summer months the stock declined somewhat, but ended the year about where it opened in January.

Centennial-Eureka attracted great attention at the hands of investors by reason of its regular dividend payments. The stock advanced to above $75 per share, a gain of over $15, and closed the year practically out of the market, on account of a pooling of interests. For 1896 the company paid $450,000 in dividends, regularly $1 per share monthly, or $30,000, and several times $2 per share.

Dalton did little during the year. The stock sold down to its lowest point and did not recover materially, going around 3c. Work of a developmental nature was prosecuted continually during the year. Two assessments of 1c. each were levied.

With the settlement by contract purchase of its legal difficulties with its neighboring companies, the Dalton & Lark Company placed a portion of its capital stock on the market at 50c. Dividends of ½c. per share commenced at once and were continued for six months, when a suspension was ordered by the directors on account of the low price of the metals. Very little of the stock was taken by local investors, but it is understood that the promoters placed considerable of it in the East. The company is too heavily capitalized, having 2,500,000 shares, and this is one reason the stock never took with home buyers. Very light inquiry and fairly heavy offerings caused the close to be weak, the stock being offered as low as 30c.

Stockholders in the Daly were very agreeably surprised in August by the payment of a dividend of 25c. per share. It was then supposed that payments would be repeated at least quarterly, but no further dividends were made. The stock fluctuated considerably and closed nearly $1 a share under the opening.

Daly-West made heavy gains, as the result of an improved showing at the properties and an increase in interest by investors. The stock showed gains of $1.50 for the year, closing above $8.

Since early in the year no work was done at the Eagle and the stock came down from 25c. to 5c., with no activity. The properties are located in the Mercur district.

Starting at 50c., boomed by "wash" sales and matched orders to 60c. and then gradually declining to ½c. per share, with a pending assessment of that amount, was the history of the East Golden Gate stock. As was the case with Dalton & Lark, little of this stock was sold locally, but great quantities were placed throughout the East and West at 50c. and 60c. per share. The company owns two claims in the Mercur district and sank 1300 ft. with a 6-in. drill, in the hopes of striking the vein, but secured no pay values. The chief promoters of the company, and, seemingly at least, the only people who made any money out of the deal, except the original holders of the ground, changed their base of operations to San Francisco. An assessment of ½c. per share was levied to pay debts against the company. A very good hoisting plant is on the company's ground.

There was little real change between the opening and closing price of Four Aces. At one time, just after the close of a sale of treasury stock, an advance to 45c. resulted from speculative buying, but the inevitable break came and the stock sold down to 10c., recovering to 20c. The company is working its claims in the Tintic district energetically, and makes occasional shipments of silver-lead ore. With depth the ore showing is expected to very materially improve, as has been the case with both its neighbors, the Swansea and South Swansea.

Although the Galena sustained a slight loss in quotations, there was no good reason, as the company paid regularly a dividend of 5c. per share, paying a total of $55,000. The company contemplates some important improvements for 1897, including a new steam hoist and a new working shaft. At one time, early in the year, the stock sold above $2, but closed about $1.50. It was March before the stock became at all active.

In January the Geyser came on the market at $1.40 per share, but this figure was not long sustained, the close being about $1 with the tendency upward. The company owns properties in the Mercur district and its 50-ton mill has been increased to a daily capacity of 100 tons.

Horn Silver paid but one quarterly dividend during the year, that in January, of 12½c. per share. The company continued operations throughout the year and rebuilt its surplus fund, although evidently not sufficiently to warrant a continuation of dividends.

The market price of Lucky Bill was almost doubled as the result of the uncovering of ore bodies in the properties. The usual 2c. assessment came regularly until December, but it is thought the company will soon be self-supporting.

Commencing in May, the Mammoth resumed dividend payments. In certain quarters a dividend of at least 10c. per share was expected, and on these expectations the stock sold up as high as $4, but a break came and the fluctuation carried the security down as low as $1.50, with variations. The company paid an even $100,000 in dividends, and in October ceased payments in order that some heavy improvements at its properties might be made. Two increases in the stamp capacity of the company's mill were made.

After the first few months of the year, Morgan (Meears) did little and the stock suffered materially. The company owns good ground at Park City. At the close stock was offered at 20c.

Mercur maintained its dividend record in good shape, and the stock was much stronger at the close than at the opening. A total of $225,000 was paid in dividends, and the company equipped three hoisting works, increased the capacity of its mill, and purchased an adjoining group of rich ground in addition.

Northern Light was one of the new stocks to come upon the board and it made a good record, advancing about 30% within the past two months. The stock sold as low as 35c., but closed between 60c. and 70c.

With the month of January the Ontario resumed dividends, paying 10c. per share and distributing a total of $180,000 for the year. The stock sold up as high as $14, but on the reaction went down to $9 and closed at $10.

Through the suspension of operations at the company's properties, Rover suffered a decided decline, and instead of selling at $1.25 as at the opening, it closed at 45c., but with the tendency upward. Operations are to be resumed in the near future.

During 1896 Silver King paid in dividends the total sum of $412,500, missing one month through litigation involving its water supply. The stock gained $3 in value and is phenomenal for the light offerings. The company paid regularly 25c. per share monthly.

Incorporated with a capital stock of $5,000,000, upon which it immediately commenced dividend payments of $5000 monthly, or ½c. per share, the Sacramento stock came on the market in the last month of the year. Having no treasury stock, it is taken for granted that the company's principal stockholders are marketing some of their holdings. The Sacramento properties are near those of the Mercur and the company operates a 50-ton cyanide plant.

Sunshine proved a disappointment to its friends, although the majority of its stockholders are still warm in the faith. The company owns a group of mines in the Mercur district, but has not as yet been able to save all of the values in its ores with its cyanide plant. A number of improvements are to be made at once, and these are expected to remedy all defects. Although able to pay expenses with the proceeds of its cyanide sales, the company was not able to pay the expected dividends, and as a result the stock closed in the neighborhood of $1, as compared with $3.50 at the opening.

A good record was made by the Swansea, a new stock, and one of the new dividend-payers. Although a silver-lead proposition, the ore is found in such quantities that the company was enabled to pay dividends of 5c. per share. The stock came on the market under 20c., and by September 1, when dividends were commenced, had advanced to $2.50, where it closed.

South Swansea is a direct neighbor of the Swansea, both being in the Tintic district, and the same conditions surround it as at the Swansea. The company commenced dividend payments with the close of the year, and its stock reached $1.60, with the tendency upward. The South Swansea is capitalized more heavily than is the Swansea, its number of shares being 150,000, but it paid 5c. per share monthly. On the assessment of 1c. per share Tetro pressed development work continuously. Utah paid regular dividends of 2c. per share up to November, when it ceased in order that improvements might be made at the properties with the money which had always gone to the stockholders. Its

dividends for the year aggregated $20,000. The stock closed higher than it opened, although at one time it sold at considerably higher figures.

One dividend of 2c. per share was paid by the Utah Consolidated of Tintic, aggregating the sum of $2000. The low price of the metals curtailed shipping, however, and as a result the stockholders reaped little benefit from their holdings.

Although only a prospect at the commencement of the year, the Mercur Gold Dust developed into good property under energetic management. The properties of the company are located in the Mercur district. The treasury stock of the company was placed at 50c. per share, and no sales have taken place under that figure.

The San Francisco Mining Stock Market in 1896.

Gloom again settled down over the Mining Stock Exchange in this city and the year 1896, during which at one time things looked so propitious, closed with a darker outlook than ever noted before. The number of dealers operating on the market has been decreasing rapidly, and most of the old-time leaders have left in disgust, the margin of profits in comparison with the risk being too small to suit their ideas. The main objection on their part is that they have no one of their own caliber to play against.

It will be remembered, however, that the situation was very much the same at the close of 1895, with prices even lower. It looked then as though the end had come, yet within a very few weeks the street was ablaze with excitement over a strike of ore on the Brunswick ground, newly acquired by the companies. The reported failure of this same development plunged the market and all connected with it into the depths of despair.

The majority of the heavier class of operators lost considerable money during the year on the operations, meeting a Waterloo at last in the bear raid on Brunswick shares. These people backed Chollar to win out as a bonanza on development at depth, but hopes were not realized, although the prospects all along have been most favorable and alluring. The ore encountered in the spring, which sent prices booming in all directions, was first cut on the south drift from the Norcross-Chollar shaft a few feet south of the Chollar line pitching toward Potosi. On opening up the 300-ft. level, the same ore was found in a crosscut run east from the south drift, 54 ft. from the foot wall, which was tapped at this point. The ore was of fine milling character and high grade, with a large percentage of gold. Another ledge was cut on this level about 3 ft. from the foot wall, of an entirely different character of ore of lower grade and very irregular. At the same time some streaks of ore were cut north of the Chollar line in Norcross territory, but they pinched out, conveying the impression that the mineralization of the ground was bunchy.

The hopes of the management were centered upon the 400-ft. level, to which the inclined shaft had been sunk in the meantime. Should they get the ledge there cut up on the 200 and 300-ft. levels above, they would have enough ore in sight to sustain their belief in a large existing deposit of ore at that point. The price of the stock was well maintained up to this period, Chollar selling on one spurt up as high as $4.30, with a corresponding advance on a smaller

FLUCTUATIONS OF MINING STOCKS

Name and Location of Company.	Par Value.	January.		February.		March.		April.		May.		June.	
		H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alpha, Nev.	$100	.29	.18	.18	.10	.14	.08	.15	.05	.37	.10	.25	.12
Alta Nev.	100	.14	.06	.15	.06	.19	.12	.16	.07	.25	.06	.28	.07
Andes, Nev.	100	.30	.17	.18	.10	.36	.25	.33	.24	.55	.26	.58	.27
Belcher, Nev.	100	.38	.25	.28	.20	.34	.18	.47	.20	1.10	.45	.90	.41
Benton Cons., Nev.	100	.....	.....	.....	.....	.....	.....	.25	.15	30	.10	.30	.15
Best & Belcher, Nev.	100	1.05	.81	.85	.68	.72	.59	.65	.39	2.10	.50	1.70	.76
Bodie, Cal.	100	.47	.38	.40	.30	.35	.16	.50	.22	1.10	.48	.72	.37
Bonanza, Nev.	100	.....	.....	.....	.....	.....	.....	.....	.....	.25	.17	.15	.....
Bullion, Nev.	100	.22	.16	.17	.10	.13	.06	.15	.02	.41	.13	.45	.15
Bulwer, Cal.	100	.....	.....	.....	.....	.....	.....	.43	.19	.51	.27	.45	.25
Caledonia, Nev.	100	.11	.08	.09	.06	.10	.06	.08	.02	.20	.04	.20	.10
Challenge, Nev.	100	.87	.55	.66	.47	.51	.42	.40	.21	.81	.35	.56	.30
Chollar, Nev.	100	.....	.....	.....	.....	.....	.....	.68	.33	3.10	.55	4.30	2.15
Confidence, Nev.	100	1.25	1.00	1.10	.85	1.05	.78	.90	.65	1.70	.90	1.35	.90
Cons. Cal. & Va., Nev.	100	2.70	2.35	2.40	1.75	1.80	1.55	1.90	1.55	3.50	1.65	3.20	1.80
Cons. Imp., Nev.	100	.03	.02	.02	.....	.02	.01	.02	.01	.02	.....	.04	.02
Cons. New York, Nev.	100	.06	.....	.06	.....	.04	.03	.05	.03	.15	.03	.10	.04
Crown Point, Nev.	100	.34	.19	.40	.18	.66	.29	.42	.25	.96	.31	.79	.42
East Sierra Nevada, Nev.	100	.....	.....	.....	.....	.....	.....	.05	.04	.06	.02	.06	.03
Eureka Cons., Nev.	100	.....	.....	.....	.....	.....	.....	.25	.....	.25	.22	.25	.....
Eureka Cons. Drift, Cal.	1	.....	.....	.....	.....	.....	.....	.50	.....	.....	.....	.....	.....
Exchequer, Nev.	100	.10	.06	.07	.03	.06	.03	.05	.03	.20	.03	.15	.03
Gould & Curry, Nev.	100	.59	.47	.46	.31	.35	.26	.44	.18	2.00	.41	1.75	.85
Hale & Norcross, Nev.	100	1.45	1.05	1.45	1.10	1.45	1.05	1.40	1.05	3.10	1.10	3.40	2.00
Jackson, Nev.	100	.....	.....	.....	.....	.....	.....	.50	.40	.40	.....	.40	.....
Julia, Nev.	100	.03	.....	.07	.06	.....	.....	.07	.04	.26	.08	.20	.04
Justice, Nev.	100	.05	.02	.12	.02	.11	.10	.10	.02	.25	.03	.13	.04
Kentuck, Nev.	100	.08	.04	.07	.03	.07	.05	.06	.02	.11	.03	.08	.02
Lady Washington, Nev.	100	.02	.01	.06	.01	.05	.04	.05	.....	.06	.04	.06	.03
Mayflower Gravel, Cal.	20	.....	.....	.....	.....	.....	.....	.....	.....	1.05	1.00	.....	.....
Mexican, Nev.	100	.72	.57	.62	.50	.57	.49	.38	.25	1.40	.31	1.35	.66
Mono, Cal.	100	.....	.....	.....	.....	.....	.....	.16	.10	.20	.05	.11	.04
Nevada Queen, Nev.	100	.....	.....	.....	.....	.....	.....	.05	.....	.05	.....	.06	.04
North Gould & Curry, Nev.	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.09	.....
Occidental, Nev.	100	1.20	.62	1.20	.81	1.05	.76	1.50	.84	2.35	1.10	1.80	.78
Ophir, Nev.	100	1.65	1.30	1.35	1.20	1.25	1.15	1.30	.83	2.90	1.15	2.40	1.15
Overman, Nev.	100	.26	.16	.18	.13	.19	.12	.14	.08	.30	.09	.47	.13
Pioneer, Cal.	10	.....	.....	.....	.....	.....	.....	.....	.....	1.50	1.00	1.50	1.10
Potosi, Nev.	100	.70	.50	.55	.39	.45	.36	.42	.20	1.90	.26	2.00	.90
Savage, Nev.	100	.39	.26	.52	.32	.50	.31	.37	.23	1.65	.28	2.10	.89
Scorpion, Nev.	100	.04	.02	.03	....	.05	.02	.05	.02	.11	.02	.15	.05
Seg. Belcher & Mides, Nev.	100	.13	.08	.09	.06	.11	.04	.17	.03	.31	.14	.26	.11
Sierra Nev, Nevada	100	.70	.41	.43	.33	.75	.30	.67	.54	1.60	.59	1.30	.55
Silver Hill, Nev.	100	.03	.02	.02	.....	.02	.01	.02	.01	.08	.01	.08	.02
Silver King, Ariz.	100	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Syndicate, Cal.	100	.....	.....	.....	.....	.....	.....	.03	.01	.05	.02	.05	.02
Union Cons. Nev.	100	.54	.35	.55	.33	.78	.43	.66	.45	1.35	.50	1.30	.52
Utah, Nev.	100	.15	.04	.14	.08	.11	.07	.10	.03	.30	.04	.21	.07
Yellow Jacket, Nev.	100	.59	.41	.45	.32	.53	.30	.42	.29	.90	.34	.85	.40

scale in all of the other Brunswick shares. The south drift was run out after considerable delays, and finally crosscutting began north and south of the Chollar line. The Norcross people ran out east about 12-ft. north of the line on the ground for a distance of 90 ft. without finding the hanging wall, cutting a streak of high-grade ore, which gave out and they stopped discouraged. To the south in Chollar west, crosscuts were put into the foot wall without finding the streak cut there on the 300-ft. level, and two east crosscuts run toward the hanging wall failed to find the ledge cut above on the 200 and 300-ft. levels, although in the most southerly a small streak was found, supposed to be the same as that cut over the line in Norcross, 21 ft. in from the lateral drift.

This unsatisfactory result of exploration gave rise to the cry of failure and Chollar dropped below the dollar mark again, carrying the whole line with it. The Norcross Company drew out of the struggle, and not only stopped work on the Brunswick, but also in the old mine on the Comstock, pending the final issue of the celebrated litigation, which seems as far off as ever. The Chollar

AT SAN FRANCISCO DURING 1896.

Name and Location of Company.	July.		August.		September.		October.		November.		December.	
	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.	H.	L.
Alpha, Nev	.17	.09	.12	.03	.20	.07	.18	.10	.15	.08	.12	.06
Alta, Nev	.12	.06	.14	.07	.15	.04	.15	.04	.12	.06	.08	.03
Andes, Nev	.36	.26	.35	.26	.35	.24	.31	.23	.25	.17	.20	.10
Belcher, Nev	.57	.32	.41	.23	.59	.29	.67	.49	.49	.29	.40	.27
Benton Cons., Nev	.25	.15	.25	.10	.20	.15	.15	.....	.15	.12	.15	.08
Best & Belcher, Nev	.94	.50	1.20	.60	1.15	.79	1.20	.83	.94	.79	.76	.49
Bodie, Cal	.68	.35	.70	.43	.75	.50	.65	.50	.65	.50	.58	.50
Bonanza, Nev	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Bullion, Nev	.24	.15	.21	.06	.21	.08	.23	.16	.18	.12	.17	.07
Bulwer, Cal	.45	.25	.44	.25	.50	.37	.48	.37	.46	.39	.43	.36
Caledonia, Nev	.16	.10	.15	.10	.16	.09	.16	.10	.12	.08	.12	.07
Challenge, Nev	.58	.30	.60	.40	.70	.39	.60	.45	.39	.26	.36	.24
Chollar, Nev	3.25	1.75	2.80	1.80	2.50	1.90	2.35	1.85	2.25	1.65	1.90	.68
Confidence, Nev	1.50	.73	1.80	1.20	1.50	1.20	1.10	.88	1.15	.78	.92	.60
Cons. Cal. & Va., Nev	2.15	1.55	2.00	1.60	2.00	1.55	2.60	1.60	1.85	1.65	1.60	1.00
Cons. Imp., Nev	.03	.01	.03	.01	.03	.01	.03	.02	.02	.01	.02	.01
Cons. New York, Nev	.05	.03	.05	.04	.05	.04	.04	.....	.04	.02	.04	.03
Crown Point, Nev	.55	.34	.40	.24	.53	.28	.57	.47	.42	.28	.29	.15
East Sierra Nevada, Nev	.05	.04	.05	.04	.05	.02	.04	.....	.04	.....	.04	.....
Eureka Cons., Nev	.25	.....	.25	.10	.40	.25	.30	.25	.30	.25	.25	.....
Eureka Cons. Drift, Cal	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Exchequer, Nev	.07	.04	.06	.02	.06	.03	.05	.03	.05	.02	.02	.01
Gould & Curry, Nev	1.20	.61	.76	.45	.65	.40	.91	.43	.71	.54	.55	.27
Hale & Norcross, Nev	1.50	1.05	1.50	1.05	1.60	1.30	1.55	1.25	1.35	1.05	1.10	.84
Jackson, Nev	.50	.25	.50	.25	.50	.25	.40	.25	.40	.25	.40	.25
Julia, Nev	.13	.08	.10	.06	.10	.04	.08	.04	.06	.04	.06	.05
Justice, Nev	.07	.03	.07	.04	.08	.04	.08	.04	.10	.01	.08	.04
Kentuck, Nev	.09	.01	.09	.04	.09	.05	.08	.05	.05	.02	.07	.01
Lady Washington, Nev	.06	.04	.05	.02	.04	.02	.04	.02	.04	.02	.03	.02
Mayflower Gravel, Cal	1.50	.....	1.25	.99	.80	.75	.80	.....	.80	.....	.....	.....
Mexican, Nev	.70	.50	.64	.46	.66	.45	.53	.45	.65	.49	.54	.35
Mono, Cal	.20	.04	.23	.15	.30	.15	.20	.15	.16	.15	.20	.15
Nevada Queen, Nev	.04	.03	.05	.03	.04	.03	.04	.03	.04	.03	.04	.....
North Gould & Curry, Nev	.08	.03	.10	.07	.....	.....	.09	.05	.....	.....	.....	.....
Occidental, Nev	1.20	.55	.73	.50	.65	.32	.54	.29	.39	.20	.21	.19
Ophir, Nev	1.35	.95	1.15	.89	1.15	.83	1.45	.84	1.35	1.15	1.20	.84
Overman, Nev	.25	.16	.23	.12	.20	.11	.19	.12	.12	.06	.07	.04
Pioneer, Cal	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Potosi, Nev	1.55	.90	1.35	.80	1.10	.68	.85	.60	1.05	.67	.88	.45
Savage, Nev	.92	.61	.80	.47	.69	.45	.79	.41	.66	.51	.54	.31
Scorpion, Nev	.08	.05	.07	.05	.10	.06	.10	.06	.08	.01	.04	.03
Seg. Belcher & Mides, Nev	.16	.10	.13	.06	.18	.06	.11	.05	.15	.03	.13	.09
Sierra Nevada, Nev	.92	.50	.67	.39	.83	.40	.74	.63	.69	.51	.51	.35
Silver Hill, Nev	.02	.01	.03	.01	.03	.01	.03	.01	.03	.01	.02	.01
Silver King, Ariz	.....	.....	.....	.....	.50	.15	.63	.15	.35	.15	.35	.30
Syndicate, Cal	.06	.02	.04	.02	.05	.02	.03	.02	.03	.02	.03	.02
Union Cons., Nev	.62	.40	.50	.32	.57	.33	.64	.51	.58	.45	.46	.31
Utah, Nev	.15	.07	.11	.06	.10	.04	.16	.06	.13	.06	.08	.04
Yellow Jacket, Nev	.52	.32	.42	.26	.42	.26	.60	.27	.50	.28	.36	.24

management was thus forced to carry on the work from this time forward with Potosi.

The North End mines operating on the Brunswick were not any more successful. With the exception of a little ore cut early in the year in a surface tunnel on the Gould & Curry location, nothing was found of value, although much was expected from this portion of the lode owing to some rich finds in surface workings many years ago. Best & Belcher was the favorite gamble at this end, and large quantities of the stock are held by people who hope that a strike on the Brunswick will enable the company to show at least one bar of bullion in return for the expenditure of many thousands of dollars before the company fades out of existence. Occidental, the main Brunswick location, received another black eye during the year owing to repeated failures to justify the hopes of investors, and the stock fell to bed-rock figures after working up at one time to $2.50. The ore, when found, defied all methods of reduction hitherto adopted, being of most unfortunately rebellious disposition.

All eyes again turned for relief to the Comstock, and here, it must be said, the prospects are anything but bright. Still, with the history of past revivals before them, dealers cheer themselves with the theory borne out by previous events that on Pine Street the darkest hour precedes the dawn. At the north end, Ophir and Consolidated California & Virginia are looked upon as the possible factors in the near future for another upheaval in prices. In other directions the prospects are gloomy, to say the least. Work on a very limited scale is now being carried on in the middle and south end properties. Savage, Chollar, and Crown Point, when the mines are open, produce a little bullion, not enough, however, to bar the necessity for levying assessments.

Unless some unexpected change takes place soon, the year 1897 will see the end of the speculative mining market in San Francisco. The business is virtually on its last legs now. The mines will then pass into the hands of inside syndicates to be worked on more business-like principles for low-grade ores, large reserves of which are available in all the leading properties. Worked on the system usually adopted by private individuals, there is no reason why these mines should not be running to-day on a highly remunerative business with expenses of operation cut down to the level of other mining companies. Wages are still kept up to the standard of $4 per day at Virginia City, which has handicapped the companies to a great extent, the Miners' Union taking the stand that extravagant salaries in the managerial department justify equally high rates for the working force. The stockholders are the only sufferers by this division of opinion.

A combination of all the active Bodie mines was effected during the year. Bulwer, Bodie, and Mono were absorbed by the Standard Consolidated Company, which in the future will represent the entire interests of the group. This is, of course, a subject for congratulation upon the part of the shareholders in the three companies which have now linked their fortunes with that of a prosperous concern which has managed to keep its head well above water for the past 20 years. The combination cuts off a great many sinecure positions in this city, the incumbents of which have fattened a salary list always well maintained by a never-failing assessment system. Standard Consolidated shares are now listed on the local stock exchange for the first time in the history of the company.

The Tuscarora mines have, with one or two exceptions, been weeded from the list of the exchange. The decline in silver has ended the mission of these companies for the time being, and the camp has faded out of the memory of speculators on the street. A few California gold mines in a position to pay dividends were added in their place during the year, but they failed to attract local investors, who find properties of this class too slow and tame a road to fortune.

The San Francisco Gold Mining Exchange was organized toward the close of 1895, and opened early in 1896 with the hope of reviving interest in mining stock speculation by putting on the market new properties in which the public could take an interest and feel some confidence. The idea of the exchange was to examine all properties before they were listed and dealt in; and in addition to serve as an agency in promoting the sale of properties and the organiza-

tion of new companies. For this purpose a staff of experts was engaged and the exchange prepared for business on a large scale. The idea was a good one; but for some reason the public did not respond. The number of properties submitted to the exchange was not large. Dealings on the call board were limited to so narrow a range that it seemed impossible to excite general interest, and the transactions continued to fall off. An attempt to enlarge them by introducing some of the old Comstock companies on the list did not have a fortunate result. The public still refused to come in, and the daily transactions dwindled to a point where they were no longer worth reporting. In spite of this the exchange was kept open during the year, but finally closed its doors early in 1897. This ended an experiment which possessed some elements of merit in its plans, and which failed mainly because of the suspicion with which working gold mining companies have come to regard the methods of the San Francisco stock operators.

British Columbia Mining Stock Markets in 1896.

The rapid development of mining in the Slocan, Kootenay, Trail Creek, and other districts in British Columbia, and the organization of a large number of companies to work the mines, led to the formation of mining stock markets at Rossland and Vancouver. At the former place especially a large number of brokers and promoters are found and regular quotations of shares are made.

While a number of the new British Columbia companies have a home ownership, probably a majority of them are held by parties in the United States. This is a natural result, as the country has been very largely prospected and opened by miners from across the boundary line.

The attempts made to place the mines in London have not met with much success as yet, but a great deal of interest has been developed in the eastern provinces of Canada. Investors from Ontario and Quebec have been ready purchasers of stocks and it is probable that they will be largely placed there. Montreal is becoming a center for these investments.

About the close of 1896 a mining stock exchange was organized at Spokane, in the State of Washington, which is now connected with Rossland by rail. On this exchange British Columbia stocks lead in the dealings, and a number of the companies operating in the province have their headquarters in Spokane.

The London Mining Stock Market in 1896.

The mining stock market in London during 1896 was characterized by one long-continued slump. The great boom of 1895 expended itself in the late autumn of that year, and the markets became duller and duller, and quotations fell lower and lower, until business was nearly at a standstill, with no prospect of any revival. These remarks apply in a most marked degree to South Africans. Of West Australians the same may be said, though it ought to be added that during the earlier months of the year they enjoyed a temporary boom. Other main features of the market were the attempt, only partially successful, to boom New Zealand gold mines, the advance of Indian gold shares in popularity, the turning of the attention of London promoters to

British Columbia as a profitable field for their enterprise, and the introduction of several large mining properties from the United States and Mexico by millionaire promoters.

The South African market during 1896 was influenced more by politics than by mining developments. The great anxiety at the beginning of the year and the continued uncertainty as to the treatment of foreigners in the Transvaal quite prevented any new enterprise in that State, and the loss of its grip on its territories by the Chartered Company made prospecting and development impossible in Rhodesia. The closing down of some Transvaal mines, such as Langlaagte Royal, and the rumors of further moves of like character, rumors which were not entirely groundless, caused some nervousness as to the future of gold mining in the Rand. The poor returns from such deep level properties as had commenced operations and the increased cost of mining under such conditions did not help to brighten the outlook. But perhaps the most potent factor of the slump was the inability of English and continental investors to absorb more stock. A limit to such capacity always exists, and when it was reached in this case in October, 1895, promoters, jobbers, and speculators alike became embarrassed by their holdings, failures occurred, and prices became depressed by forced sales. This course of things has been going on for quite a year and the end is not yet. Some promoters had the hardihood to commence new business, but all proved failures and some had to be wound up. Of the latter the most conspicuous was the Barnato Bank. In general terms it may be said that the quotations of South African stocks are now only one-half or one-third of what they were at the highest point of 1895, as far as speculative and promoting companies are concerned, such as Chartered, Consolidated Gold Fields, Rand mines, the Barnato group, etc. The fall in gold producers was not so marked, but the prices are still far higher than their real values.

The political troubles caused considerable interruption during the early months of the year, and the output of the Witwatersrand was reduced quite 25% for four months. On the outbreak of the troubles the native laborers fled and many mines were unable to supply sufficient ore to keep the mills going.

In the early months of the year the West Australian market had a temporary revival, owing chiefly to the brokers and jobbers leaving the South African market on the receipt of the bad news from the Transvaal. For a time the whole of the mining speculation was centered round the West Australian section, and prices were pushed up. New companies were brought out by the dozens, to provide work for the new influx of jobbers, but it is doubtful whether any great amount of selling actually occurred. The boom soon died away, because it had nothing substantial to sustain it. The failure of many companies to find the mines they were supposed to have obtained, and the inability to work on anything but rich ore, caused notable collapses. Then the lack of transport facilities, and the absence of official reports relating to the output, discouraged intending investors, so that at the present time the market is in a very weak condition. Backers of the market point to the richness of the ore and the large production at some of the mines, but as the ore may be picked, the results do not show the average yield of the ore won. No mine has as yet shown such good results as Bayley's Reward in its palmy days, but Great Boulder and

Hannan's Brownhill promise to become great mines. Dividends from companies working mines are as yet extremely scarce, and the only companies that do pay dividends in the West Australian market are the promoting companies, and the majority of such dividends are in shares and not cash.

During the summer there was a small boom in New Zealand gold mines and many new companies were organized. The public were, however, glutted with South Africans, which were slumpy, and West Australians, which did not prove good speculations, so that the boom never reached large proportions. As far as it went there was more genuine buying and more real gold mines brought before the public than in the West Australian boom. Perhaps the great distance of New Zealand from England was also a cause of the comparative failure of the boom.

The Indian mining market was, as usual, a restricted one, but during the days of general dullness some speculators commenced to handle the shares and to point out the regularity of the output of the established mines and the chances of success of some new flotations. The shares in Mysores and Champion Reefs were soon rushed up from £3 to £8. Though the market manipulations passed away, the shares retained their price and their good reputation.

The copper market was firm and rising throughout the whole year, and the shares of copper mining companies have risen in accordance. Rio Tinto, Cape Copper, Copiapo and others all rose considerably, the first named standing at a figure just double that ruling at the close of 1895. The scaling down of the debenture interest placed Rio Tinto in a much better financial position. In December another new financial arrangement was introduced—the splitting of each £10 share into two of £5 each, one to bear fixed preference interest and the other to be ordinary. In this way the shares are made available both for investment and speculation, so that their popularity is sure to be increased. The purchase of half the Anaconda stock, and its introduction on the London and Paris markets was an important feature of the year, but so far it has not formed a very strong market for itself. The floating of the Mountain Copper mines in Shasta County, Cal., by the Mathesons is the latest matter of interest in the copper market.

Of metallurgical matters of public interest, the most important during the year 1896 was the judgment in the Transvaal declaring the MacArthur-Forrest cyanide patents invalid. The release of the Rand companies from this bondage will have a favorable effect on gold-mining there; but very little use of it was made on the London market. No new process for treating gold ores of any other than minor importance was introduced during the year. Among new processes for the treatment of zinc-lead sulphides the most promising appeared to be that of the Burnham syndicate. This process is in operation at Swansea, and very large quantities of ore are being treated, but so far no results have been published. The Ashcroft process was sold to a limited company with large capital and under influential support. It is doubtful, however, whether the process, as described, will ever be a practical success.

The American mining market was during the past year confined between narrow limits, but there are signs of a revival on different lines from what we have hitherto been accustomed to. The companies which were formed 5 to 10

years ago have very nearly disappeared and their mines worked out; and their promoters have transferred their attention to districts which are more fashionable nowadays. On the other hand, several new mines have been introduced during the past year, but they are all large mines that have arrived at the dividend-paying period and possess immense reserves. Such mines have been bought for large sums in cash and sold on the market for still larger sums, but obviously it is only promoters of the most wealthy sort that can carry out such business. Among American mines of this character which were introduced here are the Anaconda, the Minas Prietas (renamed Grand Central), and the Mountain Mines, Shasta County, Cal. Quite a number of other such mines are in treaty for and within the next six months their presence in the London market will exercise an important influence. Cripple Creek was a failure in London. No influential promoters handled any of the mines, and it was left to others to attempt to introduce comparatively worthless properties. It is doubtful if any of the flotations were a success.

All during the year there was an expectation of a boom in British Columbian mines, but owing chiefly to promoters and jobbers being filled up with other stock the boom did not arrive. In the summer months several promoting companies and a number of companies to work river gravel and placers in Cariboo were brought before the notice of the public, but they met with a cool reception. No companies were floated to work the mines of Kootenay, though most promoters have acquired options on properties there and are only waiting for a little encouragement to bring them forward.

Other districts in Canada that are receiving attention are the Lake of the Woods and Rat Portage gold mining districts. Several companies were floated to work mines there and some were taken up privately. It is said that some very influential promoters have interested themselves there.

The companies which used to form the backbone of the American mining market have mostly got into low water. Richmond Consolidated has transferred its attention to West Australia. Jay Hawk has abandoned its Montana property and acquired an interest in New Zealand. Poorman, after being reconstructed last year under new management, has again come to the end of its money and has been again reconstructed. The hope for the future is very small. The repeated warnings of the *Engineering and Mining Journal* concerning this property have been fully confirmed and justified by the facts brought out during the past few years. Of Springdale nothing is now heard except demands by its shareholders for an investigation into its management. Golden Leaf has gone to West Australia and shipped its machinery thither. Golden Feather and Golden Gate have made changes, but their position does not seem to be much improved. Palmarejo is still at law with the Mexican Mineral Railway, and its production is practically at a standstill. The Montana Company did not maintain its advance of last year, as the new strike did not prove a permanent body of ore. Twin Lake Placers pays now a much smaller dividend than formerly, as its gravels yield a very small return. New Guston works without profit.

Of miscellaneous new American companies introduced to the British public during the year, the only one of any importance was the Prize Gold Mines,

which formed part of the Hearst estate, and was promoted in London by Irwin Stump. Another was the Gold Fields, of Mexico, formed to take over the Hurnapa properties from the Palmarejo company. Finally, the stock of Playa de Oro (Ecuador) was introduced here and a considerable amount sold.

PRICES OF MINING STOCKS AT LONDON DURING 1896.

Company.	Location.	Product.	Par Value.	Opening.	Highest.	Lowest	Closing
			£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
North Americans:							
Alaska-Mexican	Alaska	Gold	1 0 0	1 10 0	2 0 0	1 0 0	1 7 6
Alaska-Treadwell	Alaska	Gold	5 0 0	5 5 0	5 15 0	4 15 0	4 15 0
De Lamar	Idaho	Gold and silver	1 0 0	18 0	1 1 6	11 0	11 0
Hall Mines	British Columbia	Copper and silver	1 0 0	2 15 0	3 5 0	1 3 9	1 7 6
Montana	Montana	Gold and silver	1 0 0	7 3	11 0	2 6	3 9
Palmarejo	Mexico	Gold and silver	1 0 0	1 0	3 6	6	1 0
Pinos Altos	Mexico	Gold and silver	1 0 0	2 6	5 0	1 3	1 3
Plumas-Eureka	California	Gold	2 0 0	12 6	16 3	2 6	2 6
Richmond	Nevada	Gold, silver, lead	5 0 0	12 6	1 10 0	5 0	12 6
Sierra Buttes	California	Gold	2 0 0	8 9	11 3	2 6	2 6
Springdale	Colorado	Gold	4 0	1 3	1 9	3	3
South Americans:							
Colombia Hydraulic	Colombia	Gold	1 0 0	8 9	12 6	6 3	6 3
Frontino & Bolivia	Colombia	Gold	1 0 0	1 1 3	1 12 6	17 6	1 7 6
St. John del Rey	Brazil	Gold	1 0 0	18 9	1 3 6	15 0	12 6
Copper:							
Anaconda	Montana	Copper and silver	5 0 0	6 0 0	7 15 0	5 12 6	5 15 0
Cape Copper	South Africa	Copper	2 0 0	1 17 6	2 17 6	1 17 6	2 10 0
Copiapo	Chile	Copper	2 0 0	1 15 0	3 0 0	1 13 9	1 13 9
Mason & Barry	Portugal	Copper & sulphur	4 0 0	2 10 0	3 10 0	2 7 6	3 2 6
Rio Tinto	Spain	Copper	10 0 0	14 12 6	25 6 3	14 12 6	24 8 9
Tharsis	Spain	Copper & sulphur	2 0 0	4 10 0	6 2 6	4 5 0	5 17 6
Australians:							
Bayley's Reward	West Australia	Gold	1 0 0	4 6	7 6	1 6	1 6
Broken Hill Proprietary	New South Wales	Silver	1 0 0	2 11 3	3 2 6	2 1 3	2 10 0
Mount Morgan	Queensland	Gold	1 0 0	2 17 6	4 0 0	2 16 3	3 3 9
South Africans:							
British South Africa	South Africa	Lands and explo.	1 0 0	3 5 0	5 11 3	2 1 3	2 1 3
City and Suburban	Transvaal	Gold	4 0 0	4 0 0	5 10 0	3 2 6	3 2 6
Crown Reef	Transvaal	Gold	1 0 0	9 10 0	12 7 6	9 0 0	9 0 0
De Beers Cons	Cape Colony	Diamonds	5 0 0	20 17 6	31 11 3	20 0 0	28 7 6
Ferreira	Transvaal	Gold	1 0 0	14 10 0	22 0 0	14 0 0	18 0 0
Geldenhuis Estate	Transvaal	Gold	1 0 0	3 7 6	4 15 0	2 10 0	2 10 0
Henry Nourse	Witwatersrand	Gold	1 0 0	5 17 6	7 15 0	4 10 0	5 17 6
Heriots (New)	Witwatersrand	Gold	1 0 0	9 7 6	10 2 6	7 0 0	7 0 0
Jagersfontein	Orange F. S.	Diamonds	5 0 0	8 10 0	12 10 0	7 10 0	10 2 6
Langlaagte Estate	Transvaal	Gold	1 0 0	4 15 0	6 11 3	3 17 6	3 17 6
Primrose (New)	Transvaal	Gold	1 0 0	5 0 0	6 15 0	3 15 0	3 15 0
Robinson	Transvaal	Gold	5 0 0	8 12 6	10 7 6	7 15 0	7 15 0
Sheba	Transvaal	Gold	1 0 0	1 12 6	2 6 3	1 5 0	1 18 9
Simmer & Jack (New)	Transvaal	Gold	5 0 0	4 10 0	7 5 0	3 17 6	3 17 6
Wemmer	Witwatersrand	Gold	1 0 0	10 0 0	11 12 6	6 5 0	6 5 0

The state of the Cornish tin industry continued to cause much alarm among those who are interested in its welfare. Strong exertions were made to rouse the owners of the land and mines and also the miners to the fact that their methods of mining are obsolete, and that the cost-book system prevents the subscription of further capital. The management of Dolcoath, under the new limited company, proved successful, and already several other mines have followed its example. But the regeneration has not yet been sufficiently widespread to work much improvement in Cornish mining. The chief obstacle to the change is the exacting disposition of the "lords" or owners of the fee, who seem to look more for immediate profits than much greater future benefits.

The Paris Mining Stock Market in 1896.

By Azote.

The year 1896 was a better one than its predecessor, for in it we saw the fair beginning of a new period of active trade, though in some directions

speculation has suffered. It has been speculation only, however, and real trade has increased largely.

In 1895 we saw the height of the South African fever. For two years our people had been investing in Transvaal gold stocks, and believing all the wild stories which had been set afloat in London and Johannesburg. These stocks had risen to great prices, and we had come to believe that there was no limit to the gold prodction of the Transvaal. In a few years it had risen from nothing to more than 200,000,000f. a year; why should it not be 300,000,000f. in another year, then 500,000,000f., even a milliard in 1900?

The speculation had passed the extreme point, and the decline had begun when the attempt of Rhodes and Jameson to seize the Transvaal and the "revolt" of the Uitlanders came. What the result has been we all know. The political disturbances served for some months as an excuse; but they passed over, labor was reorganized, the mines were all at work, and still production did not increase. At last those who think began to see that perhaps it could not increase—that perhaps the best had been done and that the great growth promised was a promise only, never to be realized. As this belief gained ground people began to sell their stocks when they could. Month after month prices dropped, and there would have been a real crash but for the caution of certain leaders, who did their best to support the market and check selling whenever they could. Thanks to this and to the conservatism of our holders, there was no sharp crisis.

It was a gradual descent, not an avalanche.

This Kaffir affair, however, though it was in a measure a discouragement, did not prevent the rest of the market from doing well. The strongest section all through the year was found in the metallurgical shares. And with reason, for these companies have done and are doing better than for a long time. For several years there have been complaints of low prices and of the competition of the Belgian and the German iron masters. In 1896 the complaint was of delay in filling orders, of the difficulty of getting raw material fast enough. The blast furnaces and the forges could not turn out their products in quantity sufficient. The rise was marked; Creusot, which opened at 1650f. went to 2000f.; Firminy from 1500f. to 1723f.; Fives-Lille from 640f. to 800f.; Longwy from 600f. to 890f.; Acieries de la Marine from 821f. to 1025f. And in nearly every case the close was at the highest price.

The coal shares naturally gained also. Thus, for instance, Anzin, which opened at 4125f. closed at 4500f.; Bruay rose from 18,800f. to 23,500f.; Courrieres from 4200f. to 4589f.

Copper, like iron, was in great demand by the workers, and the price was well maintained through the year, in spite of the great quantities which American mines sent to our markets. The copper shares were consequently strong, and Rio Tinto went up from 390f. to 630f. Boleo opening at 1170f., closed at 1380f.; Tharsis rose from 117f. to 155f.

The stocks of the lead and zinc companies also improved very much. Malfidano rose from 799f. to 1018f., and Laurium from 588f. to 670f. Vieille Montagne went up from 431f. to 660f., but dropped to 505f. at the close. Huanchaca suffered less from misfortune than in 1895, and showed at the close

a price of 85f. against 75f. at the opening. Nickel varied considerably, going up to 200f. at one time and down to 135f. at another; it closed at 169f. Penarroya, in spite of general depression in Spain, closed at 1640f., 540f. above the opening price.

The Russian coal and iron stocks were especially in favor and have shown very good quotations. Dumbrowa, Donetz, Briansk, Huta-Bankowa, all showed gains and were bought largely for investment. Probably the number of these stocks will be increased in 1897 by those of several gold-mining companies.

American stocks have not been in favor here for several years. Only two showed many dealings in 1896. Fraser River, which opened at 31f., went at one time to 53f., but dropped again to 32f. at the close. Rebecca, a Cripple Creek stock, opened at 27f.; it reached the highest at 34f., but closed low at 7.50f.

PRICES OF MINING STOCKS AT PARIS DURING 1896.

Name of Company.	Country.	Product.	Par Value	Opening.	Highest	Lowest	Closing
			Fr.	Fr.	Fr.	Fr.	Fr.
Acieries de Creusot	France	Steel mfrs	2,000	1,650.00	2,000.00	1,650.00	1,915.00
Acieries de Firminy	France	Steel mfrs	500	1,500.00	1,723.00	1,430.00	1,700.00
Acieries de Fives-Lille	France	Steel mfrs	500	640.00	800.00	640.00	794.00
Acieries de la Marine	France	Steel mfrs	500	821.25	1,025.00	821.25	1,025.00
Acieries de Longwy	France	Steel mfrs	500	600.00	890.00	600.00	890.00
Anzin	France	Coal	.....	4,175.00	4,720.00	4,175.00	4,500.00
Boleo	Lower California	Copper	500	1,170.00	1,575.00	1,160.00	1,380.00
Briansk	Russia	Coal and iron	500	1,390.00	1,400.00	1,290.00	1,375.00
Bruay	France	Coal	400	18,800.00	23,900.00	18,800.00	23,500.00
Callao	Venezuela	Gold	125	10.50	10.50	4.00	4.00
Champ d'Or	South Africa	Gold	25	65.00	65.00	31.00	33.00
Courrieres	France	Coal	300	4,200.00	4,665.00	4,200.00	4,589.00
De Beers Cons	South Africa	Diamonds	125	635.00	790.00	555.00	735.00
Dombrowa	Russia	Coal	500	535.00	640.00	534.00	540.00
Dynamite Centrale	France	Explosives	500	485.00	550.00	448.00	500.00
Fraser River	British Columbia	Gold	25	31.00	53.00	30.00	32.00
Huanchaca	Bolivia	Silver	125	75.00	90.00	61.25	85.50
Huta-Bankowa	Russia	Iron and steel	500	2,640.00	2,930.00	2,625.00	2,923.75
Laurium	Greece	Zinc and lead	500	587.50	670.00	555.00	670.00
Malfidano	Italy	Zinc	500	799.00	1,070.00	730.00	1,018.00
Metaux, Cie. Française des	France	Metal dealers	500	355.00	557.50	355.00	546.00
Mokta-el-Hadid	Algeria	Iron	500	705.00	780.00	690.50	756.00
Nickel	New Caledonia	Nickel	500	187.00	200.00	135.00	169.00
Penarroya	Spain	Coal, etc	500	1,100.00	1,640.00	1,100.00	1,640.00
Rebecca	Colorado, U.S	Gold	25	27.00	34.50	7.50	7.50
Rio Tinto	Spain	Copper	250	390.00	655.00	372.50	627.50
Saint Elie	French Guiana	Gold	25	23.00	23.00	18.25	18.50
Salines de l'Est	France	Salt	500	260.00	350.00	260.00	295.00
Sels Gemme de la Russie Merid	Russia	Salt, etc	500	690.00	699.00	605.00	620.00
Tharsis	Spain	Copper	50	117.00	155.00	113.00	154.50
Vieille Montagne	Belgium	Zinc	80	431.25	660.00	425.00	505.00

Upon the whole the market—outside of the Transvaal—was strong and the speculators for the rise had the best of it nearly all through the year. In November and December a growing scarcity of money, causing higher rates for loans and for carrying stocks, did discourage speculation a little, but had no marked effect on quotations. Some exports of gold were made, and for the first time since 1890 the gold reserve of the Bank of France showed decreases. This movement is at an end; but the demand for money in trade and in manufacturing is so large that we may say we have seen the end of 2% loans for a time.

Of questions a little outside the stock market, much attention was given to the renewal of the charter of the Bank of France, to the proposed changes in taxation, and to acquisition of Madagascar.

There is a movement on foot to reorganize our Bourse and to give the *coulissiers* some of the official standing which now belongs to the *agents de change* only. It may succeed after some delay.

Under the present system only the dealings on the *banquette* and through the *agents de change* are official. The *coulisse* is purely an outside market, though it makes its own rules and is strictly governed by them. In it there are dealings in many stocks which are not on the regular lists, and the brokers are compelled by general opinion to conduct themselves in accordance with the code. Many are opposed to the changes which have been discussed, believing that there are advantages in the present system which would be lost if it were modified. At present the *coulisse* offers a free market for many stocks which would be excluded were the somewhat narrow regulations obtaining in the official exchange applied to all dealings. Certainly the field of operations would be restricted. It may well be questioned whether the greater security given by official regulation and supervision would make up for the loss of elasticity which would result from the changes proposed.

Financially the year showed few movements of importance. Austria has continued to accumulate gold in preparation for her change in currency, but the operation is not yet fully completed and all we can do is to wonder at its slow progress. Russia has begun preparations for the resumption of specie payments, and will apparently carry them into execution before long. Under the plan proposed a new standard is adopted and a new gold rouble, equivalent in value to the present paper rouble will be introduced. The new coins will be worth about two-thirds of the old rouble. Russia has an enormous stock of gold, the accumulation having been continued for several years. She is moreover a gold mining and producing country, and the output of her mines is added to the balance of coin in reserve. Her financial position is strong.

Political matters have affected the market by causing several alarms and by diverting speculation to international securities. The Eastern question is always with us, but especial attention was called to it this year by the Armenian slaughters, the Cretan and Macedonian troubles. There is some prospect now that the *entente* between France and Russia may help to settle this question.

Egypt has been a vexation rather than an alarm.

The Chinese question is settled for the present by the controlling influence of Russia, and France may derive some advantage therefrom in trade.

The Italian defeat in Abyssinia caused some trouble. The weakest case politically was in Spanish securities. The inability of Spain to control her revolted colonies and her fast-approaching bankruptcy were matters of anxiety.

Upon the whole 1896 was a good year for us and left us with still better prospects for 1897.

Dividends and Assessments of American Mining Companies.

We give in the following pages tables showing the dividends paid and assessments levied by American mining companies for a series of years ending with 1896. The tables are as complete as they can be made, but they are necessarily imperfect, because many companies do not make reports, and the facts cannot be ascertained by inquiry. There are many mines also which are

carried on by private partnerships, and if the profits of these could be given the amount of dividends shown would be very largely increased. With regard to assessments, a few States require the publication of notices by all incorporated companies, but in others no such regulation exists. In many cases companies have willingly aided us and have given all necessary information; if all would do this our work would be very much lightened and our tables made full and complete. The list of dividend payers has increased during the year and promises to show a continued growth. In the assessment list the Comstock companies are still most prominent, and continue to call regularly on their stockholders.

DIVIDENDS PAID BY AMERICAN MINES. ($1 = $1000; total, full amount.)

Company.	1887	1888	1889	1890	1891	1892	1893	1894	1895	1896	Totals.
Adams, S., L., Colo.	15	....	....	....	75	8	....	....	6	....	$693,500
Ætna Cons., Q., Cal.	....	....	....	....	....	....	....	20	20	40	80,000
Alaska-Mexican, G., Alaska	....	....	....	....	....	....	....	26	80	70	173,031
Alaska-Treadwell, G., Alaska (*a*)	....	....	....	....	450	300	375	375	400	350	3,025,000
Alice, S., G., Mont.	....	25	25	95	75	....	....	....	....	....	975,000
Alma, G., Idaho	....	....	15	....	....	....	....	....	....	....	60,000
Alturas, G., Idaho	....	113	....	....	....	....	....	....	....	....	262,250
Amador Gold, Ltd., G., Cal.	....	....	....	31	....	....	....	....	....	....	31,250
American & Nettie, G., Colo.	....	....	150	....	45	30	....	....	....	....	225,000
American Belle, S., C., Colo.	....	....	....	....	50	....	....	....	....	....	50,000
American Coal, Md.	....	....	....	....	90	90	90	90	....	....	360,000
American Dev. & Mg., G., S., C., Mont.	....	....	....	....	....	....	....	52	24	....	76,422
American Turquoise, N. M.	....	....	....	....	....	....	60	....	....	....	60,000
Amethyst, Colo.	....	....	....	....	....	....	....	....	36	....	36,000
Amy & Silversmith, Mont.	43	....	....	....	....	....	....	....	....	....	247,530
Anaconda, G., Colo.	....	....	....	....	....	13	....	....	....	....	12,500
Anaconda Copper, Mont.	....	....	....	....	....	....	....	....	....	2250	2,250,000
Anchoria-Leland, Colo.	....	....	....	....	....	....	....	....	....	30	30,000
Argentum-Juniata, S., Colo.	....	....	....	....	....	....	....	....	39	....	39,000
Argyle, G., Colo.	....	....	....	....	....	20	....	....	....	....	20,000
Aspen, S., L., Colo.	....	80	280	200	100	100	100	40	....	....	900,000
Atlantic, C., Mich.	40	120	80	100	40	....	....	....	....	....	700,000
Aurora, I., Mich.	150	....	....	....	200	100	200	....	....	50	700,000
Badger, S., Can.	....	....	....	38	....	....	....	....	....	....	37,500
Bald Butte, G., Mont.	....	....	....	20	30	20	100	200	128	32	470,000
Ballarat Smuggler, G., Colo.	....	....	....	....	6	....	....	....	....	....	6,000
Bangkok-Cora Belle, S., Colo.	....	....	3	42	....	....	....	....	54	6	107,510
Bannister, S., Mont.	....	....	....	24	72	6	....	....	....	....	102,000
Bassick, S., G., Colo.	....	....	....	....	....	....	....	....	....	....	400,000
Bates-Hunter, G., Colo.	....	....	....	....	68	....	....	....	....	....	67,500
Belden, F. E., M., N. H.	....	....	....	....	....	45	60	60	48	4	217,000
Bellevue, S., L., Idaho	38	....	....	....	13	....	....	....	....	....	200,000
Best Friend, S., Colo.	....	....	....	....	70	20	....	....	....	....	90,000
Big Bend, G., Cal.	48	....	....	....	....	....	....	....	....	....	258,000
Big Six, G., S., Colo.	....	....	....	....	....	....	....	....	....	2	2,500
Bimetallic, S., G., Mont.	....	....	....	290	840	200	190	....	....	....	1,630,000
Bodie Cons., G., Cal	....	....	....	....	....	....	....	75	....	....	1,677,572
Bonanza King, S., Cal.	....	....	....	....	....	....	....	....	....	....	185,000
Boreel, S., Colo.	....	....	....	....	....	....	60	23	....	....	105,000
Boston & Montana, C., S., Mont.	....	400	500	625	500	....	....	275	1050	1500	4,925,000
Brooklyn Lead, L., S., Utah	25	....	....	....	....	....	....	....	....	....	127,000
Brotherton, I., Mich.	....	....	....	....	....	40	80	....	....	....	120,000
Bull-Domingo, L., S., Colo.	....	....	....	20	29	4	....	....	....	....	53,000
Bullion, Beck & Champion, Utah.	....	....	....	....	....	....	....	425	325	290	1,947,000
Bulwer Cons., G., Cal.	....	....	....	....	....	15	....	....	....	....	190,000
Bunker Hill & Sullivan, S., L., Idaho	....	20	....	....	....	....	....	....	....	....	150,000
Buxton, So. Dak.	5	....	....	....	....	20	....	....	....	....	25,000
Caledonia, G., Dak.	....	16	80	56	....	56	....	....	....	....	192,000
California, G., Colo.	....	....	....	20	....	....	....	....	....	....	116,500
Calliope, S., Colo.	....	....	50	85	5	....	....	....	....	....	140,000
Calumet & Hecla, C., Mich.	1000	2000	2000	2000	2000	2000	2000	1500	2000	2500	46,850,000
Carbonate Hill, S., L., Colo.	....	....	....	....	....	....	....	....	....	....	80,000
Cariboo, B. C.	....	....	....	....	....	....	....	....	49	76	125,410
Carlisle, G., N. M. (*b*)	....	175	....	....	....	....	....	....	....	....	175,000
Catalpa, S., L., Colo.	....	....	....	....	....	....	....	....	....	....	270,000
Centennial-Eureka, S., G., L., Utah.	....	....	....	150	330	90	188	195	510	390	1,860,000
Central, C., Mich.	40	70	40	20	20	....	....	....	....	....	1,970,000
Champion, G., Cal.	....	....	....	27	43	41	41	41	27	....	219,000
Charleston, P., S. C.	....	....	....	....	....	....	140	....	....	....	140,000
Chrysolite, S., L., Colo.	....	....	....	....	....	....	....	....	....	....	1,650,000
Church, G., Cal.	....	....	....	....	....	....	....	....	5	....	5,000
Clay County, G., Colo.	....	....	....	8	48	....	....	....	....	....	56,000

DIVIDENDS PAID BY AMERICAN MINES—*Continued.*

Company.	1887	1888	1889	1890	1891	1892	1893	1894	1895	1896	Totals.
Cleopatra, G., S., So. Dak	....	....	....	....	....	....	450	....	....	....	$450,000
C. O. D., G., Colo	....	....	....	....	....	....	....	....	20	5	25,000
Cœur d'Alêne Silver-Lead Mg. Co., S., L., Idaho	....	....	70	160	80	72	30	....	....	....	340,000
Colorado Central, S., Colo	83	83	55	....	14	55	28	....	....	....	502,661
Colorado Fuel & Iron, Colo	....	....	178	189	252	67	67	....	....	....	752,700
Commonwealth, S., Nev	....	....	....	20	....	....	....	....	....	....	20,000
Confidence, S., Nev	....	175	25	....	....	....	....	....	....	....	277,680
Cons. Cal. & Va., S. G., Nev. (*c*)	1118	1118	756	162	216	....	....	108	108	....	3,898,800
Cons. New York, S., G., Nev	....	....	....	....	....	....	10	....	....	....	10,000
Consolidation Coal, Md	....	....	....	....	....	205	....	....	....	....	205,000
Contention, S., Ariz	....	....	....	....	....	50	....	....	....	....	2,637,500
Cook's Peak, S., L., Colo	....	....	....	....	55	60	....	....	....	....	119,532
Copper Bell, S., Mont	....	....	....	....	14	....	....	....	....	....	13,500
Copper Queen, C., S., Ariz	....	140	70	210	....	140	300	200	150	....	1,910,000
Coptis, S., Nev. (*d*)	....	....	....	....	....	....	....	....	1	....	77,000
Coronas, S., G., Mex	....	....	....	....	....	....	....	....	....	5	5,000
Cortez, S., Nev	....	....	....	173	250	95	45	....	....	....	735,000
Cosmopolitan, S., Utah	....	....	....	....	....	....	....	....	....	....	75,000
Crescent, S., L., Utah	18	....	....	....	....	....	....	....	....	....	238,000
Dalton & Lark, G., S., L., Utah	....	....	....	....	....	....	....	....	....	88	87,500
Daly, S., L., Utah	375	488	450	450	450	450	188	....	....	37	2,887,500
Deadwood-Terra, G., So. Dak. (*e*)	100	....	....	....	50	100	....	....	....	100	1,240,000
Deer Creek, S., G., Idaho	....	10	10	....	....	....	....	....	....	....	20,000
De Lamar, S., G., Idaho	....	....	....	....	150	200	450	500	450	500	2,250,000
Della, S., Colo	....	....	....	....	....	....	....	50	....	....	50,000
Derbec, G., Cal	20	....	30	30	20	....	....	....	....	....	280,000
Dexter, S., Nev	....	....	....	....	....	....	100	....	....	....	100,000
Dunkin, S., L., Colo	30	100	40	....	....	....	....	....	....	....	390,000
Dunstone, G., S., L., Mont	....	6	....	....	....	....	....	....	....	....	6,000
Eclipse, L., S., Colo	20	....	....	....	....	....	....	....	....	....	20,000
Elkhorn, S., L., Mont	20	....	....	125	300	303	225	142	50	....	1,265,000
Elkton Cons., G., Colo	....	....	....	....	....	....	....	60	....	90	166,960
Empire, G., Mont. (*f*)	71	....	....	....	....	....	....	....	....	....	70,500
Enterprise, S., G., Colo. (*g*)	....	....	....	....	250	450	125	....	....	....	825,000
Eureka Cons., S., L., Nev	50	88	....	38	50	13	....	....	....	....	5,112,500
Evening Star, S., L., Colo	....	13	25	....	....	....	....	....	....	....	1,437,500
Father de Smet, G., So. Dak	....	....	....	....	....	....	....	....	....	....	1,125,000
Florence, S., Mont	....	....	....	....	....	....	....	....	35	54	89,348
Forepaugh, Colo	....	....	....	....	....	....	....	....	16	....	16,000
Franklin, C., Mich	40	160	80	80	80	160	120	80	....	....	1,240,000
Freeland, S., Colo	....	....	....	....	....	....	....	....	....	....	190,000
Galena, S., L., G., Utah	....	....	....	....	....	....	....	....	6	49	61,000
Garfield, S., G., Nev	13	25	....	....	....	....	....	....	....	....	85,000
Garfield Grouse, G., Colo	....	....	....	....	....	....	....	....	....	24	24,000
Glengarry, S., G., Mont	....	....	....	....	10	....	....	....	....	....	10,000
Golconda, S., G., Idaho	....	120	....	....	....	....	....	....	....	....	120,000
Gold Coin, G., Colo	....	....	....	....	....	....	....	....	15	85	100,000
Golden Eagle, G., Colo	....	....	....	....	....	....	....	....	....	10	10,000
Golden Fleece, Colo	....	....	....	....	....	....	....	....	192	162	563,179
Golden Reward, G., So. Dak	....	20	....	....	....	60	60	....	....	....	125,000
Gold & Globe, G., Colo	....	....	....	....	....	....	....	....	11	25	36,000
Gold Rock, G., Colo	....	....	....	....	29	....	....	....	....	....	28,750
Granby, Z., Mo	....	....	20	....	....	....	....	....	....	....	20,000
Granite, S., L., Idaho	....	....	8	20	....	....	....	....	....	....	83,400
Granite Mountain, S., G., Mont	2000	1600	2400	2400	1400	520	....	....	....	....	12,120,000
Great Western, Q., Cal	....	....	....	....	25	137	25	....	....	....	388,366
Hale & Norcross, S., G., Nev	....	224	....	....	....	....	....	....	....	....	1,822,000
Harquahala, Ariz	....	....	....	....	....	....	....	72	....	....	126,000
Hecla Cons., S., L., Mont	30	100	180	123	180	90	60	120	90	45	2,145,000
Helena & Frisco, L., S., Idaho	....	....	....	79	190	20	....	15	10	50	475,000
Helena M. & R. Co., Mont	....	....	....	....	....	....	....	....	....	....	197,970
Helena & Victor, S., Mont	....	....	....	....	20	....	....	....	....	....	70,000
Highland, G., So. Dak	....	....	....	....	....	....	....	....	50	140	3,224,918
Holmes, S., Nev	....	....	....	....	....	25	....	....	....	....	75,000
Homestake, G., So. Dak	300	300	188	150	150	150	150	256	344	375	6,087,500
Honorine, S., Utah	38	....	....	....	....	....	....	....	....	....	125,000
Hope, S., Mont	25	50	....	....	....	100	175	75	10	60	652,252
Horn Silver, S., L., Utah	....	....	50	200	200	200	230	150	50	50	5,130,000
Hubert, G., Colo	....	18	5	....	....	....	....	....	....	....	247,000
Idaho, G., Cal	140	357	178	29	93	42	105	....	....	....	5,489,000
Ideal, S., L., Colo	....	....	....	....	....	....	....	....	....	....	15,000
Illinois, S., N. M	25	....	20	20	....	....	....	....	....	....	65,000
Iowa, G., S., L., Colo	....	....	....	....	....	....	....	....	....	50	60,000
Iron Hill, So. Dak	44	....	....	....	....	....	....	....	....	....	156,250
Iron Mountain, S., Mont	....	....	....	50	25	....	30	50	105	82	492,500
Iron Silver, S., L., Colo	300	300	100	....	....	....	....	....	....	....	2,500,000
Isabella, G., Colo	....	....	....	....	....	....	....	....	23	180	202,500
Ivanhoe, Colo	....	....	10	....	....	....	....	....	....	....	10,000
Jackson, G., S., Nev	....	....	5	....	5	....	....	....	2	....	80,000
Jay Gould, G., S., Mont	95	226	74	22	....	....	....	....	....	....	459,000
Jay Hawk, S., G., Mont. (*h*)	....	....	....	....	....	33	....	....	....	....	33,375
Jumbo, G., Colo	33	....	....	....	....	....	....	....	....	....	33,000
Kearsarge, C., Mich	....	....	....	80	....	....	....	....	40	....	120,000
Kennedy, G., Cal	....	....	....	....	360	500	480	540	184	....	1,796,000

DIVIDENDS PAID BY AMERICAN MINES—*Continued.*

Company.	1887	1888	1889	1890	1891	1892	1893	1894	1895	1896	Totals.
Kentuck, S., G., Nev.	....	....	....	....	....	....	....	....	....	....	$1,350,000
Lady Franklin, N. M.	100	....	....	....	....	....	....	....	....	....	100,000
Lake Superior, I., Mich.	....	....	....	....	....	400	....	....	84	....	484,000
Last Chance, G., B. C.	....	....	....	....	....	....	....	....	....	20	20,000
Last Chance, S., Colo.	....	....	....	....	....	650	....	....	....	....	650,000
Leadville, S., L., Colo.	20	....	....	....	....	12	12	....	....	....	316,000
Le Roi, G., B. C.	....	....	....	....	....	....	....	....	25	225	250,000
Lexington, G., Colo.	....	....	....	....	....	36	....	....	....	....	36,000
Lexington, S., Mont. (*i*).	....	....	64	....	....	....	....	....	....	....	609,000
Little Chief, S., L., Colo.	....	....	....	20	....	....	....	....	....	....	820,000
Little Rule, S., Colo.	....	....	....	100	120	....	....	....	....	....	220,000
Maid of Erin, S., L., C., Colo. (*j*).	....	....	....	....	....	....	....	....	60	....	740,000
Mammoth, Utah.	20	50	130	470	320	....	....	....	....	60	1,150,000
Manhattan, S., Nev.	....	....	....	....	....	....	....	....	....	....	437,500
Martin White, S., Nev.	....	....	....	....	....	....	....	....	....	....	140,000
Mary Murphy, S., Colo.	70	....	....	....	....	....	....	....	....	....	175,000
Maxfield, S., L., Utah.	....	....	....	....	36	18	....	....	....	....	117,000
Mayflower Gravel, G., Cal.	....	....	....	....	....	....	....	....	....	....	166,897
May-Mazeppa, S., L., Colo.	....	....	....	70	110	....	....	....	....	....	180,000
Mercur, G., Utah.	....	....	....	....	....	....	50	150	175	200	575,000
Metropolitan, I., Mich.	....	....	....	....	....	....	....	....	....	....	3,912,500
Minas Prietas, S., Mex.	....	....	....	50	....	....	....	....	....	....	50,000
Minnesota, I., Minn.	....	....	....	....	840	840	495	....	....	495	3,240,000
Mollie Gibson, S., Colo.	....	....	....	....	1000	1700	1230	100	50	....	4,080,000
Monitor, So. Dak.	....	....	38	7	....	....	....	....	....	....	45,000
Mono, G., Cal.	....	....	....	....	....	....	....	....	....	....	12,500
Montana Limited, G., S., Mont.	719	413	206	178	83	....	....	....	205	....	2,890,000
Montana Ore Purchasing, Mont.	....	....	....	....	....	....	....	....	160	320	480,000
Moon Anchor, G., Colo.	....	....	....	....	....	....	....	....	....	24	24,000
Moose, Colo.	....	....	....	....	....	....	....	108	72	6	186,000
Morning Star, S., L., Colo.	75	....	....	....	50	....	....	....	....	....	1,025,000
Morning Star Drift, G., Cal.	....	....	....	....	23	83	72	106	154	132	464,400
Moulton, S., Mont.	60	....	....	....	....	30	....	20	30	....	460,000
Mount Diablo, S., Nev.	....	40	40	20	30	....	30	....	....	....	225,000
Mount McClellan, S., Colo.	....	....	....	....	22	....	....	....	....	....	21,936
Mount Pleasant, Cal.	45	....	....	....	....	....	....	....	....	....	180,000
Mount Rosa, Colo.	....	....	....	....	....	....	....	5	5	20	30,000
Napa, Q., Cal.	....	....	30	40	40	70	70	50	80	70	810,000
Navajo, S., G., Nev.	....	....	40	....	....	....	....	....	....	....	226,111
New Elkhorn, Colo.	....	....	....	....	....	....	....	....	....	72	72,000
New Guston, S., Colo.	....	100	188	170	440	124	....	....	....	....	1,198,120
New Hoover Hill, G., N. C.	....	....	....	....	....	....	....	....	....	....	37,200
Newton, Cal.	....	....	....	....	10	....	....	....	....	....	10,000
North Banner Cons., G., Cal.	....	....	....	....	20	....	....	....	....	....	20,000
North Belle Isle, S., G., Nev.	....	200	....	....	....	....	....	....	....	....	230,000
North Commonwealth, S., G., Nev.	....	....	....	....	25	....	....	....	....	....	25,000
North Star, G., Cal.	....	150	100	....	50	50	100	....	....	....	450,000
Nugget, G., Colo.	....	....	....	....	....	....	....	5	5	....	10,000
Omaha, G., Cal.	....	....	....	13	....	7	43	43	....	....	106,100
Ontario, S., Utah.	900	900	900	1650	900	750	....	....	....	180	13,355,000
Original, S., C., Mont.	12	6	3	....	....	....	....	....	....	....	138,000
Oro, G., S., L., Colo.	....	....	....	95	....	....	....	....	....	....	95,000
Oro Grande, G., Cal.	....	....	....	....	....	....	....	....	....	....	188,860
Osceola, C., Mich.	100	150	50	225	150	150	100	....	100	125	2,072,500
Pacific Coast Borax, B., Cal.	....	....	....	....	180	180	63	....	....	....	422,500
Pamlico, Nev.	....	21	12	....	....	12	....	....	....	....	189,080
Pandora, Mont.	....	....	....	....	3	3	....	....	....	....	6,000
Paradise Valley, S., Nev.	10	....	....	....	....	....	....	....	....	....	160,000
Parrott, C., S., Mont.	54	144	144	252	360	216	138	67	....	....	1,569,000
Peacock, N. M.	....	....	....	....	....	....	....	....	....	....	50,000
Petro, S., Utah.	....	....	....	....	18	....	....	....	....	....	17,500
Pharmacist, G., Colo.	....	....	....	....	....	36	44	....	....	....	80,000
Pittsburg, G., Nev.	....	30	....	....	....	....	....	....	....	....	29,850
Pleasant Valley, C., Utah.	....	....	....	....	....	....	20	....	....	....	593,056
Plumas Eureka, G., Cal.	18	70	123	....	70	25	53	....	....	....	2,696,294
Plutus, S., Colo.	....	....	....	....	....	....	....	....	....	....	20,000
Plymouth Cons., G., Cal.	375	80	....	....	....	....	....	....	....	....	2,280,000
Poorman, G., Colo.	....	25	15	85	....	....	....	....	....	....	125,000
Poorman, S., Idaho (*k*).	....	....	....	....	....	57	....	....	....	....	56,935
Portland Gold, G., Colo.	....	....	....	....	....	....	....	67	556	240	863,000
Quicksilver, Q., Cal. (*l*).	129	283	193	257	118	....	....	....	....	....	2,475,082
Quincy, C., Mich.	800	360	280	320	400	350	300	400	600	1000	8,670,000
Red Cloud, S., L., Idaho.	....	....	....	20	80	70	10	....	....	....	180,000
Reed & National, S., G., Colo.	....	....	....	45	....	....	....	....	....	....	45,000
Rescue, N. M.	....	....	....	....	....	12	....	....	....	....	12,000
Retriever, So. Dak.	....	....	....	....	13	....	....	....	....	....	12,500
Rialto, G., Colo.	....	....	....	....	32	18	....	....	....	....	50,250
Richmond Cons., S., L., Nev.	68	....	....	....	34	....	14	....	....	....	4,386,780
Rico-Aspen, Colo.	....	....	....	....	....	....	50	300	....	....	350,000
Robinson, S., Colo.	....	....	....	....	....	....	....	....	....	....	585,000
Rocky Fork Coal, Mont.	....	....	....	....	100	100	....	....	....	....	200,000
Rooks, G., Vt.	....	....	....	....	....	....	....	....	....	....	61,000
Running Lode, Colo.	....	....	....	5	15	6	1	....	....	....	27,000
Russell, G., Cal.	30	....	....	....	....	....	....	....	....	....	30,000

DIVIDENDS PAID BY AMERICAN MINES—*Continued.*

Company.	1887	1888	1889	1890	1891	1892	1893	1894	1895	1896	Totals.
Sacramento, G., S., L., Utah	....	....	....	....	....	....	....	....	....	5	$5,000
Saint Joseph, L., Mo	90	....	....	....	150	150	150	150	150	....	2,524,000
Security, G., Colo	....	....	....	....	....	....	....	....	....	....	56,000
Sheridan, S., G., Colo	....	....	....	....	75	....	....	....	....	....	300,000
Sherwood, Z., Mo	3	....	....	....	....	....	....	....	....	....	3,000
Sierra Bella, S., N. M	....	....	....	....	....	....	....	....	....	....	30,000
Sierra Buttes, G., Cal	76	15	....	....	25	36	31	....	....	....	1,584,933
Sierra Nevada, Idaho	....	20	20	....	....	....	....	....	....	....	40,000
Silent Friend, S., L., Colo	....	....	....	....	60	....	....	....	....	....	60,000
Silver Cord, S., L., G., Colo	....	....	45	....	....	....	....	....	....	....	265,000
Silver King, S., Ariz	175	....	....	....	....	....	....	....	....	....	1,950,000
Silver King, Utah	....	....	....	....	....	....	....	....	263	375	825,000
Silver Mining of Lake Valley, N. M	....	25	25	180	80	....	....	....	....	....	300,137
Silverton, S., Colo	....	....	....	....	....	....	....	....	....	....	80,000
Slocan Star, B. C	....	....	....	....	....	....	....	....	50	200	250,000
Small Hopes, S., Colo	600	....	25	25	....	38	25	....	....	25	3,275,000
Smuggler Union, G., S., Colo. (*m*)	....	....	....	....	....	....	....	....	....	150	150,000
Standard, G., Cal	20	50	....	....	10	40	20	40	20	....	3,717,868
Swansea, G., S., L., Colo	....	....	....	....	....	....	....	2	....	....	39,000
Swansea, Utah	....	....	....	....	....	....	....	....	1	20	21,500
Syndicate, Cal	....	....	....	....	....	....	....	....	....	....	72,000
Tamarack, C., Mich	....	640	560	670	600	600	600	400	400	300	4,770,000
Teal & Poe, S., L., N. M	....	....	....	....	9	....	....	....	....	....	9,000
Temonj, G., Colo	....	....	....	....	....	....	....	....	10	....	10,000
Tomboy, G., Colo	....	....	....	....	....	....	....	....	300	110	410,000
Trinity River, Colo	....	....	....	....	....	....	15	....	....	....	15,000
Union, Colo	....	....	....	....	....	....	....	27	13	23	73,000
United Verde, C., S., Ariz	....	....	....	30	....	30	435	....	....	....	562,500
Utah, Utah	....	....	....	....	....	15	5	....	17	22	175,000
Utah Cons., Nev	....	....	....	....	....	....	....	....	....	3	3,000
Valencia, M., N. H	....	....	....	....	....	....	....	....	....	....	41,250
Victor, G., Colo	....	....	....	....	....	....	120	105	240	240	705,000
Victor L. & M., Colo	....	....	....	....	....	....	....	....	24	12	42,000
Viola Limited, S., L., Idaho	38	94	....	....	....	....	....	....	....	....	337,500
Ward Cons., S., Colo	....	....	20	....	....	....	....	....	....	....	20,000
War Eagle, B. C	....	....	....	....	....	....	....	....	133	55	187,500
Webb City, Z., Mo	....	....	4	....	....	....	....	....	....	....	4,400
Whale, Colo	....	....	....	....	5	....	....	....	....	....	5,000
Woodside, S., Utah	....	....	25	....	....	....	....	....	....	....	25,000
W. Y. O. D., G., Cal	....	....	....	....	6	36	24	36	24	....	108,000
Yankee Girl, S., Colo. (*n*)	187	....	....	125	260	....	....	....	....	....	520,000
Yosemite, S., Utah	....	....	....	....	....	5	....	....	....	....	5,000
Young America, G., Cal	....	10	....	....	....	....	....	....	....	....	175,000

(*G*) Gold; (*S*) Silver; (*L*) Lead; (*I*) Iron; (*C*) Copper; (*Q*) Quicksilver; (*B*) Borax; (*Z*) Zinc; (*M*) Mica.

(*a*) Formerly the Alaska Mining and Milling Company, reorganized in 1891 as the Alaska-Treadwell Gold Mining Company; the dividends credited for 1891 and 1892 are the payments of the latter company. The Alaska Mining and Milling Company paid $700,000 previously.

(*b*) Reconstructed into the Golden Leaf, Limited, of Montana.

(*c*) Previous to the consolidation in August, 1884, the California had paid $31,320,000 in dividends and the Consolidated Virginia $42,390,000.

(*d*) Formerly the Young America South Mining Company, reorganized as the Coptis in 1891.

(*e*) Previous to consolidation the Deadwood paid $275,000 and the Terra $75,000.

(*f*) Reconstructed as the Golden Leaf, Limited.

(*g*) Poorman Mines, Limited, operating the Poorman Mine at Silver City, Idaho, which paid large dividends in 1865 and 1866.

(*h*) Jay Hawk & Lone Pine Consolidated Mining Company, Limited.

(*i*) Société Anonyme des Mines de Lexington.

(*j*) Maid of Erin Silver Mines, Limited, formerly Henriette & Maid Consolidated Mining Company. The dividends for 1887, 1888, 1889, and 1890 were paid by the old company, and those for 1891 and 1892 by the new company.

(*k*) Enterprise Mining Company, of Rico, Colo.; there is also an Enterprise Mining Company in Aspen and one in Leadville.

(*l*) Including dividends paid on preferred stock and common stock.

(*m*) Previous to consolidation in 1896 the Smuggler Mining Company had paid $1,140,000 in dividends.

(*n*) Yankee Girl Silver Mines, Limited, formerly Yankee Girl Mining Company. The above statement includes the payments by both the old and new companies.

Assessments Levied by Mining Companies.

Name and Location of Company.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	Total to Jan. 1, 1897.
Ada Cons., Utah	.....	.....	.....	.....	$330	.....	$3,000	.....	$3,333
Alliance, Utah	$30,000	.....	$40,000	.....	.....	.....	50,000	.....	200,000
Allouez, Mich	40,000	$40,000	40,000	.....	.....	$16,000	.....	.....	1,440,937
Alpha, Nev	26,500	42,250	15,000	$36,750	10,000	6,000	15,750	$15,750	257,500
Alta, Nev	.....	54,000	30,000	27,000	25,200	50,400	20,160	32,400	3,579,760

ASSESSMENTS LEVIED BY MINING COMPANIES—*Continued.*

Name and Location of Company.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	Total to Jan. 1, 1897.
Anchor, Utah	$15,000	.....	150,000	.....	$90,000	.....	.....	.....	$560,000
Andes, Nev	25,000	$25,000	30,000	$25,000	.....	$25,000	$15,000	$15,000	260,000
Argenta, Nev	10,000	.....	.....	.....	.....	.....	.....	.....	335,000
Baltimore, Nev	50,000	20,000	.....	.....	.....	.....	.....	.....	145,000
Belcher, Nev	104,000	104,000	104,000	78,000	.....	50,200	50,200	52,000	3,312,420
Belle Isle, Nev	.....	15,000	.....	30,000	20,000	.....	.....	10,000	240,271
Bellevue-Idaho, Idaho	12,500	16,037	.....	.....	.....	.....	.....	.....	104,787
Benton Cons., Nev	.....	.....	.....	.....	.....	.....	.....	.....	556,000
Best & Belcher, Nev	75,200	149,485	100,800	50,000	75,600	50,400	50,400	25,200	2,606,425
Bodie Cons., Cal	75,000	50,000	.....	.....	40,000	.....	.....	15,000	745,000
Bodie Tunnel, Cal	.....	25,000	25,000	25,000	.....	.....	.....	.....	202,906
Brunswick Cons., Cal	.....	.....	20,000	20,000	.....	8,000	16,000	12,000	78,000
Buckeye, Utah	.....	.....	.....	.....	.....	.....	.....	3,750	3,750
Bullion, Nev	.....	25,000	50,000	100,000	25,000	30,000	40,000	20,000	3,030,000
Bulwer Cons., Cal	50,000	.....	15,000	.....	10,000	10,000	5,000	5,000	175,000
Bunker Hill, So. Dak	.....	.....	.....	.....	.....	.....	3,750	625	4,375
Burlington, Cal	.....	.....	.....	.....	.....	.....	.....	3,000	3,000
Butte & Boston, Mont	.....	.....	.....	.....	.....	.....	1,500	1,500	3,000
Butte Queen, Cal	.....	.....	.....	4,000	.....	.....	.....	.....	16,000
Caledonia Silver, Nev	.....	.....	150,000	.....	.....	50,000	.....	5,000	3,240,000
California, Cal	.....	.....	.....	6,000	.....	.....	4,500	.....	16,500
Centennial-Eureka, Utah	30,000	.....	.....	.....	.....	.....	.....	.....	30,000
Central North Star, Cal	.....	.....	.....	.....	10,000	.....	.....	.....	10,000
Challenge Cons., Nev	25,000	50,000	50,000	45,200	15,000	50,000	5,000	7,500	300,000
Chollar, Nev	112,000	.....	168,000	112,000	61,600	56,000	56,000	28,000	2,021,600
Cœur d'Alêne, Idaho	.....	.....	.....	.....	.....	.....	.....	.....	25,000
Commonwealth, Nev	.....	.....	.....	30,000	.....	.....	.....	.....	190,000
Comstock, Nev	.....	.....	.....	.....	.....	.....	.....	.....	30,000
Comstock, Utah	.....	.....	.....	.....	.....	.....	.....	25,000	25,000
Concord, N. C	.....	.....	.....	.....	.....	.....	.....	.....	6,000
Concordia, Nev	.....	.....	.....	.....	.....	.....	.....	.....	75,000
Confidence, Nev	.....	18,720	18,720	49,920	.....	6,240	14,976	7,488	1,636,974
Cons. Cal. & Va., Nev	.....	.....	.....	108,000	6,220	108,000	54,000	118,800	4,960,130
Cons. Imperial, Nev	25,000	75,000	150,000	26,500	.....	.....	5,000	500	2,082,000
Cons. New York, Nev	25,000	45,000	30,000	20,000	25,000	5,000	10,000	5,000	165,000
Cons. Pacific, Nev	15,000	6,000	.....	.....	.....	.....	.....	.....	198,000
Courier, Idaho	.....	.....	.....	.....	.....	.....	.....	.....	10,000
Crocker, Ariz	20,000	25,000	20,000	5,000	5,000	.....	.....	.....	180,000
Crown Point, Nev	100,000	.....	150,000	100,000	80,000	65,000	25,000	70,000	2,965,000
Dalton, Utah	.....	.....	.....	.....	3,750	5,000	5,000	5,000	43,750
Del Monte, Nev	20,000	20,000	29,050	20,000	.....	.....	.....	.....	120,000
Derbec Blue Gravel, Cal	.....	.....	.....	10,000	5,000	.....	.....	.....	15,000
Dexter, Nev	.....	8,000	.....	.....	.....	.....	.....	.....	8,000
Diana, Nev	.....	.....	.....	8,000	.....	.....	.....	.....	83,000
East Best & Belcher, Nev	.....	25,000	45,000	20,000	.....	20,000	.....	.....	110,000
East Sierra Nevada, Nev	10,000	.....	10,000	.....	.....	5,000	.....	.....	25,000
Eureka Cons., Nev	50,000	.....	.....	.....	.....	.....	12,500	.....	562,500
Eureka Cons., Utah	.....	.....	.....	.....	.....	.....	.....	5,000	5,000
Exchequer, Nev	25,000	50,000	25,000	45,000	15,000	10,000	5,000	5,000	725,000
Felice, Ariz	.....	.....	.....	.....	.....	.....	.....	.....	20,000
Fisher, Ariz	.....	.....	.....	.....	.....	.....	.....	.....	20,000
Flowery, Nev	.....	.....	.....	.....	.....	.....	.....	.....	130,000
Found Treasure, Nev	12,500	45,000	.....	50,000	.....	.....	.....	.....	131,500
Gold Belt, Utah	.....	.....	.....	.....	1,345	.....	.....	1,667	3,012
Gold Flat, Cal	.....	.....	.....	11,000	2,000	.....	.....	.....	13,000
Goodyear, Mont	.....	.....	.....	2,000	4,183	.....	.....	.....	17,183
Gould & Curry, Nev	91,800	60,400	64,800	76,400	80,200	32,400	48,600	48,600	4,850,400
Grand Prize, Nev	120,000	25,000	.....	.....	.....	.....	.....	.....	785,000
Hale & Norcross, Nev	.....	56,000	168,000	168,000	56,000	56,000	39,200	61,600	5,758,800
Hartery Cons., Cal	.....	5,000	5,000	.....	.....	2,000	2,000	2,000	33,000
Hartshorn, So. Dak	.....	6,250	.....	.....	.....	.....	.....	.....	8,750
Hayward Group, So. Dak	.....	2,000	.....	.....	.....	.....	.....	.....	2,000
Head Cent. & Tranq., Nev	.....	.....	.....	.....	.....	.....	.....	.....	22,824
Heath, Idaho	.....	.....	.....	.....	.....	.....	.....	.....	25,000
Hector, Cal	45,000	.....	.....	.....	.....	.....	.....	.....	45,000
Hidden Treasure, Cal	.....	.....	.....	.....	1,000	.....	.....	.....	1,000
Himalaya, Utah	.....	.....	1,800	1,800	.....	.....	.....	.....	10,000
Holmes, Nev	.....	25,000	.....	.....	.....	.....	.....	.....	345,000
Honorine, Utah	12,500	.....	12,500	.....	.....	.....	.....	.....	50,000
Hudson Bay, Cal	.....	.....	.....	.....	10,000	.....	.....	.....	10,000
Huron Mich	.....	.....	.....	.....	.....	.....	.....	.....	280,000
Independence, Nev	.....	.....	.....	5,000	.....	.....	.....	.....	345,000
Iron Hill, So. Dak	15,000	20,625	15,000	.....	.....	.....	.....	.....	169,375
Jack Rabbit, Cal	.....	.....	.....	15,000	13,000	5,000	.....	.....	118,000
Jackson, Nev	.....	.....	.....	.....	10,000	.....	.....	.....	247,500
John Duncan, Mich	2,000	.....	.....	.....	.....	.....	.....	.....	4,000
Julia Cons., Nev	.....	.....	11,000	.....	5,000	.....	5,500	5,500	1,490,000
Justice, Nev	.....	.....	26,250	42,000	30,000	75,000	20,000	15,750	3,665,750
Kearsarge, Mich	.....	.....	.....	.....	.....	.....	.....	.....	190,000
Kentuck Cons., Nev	.....	.....	36,750	31,500	10,500	26,250	10,500	5,250	119,500
Keystone, Nev	.....	.....	.....	.....	.....	.....	.....	.....	240,000
Keyes, Nev	30,000	.....	.....	.....	.....	.....	.....	.....	125,000
Kingman Silver, Ariz	.....	.....	5,000	.....	.....	.....	.....	.....	5,000

ASSESSMENTS LEVIED BY MINING COMPANIES—*Continued.*

Name and Location of Company.	1889.	1890.	1891.	1892.	1893.	1894.	1895.	1896.	Total to Jan. 1, 1897.
King of the West, Idaho	.....	.....	.....	.....	.....	.....	.....	.....	$45,000
Kossuth	.....	.....	.....	.....	.....	.....	.....	.....	433,000
Lady Washington, Nev	.....	.....	$21,400	.....	.....	.....	.....	$5,400	157,800
La Plata, Nev	$3,000	.....	.....	.....	.....	.....	.....	.....	3,000
Leo, Mont	.....	.....	.....	.....	.....	.....	.....	10,000	52,500
Little Pittsburg, Utah	.....	.....	.....	.....	.....	.....	$4,000	6,000	10,000
Locomotive, Ariz	10,000	$5,000	.....	.....	.....	.....	.....	.....	115,000
Lone Star Cons., Cal	.....	.....	.....	$5,000	.....	.....	.....	.....	12,500
Lucky Bill, Utah	.....	.....	.....	.....	.....	.....	.....	7,200	48,000
Manhattan, Nev	.....	.....	.....	.....	.....	.....	.....	.....	250,000
Martin White, Nev	25,000	50,000	50,000	50,000	$25,000	$25,000	.....	.....	1,350,000
Mayflower, Cal	55,000	.....	.....	.....	.....	.....	.....	.....	470,000
Mexican, Nev	50,400	25,200	50,400	75,600	75,600	75,600	75,600	40,320	3,084,080
Michigan Gold, Mich	.....	.....	.....	10,000	.....	.....	.....	.....	40,000
Mikado, Mich	.....	.....	.....	.....	.....	.....	.....	.....	15,200
Milwaukee, Mont	.....	.....	2,500	.....	.....	.....	.....	.....	12,500
Missoula Placers, Mont	.....	.....	.....	.....	.....	.....	.....	.....	4,000
Modoc Chief, Idaho	.....	.....	.....	4,375	.....	.....	.....	.....	4,375
Mollie Gibson, Colo	.....	.....	.....	10,000	.....	.....	.....	.....	20,000
Montreal, Utah	.....	.....	.....	750	.....	.....	375	.....	4,875
Mono, Cal	62,500	12,500	12,500	.....	.....	.....	5,000	10,000	807,500
Mount Diablo, Nev	.....	.....	.....	.....	.....	.....	.....	7,500	145,000
Mount Terry, So. Dak	.....	750	.....	.....	.....	.....	.....	.....	750
Navajo, Nev	10,000	15,000	15,251	20,000	20,000	.....	.....	.....	555,521
Nevada Queen, Nev	70,000	.....	15,000	25,000	25,000	.....	5,000	.....	270,000
North Banner, Cal	.....	.....	.....	.....	.....	.....	.....	10,000	21,794
North Belle Isle, Nev	100,000	20,000	50,000	20,000	38,075	.....	.....	10,000	523,075
North Bonanza, Nev	10,090	.....	.....	.....	.....	.....	.....	.....	240,000
No. Commonwealth, Nev	30,000	25,000	25,000	.....	10,000	.....	.....	.....	120,000
North Comstock, Nev	.....	.....	.....	.....	.....	.....	.....	.....	10,000
North Eureka, Utah	.....	.....	.....	.....	.....	.....	.....	1,500	1,500
North Extension, Nev	.....	.....	.....	.....	.....	.....	.....	.....	25,000
No. Gould & Curry, Nev	20,000	.....	30,000	10,000	10,000	20,000	.....	10,000	320,000
North Occidental, Nev	.....	6,000	.....	.....	.....	.....	.....	.....	13,000
North Peer, Ariz	5,000	5,000	.....	.....	.....	.....	.....	.....	21,000
Occidental Cons. Nev	50,000	75,000	25,000	50,000	55,000	30,000	30,000	65,000	453,652
Ophir, Nev	50,000	50,000	50,000	50,000	100,000	100,000	100,000	25,200	4,635,840
Original Keystone, Nev	.....	.....	.....	10,000	.....	.....	.....	.....	250,000
Osceola Cons., Cal	.....	.....	.....	.....	.....	.....	.....	1,000	3,000
Overman, Nev	57,600	28,800	79,340	126,720	60,000	34,560	23,040	34,560	4,188,560
Paradise Valley, Nev	.....	.....	.....	.....	.....	.....	.....	.....	57,000
Pennsylvania Cons., Cal	.....	.....	2,750	.....	.....	.....	.....	.....	36,050
Peer, Nev	20,000	10,000	15,000	20,000	10,000	5,000	.....	.....	215,000
Peerless, Nev	90,000	25,000	10,000	5,000	5,000	5,000	.....	.....	410,000
Phil Sheridan, Nev	35,000	.....	.....	.....	.....	.....	.....	.....	65,000
Pine Hill, Cal	.....	.....	.....	.....	3,000	3,000	5,000	5,000	20,000
Potosi, Nev	10,000	55,400	112,000	56,000	84,000	112,000	56,000	28,000	2,021,600
Queen Bee, So. Dak	.....	3,000	.....	.....	.....	.....	.....	.....	3,000
Rainbow, So. Dak	.....	.....	.....	1,250	.....	.....	438	425	5,113
Ropes, Mich	.....	.....	.....	.....	20,000	.....	.....	.....	20,000
St. Mary's Copper, Mich	.....	.....	.....	.....	2,000	.....	2,000	.....	4,000
Sampson, Utah	.....	.....	.....	.....	.....	.....	.....	.....	288,257
San Francisco, Cal	.....	.....	.....	.....	.....	.....	.....	.....	22,000
Savage, Nev	112,000	.....	112,000	122,000	112,000	100,800	67,200	67,200	1,029,000
Scorpion, Nev	120,000	90,000	.....	5,000	.....	5,000	.....	.....	415,000
Seg. Belcher & Mides, Nev	50,000	80,000	50,000	25,000	35,000	20,000	20,000	10,000	340,000
Seg. Iron Hill, Nev	.....	.....	.....	.....	.....	.....	.....	.....	8,750
Sierra Nevada, Nev	100,000	71,910	80,000	55,000	45,000	50,000	50,000	50,000	4,555,910
Silver Hill, Nev	43,200	43,200	30,000	16,200	5,400	5,400	.....	.....	1,992,600
Silver King, Ariz	.....	30,000	60,000	25,000	.....	100,000	.....	75,000	340,000
Siskiyou Cons., Cal	.....	.....	8,000	7,000	9,000	14,000	4,000	2,000	44,000
Standard Cons., Cal	.....	50,000	.....	.....	.....	.....	.....	.....	99,887
Summit, Cal	2,500	.....	.....	.....	.....	.....	.....	.....	120,000
Taylor Plumas, Cal	.....	.....	10,000	.....	.....	.....	.....	.....	20,000
Telegraph, Cal	.....	.....	.....	975	.....	.....	.....	.....	3,575
Teresa, Mex	.....	.....	.....	20,000	60,000	.....	15,000	.....	155,000
Tetro, Utah	.....	.....	.....	.....	.....	.....	.....	3,000	9,000
Tioga Cons., Cal	.....	.....	.....	.....	.....	.....	.....	.....	295,000
Triumph, Idaho	.....	.....	.....	.....	.....	.....	.....	.....	20,000
Trojan, Nev	10,000	.....	.....	.....	.....	.....	.....	.....	370,000
Tuscarora, Nev	10,000	.....	.....	.....	.....	.....	.....	.....	15,000
Union, Utah	.....	.....	.....	.....	.....	.....	.....	.....	7,000
Union Cons., Nev	75,000	50,000	80,000	50,000	45,000	35,000	20,000	40,000	545,000
Utah Cons., Utah	.....	.....	.....	.....	25,000	5,000	5,000	10,000	415,722
Valenzuela, Mex	.....	.....	.....	.....	.....	.....	2,000	.....	6,000
Victory, So. Dak	.....	.....	.....	.....	.....	.....	.....	375	2,625
Wall Street, Mont	.....	.....	.....	900	.....	.....	.....	.....	1,500
Waterloo, Cal	.....	.....	.....	.....	30,000	.....	.....	.....	30,000
Weldon, Ariz	10,000	10,000	10,000	5,000	10,000	.....	.....	.....	65,000
Wolverine, Mich	.....	.....	.....	.....	.....	.....	60,000	.....	60,000
Wood River, Idaho	.....	.....	3,000	.....	.....	.....	.....	.....	3,000
W. Y. O. D., Cal	.....	.....	22,500	.....	.....	.....	.....	.....	22,500
Yellow Jacket, Nev	60,000	.....	.....	156,000	90,000	90,000	90,000	90,000	6,144,000

THE DIFFUSION OF METALS.

By W. C. Roberts-Austen.

The diffusion of molten and solid metals, due to their molecular mobility, is of great interest, not only in relation to the constitution of matter, but also in connection with industrial undertakings. Many interesting problems are involved, even the partial elucidation of which cannot fail to be of great importance scientifically as well as in practical metallurgical work.

The analogy of alloys to ordinary saline solutions has often been pointed out, and many experiments have recently been devoted to comparing the action of osmotic pressure in saline solutions and in alloys, as measured by the lowering of the freezing point, which is caused by the addition to the solvent of a small quantity of another body. The general effect is the same whether the solvent is a liquid like water or a molten metal. Very little attention has, however, been given to the consideration of the molecular movements which enable two or more molten metals to mix spontaneously and form a truly homogeneous mass, although it is by such an investigation that the analogy of an alloy to a saline solution may reasonably be expected to be more clearly revealed than by any other method of research. A single example of the spontaneous mixing of two metals may be useful. In preparing the alloy of gold and copper used for coinage, some 33 kgms. of gold and 3 kgms. of copper are melted together in a single crucible, and the results of assays on the first and the last portions of metal poured from the crucible seldom differ by more than 0.0001 part. Such a fluid mass of standard gold owes this remarkable uniformity in composition not alone to the mechanical stirring by which the blending of the gold and copper was roughly effected. The molecular mobility of the metals has influenced the result, and the metals dissolved in each other become, by spontaneous process, spread, or diffused, uniformly; in this case the uniformity is not materially disturbed when the solidification of the mass is effected.

The following account of experiments which I have conducted on the diffusion of both molten and solid metals is condensed somewhat from the Bakerian discourse delivered before the Royal Society, February 20, 1896.

Considering the importance of the subject, and the great interest connected with the diffusivity of the metals consequent upon molecular mobility, the absence of direct experiments is remarkable. This, however, has been doubtless due to the great difficulty of conducting them.

My long connection with Professor Thomas Graham's researches on the diffusion of fluids almost imposed on me the duty of extending the investigation to the diffusion of metals. The difficulties in the way were so great that I long hesitated to undertake the labor. It was necessary to work at relatively high temperatures, and until quite recently there were no means of measuring with approximate accuracy variations of high degrees of heat. Moreover, the prevalent belief that convection currents must affect diffusion when solids were being reduced to a molten state seemed to add to the difficulties. Ostwald's* statement with reference to the diffusion of salts, that "to make accurate experiments on diffusion is one of the most difficult problems in practical physics," naturally gives rise to doubts whether any method which seemed available for conducting such investigations with molten metals would afford trustworthy results. The first efforts I made fourteen years ago were abandoned because I had not the means of measuring with sufficient accuracy the temperature at which diffusion took place. It was only two years ago that the experiments were resumed.

Prior to my researches undertaken in 1881 the literature of the subject was confined to Homberg's† paper on "Substances which Penetrate and Pass through Metals without their being Melted." He incidentally showed by experiment the extreme rapidity with which mercury penetrated a bar of iron.

In a brief communication to the Chemical section of the British Association, at the meeting at Southport, in September, 1883,‡ were embodied the results of the first experiments ever made with the direct object of investigating the diffusion of molten metals and alloys, other than those of mercury, which are fluid at the ordinary temperature. I stated that "while molten copper and antimony interpenetrate but slowly, the mobility of gold and silver in molten lead is comparatively rapid. Two months later Guthrie§ published a remarkable paper on "Certain Molecular Constants," in which the diffusion of zinc, lead, tin, sodium, and potassium in mercury was studied, and he stated that these metals, which, of course, are much lighter than mercury, "appear after a month's interval in appreciable quantity at a depth of a foot beneath the surface when the temperature is about 16° or 17°." He concluded by offering "a general curve of amalgamation," which he thought would represent the rate at which the metals examined by him alloy with mercury, and this curve he was of opinion might also "represent the relative rates of atomic and molecular diffusion generally."

Guthrie held that, as the mercury he employed was a good conductor of heat, there was not much fear of the disturbing influence of convection curves. Nevertheless, the existence of such currents gave me much anxiety during my earlier experiments undertaken in 1881.

DIFFUSION OF MOLTEN METALS.

From the outset of this research both molten lead and bismuth were chosen as suitable fluids in which the diffusion of other metals could be studied. Advantage was also taken of the fact that at temperatures well above the melting point neither of these metals unite with iron. The precious metals, also, when alloyed with lead or bismuth, do not show any tendency to unite with iron unless

* *Solutions*, Eng. Edition, 1891, p. 122.
† *Mem. de l'Acad. Royal des Sciences*, 1713, 1839, p. 306.
‡ Roberts-Austen, *Brit. Assn. Report*, 1883, p. 402.
§ *Phil. Mag.*, Vol. XVI., 1883, p. 321.

it is very clean and bright. In the first instance single wrought-iron tubes filled with lead were arranged vertically in a bath of lead which was kept well above its melting point. Weighed quantities of heated, but still solid, gold or platinum were then rapidly lowered through the lead in little covered receptacles of iron to the bottom of the tube, and when by the aid of a rigid steel wire the removal of the covers was gently effected, the gold was exposed to the lead, it became rapidly dissolved, and diffusion began. The tubes filled with molten lead, in which diffusion took place, were about 200 mm. long, and many such tubes were arranged in a single bath, which was carefully kept hotter at the top than at the bottom, so as to avoid as much as possible the carrying of the precious metal upward by any streams of lead which might rise as convection currents from the bottom of the tube. The main result of these earlier experiments was to show that samples of lead—which were removed by sucking them from the upper part of the tubes into stems of tobacco pipes—always revealed the presence of weighable quantities of gold after a lapse of the first three hours, while a sample

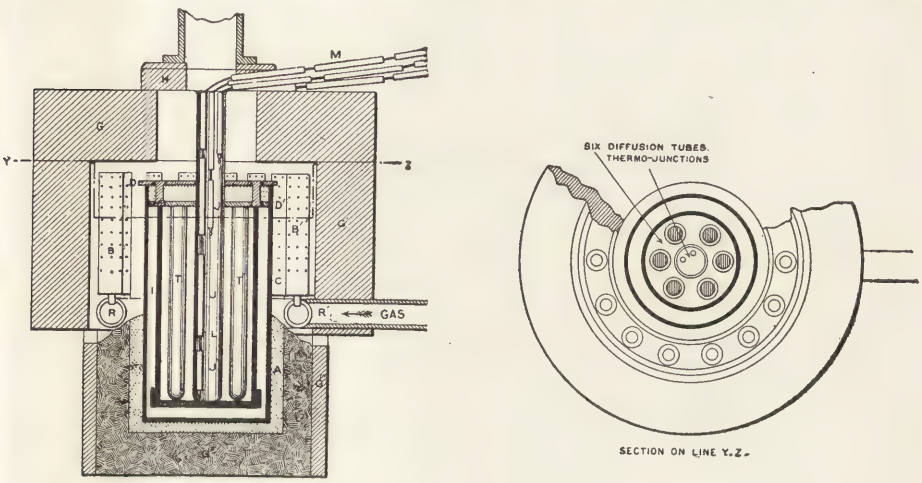


FIG. 1.

withdrawn at the end of a second period of three hours did not indicate the presence of a commensurate amount of the precious metal. This fact either pointed to defects in the method, to the transmission of gold by convection currents, or to the very rapid diffusion of gold when minute quantities of this metal are present in lead. I believe that the latter will ultimately prove to afford the true explanation of the facts observed.

The vertical tubes were then replaced by U-tubes of wrought iron, each limb of which was 230 mm. long and 10 mm. internal diameter. The tubes were filled with lead, and heated externally in a bath of molten lead, and the precious metal, in the form of a rich alloy with lead, was inserted into one limb of the tube. Experiments proved that the gold falling by gravity became rapidly and uniformly distributed through the column of lead in that side of the tube into which it was introduced, and in the rounded part of the tube at the base of the U.

It was found, however, that the use of U tubes greatly increased the difficulties of calculation and single tubes closed at the base were then adopted.

These tubes, that contained the lead in which the diffusion takes place, were arranged in a little furnace of special construction, composed of an interior iron cylinder inclosed in a cylinder of thick copper (Fig. 1).

The heating was effected by a series of clay gas-burners BB', mounted on a ring RR', the burners surrounding the upper portion of the copper cylinder. Investing cases of fireclay $GG'G''G'''$, and a lid of clay H, completed the construction of the furnace. A gas supply of constant volume was secured by means of a regulator and a delicate gauge.

It was absolutely necessary to have a perfectly trustworthy means for the measurement of the temperature, and this was secured by the aid of thermo-junctions shown at $JJ'J''$, the continuation of which are at M, to connect with the wires leading to the measuring instrument, not illustrated.

The particular appliance used in these experiments for recording the temperature was photographic and so arranged as to record by the variations of a spot of light on a sensitized plate. A full description of this apparatus will be found in the *Royal Society Proceedings*, Vol. XLIX., p. 347. In Fig. 1 will be seen a

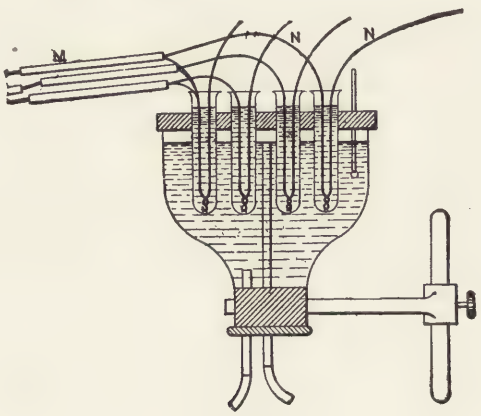


FIG. 2.

central tube of metal L, into which are inserted the thermo-junctions $JJ'J''$, each being protected and insulated by clay tubes which nearly block up the inner space and prevent the free circulation of air. Fig. 2 is a cold junction with which $JJ'J''$ communicates, M being the connection to the thermo-junction, and N the wires leading to the photographic measuring apparatus. The temperature of any given junction can at any moment be either observed on a transparent scale or recorded on a photographic plate by this instrument, and a continuous record of the temperature at three positions in the air bath may be obtained.

I found that convection currents are not established even when the temperatures at the top and bottom of the diffusion tubes only differ by 35° C., and that a graduated temperature can be dispensed with, as the difference in densities of the solvent and the diffusion metal will answer. But it is necessary to have an apparatus which will automatically maintain for lengthy periods any given temperature between the ordinary temperature and 600° C.

At the end of a period varying from 6 hours to 7 days, the diffusion tubes were

removed, cooled from below, carefully measured, and then cut into transverse numbered sections, the metallic contents of each of which were weighed and analyzed. When gold or platinum is the diffusing metal, cupellation assays and check assays of samples synthetically prepared were made simultaneously.

If the bath is unprotected below, and heated from the top, the top will be very much hotter than the bottom, and thus the rate of diffusion would be considerably greater near the top than near the bottom of each tube. The value of the results would also be much impaired if the bath were inadequately protected on the top, for if the upper surface of the bath is cooled, convection currents will be established. The lead, moreover, at the surface oxidizes, and forms a detachable slag, and so relatively increases the percentage of gold or platinum in the upper layer of lead. It was frequently noticed that the top layer in the tubes was slightly richer than the one next below it.

In the experiments the temperature, as indicated by the top thermo-junction, was gradually raised to 550°, at which point it was maintained practically without variation during the greater part of the experiment, which lasted nearly seven days. The temperature, however, slowly fell during the last two days to 500°, owing to the choking up of the burners. The middle thermo-junction showed temperature 25°, and the lowest one 35°, below this, the mean temperature shown by the middle junction being given as the temperature of the experiment. It should be pointed out that these temperatures are based on measurements which assume the melting point of gold to be 1045°, and lead 325°. Recent work seems to show that 1045° may be some 15° too low.*

It is now held that liquid diffusion is the result of osmotic pressure.† A movement of the particles (molecules or atoms) of the dissolved substance takes place, and a molecular force drives them from the place where they are more closely packed and, therefore, exert greater pressure, and impels them to positions in which they are more widely distributed. This movement continues until the concentration, and, therefore, the pressure of the diffusing metal is constant throughout the liquid. Graham's method of studying liquid diffusion consisted in filling wide-mouth vials of glass with the solutions of salts, which were allowed to diffuse outward into water contained in capacious cylinders. This method could not well be imitated in the present experiments, as the manipulation, and the calculation of the results obtained by such a method, present great difficulty. In the earliest experiments made by me, in 1883, the little spheres of precious metal obtained from each measured section of lead were arranged on a card scale at measured distances.

After the experiments were far advanced, evidence was obtained proving that the tables calculated by Stefan for diffusion of salts were applicable to the present method of investigation. By the help of these tables the diffusion constant can be determined, in the case of the single-tube experiments, if the distribution of the dissolving metal is known.

The differential equation $\frac{dv}{dt} = K \frac{d^2v}{dx^2}$ expresses the movement in linear diffusion; x represents the distance in the direction in which the diffusion takes place; v is the degree of concentration of the diffusing metal, and t the time; K

* Heycock & Neville, *Trans. Chem. Soc.*, Vol. LXVII., 1895, p. 160; Roberts-Austen, *Nature*, May 9, 1895. p. 40.
† Nernst, *Zeit. für Physikal. Chem.*, Vol. II., 1888, p. 613.

is the diffusion constant, *i. e.*, the number which expresses the quantity of the metal in grams, diffusing through unit area (1 sq. cm.) in unit time (1 day) when unit difference of concentration (in grams per cu. cm.) is maintained between the two sides of a layer 1 cm. thick.

The following table contains the results of the more recent experiments which have been made. *K*, the diffusion constant, is given in sq. cm. per diem and also in sq. cm. per second. The result for the diffusion in tin is, however, less trustworthy than the other data.

Diffusing Metal.	Solvent.	Temperature. Degrees.	*K* in Square Centimeters.	
			Per Diem.	Per Second.
Gold	Lead	492	3.00	3.47×10^{-5}
Gold	Lead	492	3.07	3.55×10^{-5}
Platinum	Lead	492	1.69	1.96×10^{-5}
Platinum	Lead	492	1.69	1.96×10^{-5}
Gold	Lead	555	3.19	3.69×10^{-5}
Gold	Bismuth	555	4.52	5.23×10^{-5}
Gold	Tin	555	4.65	5.38×10^{-5}
Silver	Tin	555	4.14	4.79×10^{-5}
Lead	Tin	555	3.18	3.68×10^{-5}
Gold	Lead	550	3.185	3.69×10^{-5}
Rhodium	Lead	550	3.035	3.51×10^{-5}

Graphically these results are expressed by curves, the horizontal lines representing distance in the direction in which diffusion takes place, and the metrical concentration. If the diagram is turned so as to make the horizontal lines vertical the actual distance the metals diffused upward will be apparent.

The results given in the preceding table show that the diffusion of metals is not increased nearly so rapidly by a rise in temperature of 50° as the ordinary aqueous diffusion of salts is. This is probably due to the fact that the resistance presented by metals to diffusing metallic molecules is not much reduced by heat, and it may be that the molecules of the diffusing substance are not so liable to disruption by a rise in temperature. It may be well to wait until more results have been obtained before attempting to deduce evidence as to the molecular condition of the metals composing these alloys, though, in this respect, metallic diffusion presents several advantages over the diffusion of salt solutions, the latter being very limited as regards the choice of a solvent and range of temperature at which experiments could be made; and, moreover, if a salt is dissociated into its ions its diffusion rates will be modified. The fact that when water is used, both the salt and the solvent are chemical compounds, renders their diffusivities less directly significant than those of metals, because, with the exception of gaseous elements, molten metals present the simplest possible case which can occur, as they represent the diffusion of one element into another.

A few general deductions may be drawn.

It will be seen that gold diffuses more rapidly in bismuth and in tin than it does in the heavier metal lead. It has also been observed that platinum diffuses faster in bismuth than in lead. The diffusion of platinum and of gold is increased in about equal ratio by the substitution of bismuth for lead as a solvent. On the other hand, platinum diffuses much more slowly in lead than gold does, although their atomic weights and their densities do not greatly differ. Rhodium, another metal of the platinum group, diffuses in lead nearly as fast as gold does, but if

allowance be made for the smaller atomic weight, it will be found to agree fairly well with platinum. This would point to the conclusion that the platinum metals are molecularly more complex than either gold or silver, as a complex molecule exerts less osmotic pressure and diffuses more slowly than a comparatively simple one.

The early workers on diffusion of salts used water as a solvent; Kawalki,* however, has recently given tables of diffusions of salts in both water and alcohol. He found that there is a fairly constant ratio between the diffusivities of salts in

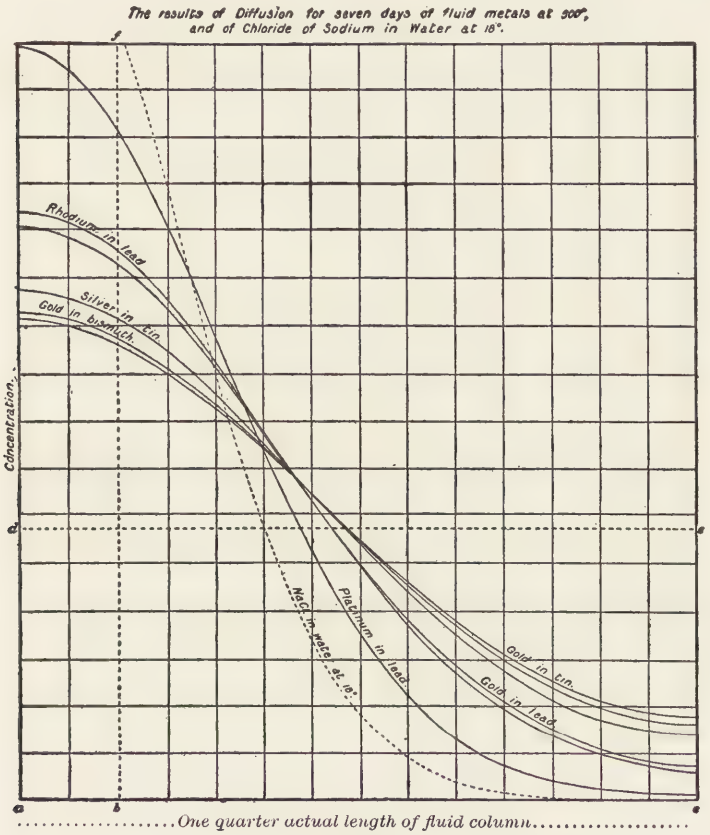


FIG. 3.

the two solvents. Experiments seem to show that this is also true of two metallic solvents such as lead and bismuth.

Diffusions of Amalgams in Mercury.—The diffusivity of gold in mercury at 11° C. is 0.72 sq. cm. per day, the diffusivity of gold in lead being 3.0 sq. cm. per day at 500°. Dr. Guthrie published in 1883 particulars of some experiments of this kind, the metals he selected being zinc, tin, lead, sodium, and potassium, diffusing in mercury. He did not make any calculations with a view to obtaining either the absolute or the relative diffusivities of these metals in mercury. Approximate results have, however, been obtained from his data, and from cer-

* *Wied. Ann.*, Vol. LII., 1894, p. 300.

tain measurements of the original apparatus, now deposited in the South Kensington Museum. These show that his inference "that potassium and sodium have a far greater diffusive energy than the heavier metals examined" is not supported by the actual result of his experiments. His results, calculated by the method given in the present paper, give the values of K, in sq. cm. per day, as follows:

Tin in mercury at about 15°	1.22
Lead in mercury at about 15°	1.00
Zinc in mercury at about 15°	1.00
Sodium in mercury at about 15°	0.45
Potassium in mercury at about 15°	0.40

DIFFUSION OF SOLID METALS.

The foregoing experiments with molten metals naturally suggested the inquiry whether gold would still permeate lead if the temperature were maintained at a point far below the melting point of lead? Would diffusion take place through solid lead at the ordinary temperature, or must a certain amount of viscosity be given to it by the application of a moderate heat? A brief historical sketch of the subject may be useful.

Kernel Roasting.—There has long been a prevalent belief that diffusion can take place in solids, and the practice in conducting certain important industrial processes supports this view. One of these processes, which is of comparatively ancient date, has certainly been employed since 1692 at Agordo, and its results are as follows: When lumps of cupriferous iron pyrites are subjected to very gradual roasting with access of air, the copper becomes concentrated as a "kernel" of nearly pure sulphide of copper in the center of a mass of ferric oxide, while, at the same time, the silver originally present in the ore travels outward and forms a glistening shell on the exterior. These complicated changes must be effected in the solid by a movement allied to diffusion.

Cementation Processes.—Of all the processes which depend on the diffusion of solids probably the most interesting is the truly venerable one by which silver may be recovered from either plates or globules of solid gold by "cementation," the name being derived from the "cement" or compound in which the plates were heated. Its nature was indicated by Pliny, and the manipulation it involved was minutely described by Geber in the eighth century, as well as by many of the early metallurgists; Savot,* for instance, pointed out in the early part of the seventeenth century that "cementation" will deprive gold of the silver it contains, "however small" the amount of the latter metal may be, so that it will be evident that the elimination of the silver from the center of a mass of solid gold must also be effected by an inter-molecular movement allied to diffusion. The evidence, however, is not conclusive, because gaseous chlorine intervenes, and may even play an important part in the penetration of the solid metal.

In another ancient "cementation" process, the conversion of strongly heated but still solid iron into steel is effected by the passage of solid carbon into the interior of the mass of iron, and the explanations which have from time to time been given of the process form a voluminous literature. Le Play considered cementation, which is really a slow creeping action of one solid into another, to

* *Discours sur les Médailles antiques*, 1627, p. 76.

be "an unexplained and mysterious operation," and he attributed the transmission of the carbon to the center of the iron solely to the action of gaseous carbonic oxide. Gay Lussac* confessed that a study of the process shook his faith "in the belief generally attributed to the ancient chemists that *corpora non agunt nisi soluta,*" for it is certain, he adds, "that all bodies, solid, liquid, or aëriform, act upon each other, but, of the three states of bodies, the solid state is the least favorable to the exercise of chemical affinity."

In 1881 M. A. Colson† communicated a paper to the Académie des Sciences, in which he showed that when iron is heated in carbon there is a mutual interpenetration of carbon and iron at so low a temperature as 250°. The interpenetration of solids, as distinguished from the diffusion of two metals in each other, has received attention from many experimenters, of whose work brief mention will only be given, as the subject of this part of the paper is the diffusion of solid metals. Colson pointed out that pure silver diffuses as chloride in dry chloride of sodium, and he states that calcium passes into platinum when the latter is heated in lime, and that silica diffuses through carbon and yields its silicon to platinum. The permeation of strongly heated porcelain by carbon has been demonstrated by Marsden, Violle, and other experimenters. Spring,‡ in 1885, showed that solid barium sulphate and sodium carbonate react on each other until an equilibrium is established.

Any lingering doubt as to whether gas need necessarily intervene in the cementation of iron was removed by an experiment of my own,§ in 1889, which showed that pure iron may be carburized by diamond in vacuo, at a temperature far below the melting point of iron and under conditions which absolutely preclude the presence or influence of occluded gas. I am indebted to my friend, M. Osmond, for a photograph (Fig. 4) of a section, magnified 100 diameters, through a piece of electro-iron, which had been heated to 1500° and carburized from the upper end by contact with the diamond form of carbon, and this section clearly shows the gradual penetration of the iron by carbon. The white grains at the right are unchanged iron. The beautiful work of Osmond on the transformations of iron, and of iron and carbon, affords, moreover, a striking proof of the molecular mobility of solid iron at a temperature which is at least 600° below its melting point.

The Penetration of Solid Metals by Gases.—This subject was, as is well known, investigated by Graham, "the leading atomist of his generation," but before his attention was specially directed to it a mass of experimental evidence led him in 1863 to express views of singular interest in the beautiful paper‖ which embodied his "speculative ideas respecting the constitution of matter." In this paper he pointed out that in solids some of the molecules may still be in the liquid or even the gaseous condition, and his words are very definite. He says, "the three conditions (solid, liquid, and gaseous) probably always co-exist in every liquid or solid substance, but one predominates over the others. . . . Liquefaction or solidification may not, therefore, involve the suppression of either the atomic or

**Ann. de Chim. et de Phys.*, Vol. XVII., 1846, p. 221.
†*Comptes Rendus*, Vol. XCIII., 1881, p. 1074; Vol. XCIV., 1882, p. 26.
‡ *Bull. de l'Acad. Roy. de Belgique*, Vol. X., 1885, p. 204.
§ *Nature*, Vol. XLI., 1889, p. 14.
‖ *Phil. Mag.*, February, 1864; *Collected Papers*, p. 299.

the molecular movement, but only the restriction of its range." He subsequently, in 1866,* gave singular point to these speculations by his discovery of the penetration of solid metals by gases.

After Graham's death evidence as to the molecular mobility of metals came slowly. E. Wiedemann† showed, in 1878, that solid metals were not necessarily inert, as changes which involve atomic movement take place in bismuth-lead alloys, and the clear evidence thus afforded of allotropic change in the solid, recently formed the subject of some experiments of my own.‡

Confirmation of the accuracy of Graham's views as to the co-existence of liquid and gaseous molecules in a solid was afforded twenty years later by Professor W. Spring.§ In 1886 his admirable investigation on the solidification of alloys of lead and tin afforded him experimental evidence that in these alloys active molecular movement is continued after the alloys have become solid. He says, and it is well to quote his interesting words, "On serait porté à penser qu'entre deux molécules de deux corps solides il y a un va-et-vient perpétuel d'atomes,"

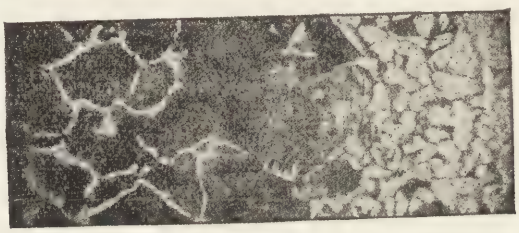


FIG. 4.

and he adds, "if the two molecules are of the same kind, chemical equilibrium will not be disturbed, but if they are different this movement will be revealed by the formation of new substances."

Formation of Alloys by Cementation.—The fact that alloys can be formed by the union of two metals at a temperature below the melting point of the more fusible of the two has long been known to metallurgists, and perhaps the most striking fact in the more modern history of the subject was recorded in 1820 by Faraday and Stodart,‖ who, in the course of an investigation on the alloys of iron with other metals, noted their failure to produce certain alloys by "cementation," but consider it "remarkable," in the case of platinum, that it will unite with steel at a temperature at which the steel itself is not affected. They also show that solid steel and platinum, in the form of bundles of wires, may be welded together "with the same facility as could be done with iron or steel," and they observe that on etching the surface of the welded mass by an acid "the iron appeared to be alloyed with the platinum." Their interest in this singular fact led them to promise some direct experiments on "this apparent alloy by cementation," that is, by the interpenetration of solids. Since this time there have been many more or less isolated observations bearing on the subject,

* *Phil. Trans. Roy. Soc.*, 1866, pp. 399-439.
† *Wied. Ann.*, Vol. III., 1878, p. 237.
‡ Second Report Alloys Research Committee, *Proc. Inst. Mech. Eng.*, 1893, p. 127.
§ *Bull. de l'Acad. Roy. de Belgique*, Vol. XI., 1886, p. 355.
‖ *Quarterly Journal of Science*, Vol. IX., 1820, p. 319.

and brief reference may be made to the more important of them in chronological order. In 1877 Chernoff* showed that if two surfaces of iron of the same nature be placed in intimate contact and heated to about 650° they will unite. In 1882 Spring† made his remarkable experiments on the formation of alloys by strongly compressing their constituent metals at the ordinary temperature, while in 1885 O. Lehmann‡ suggested, and in 1888 Hallock§ demonstrated that compression is not necessary, as alloys might be formed by placing carefully cleaned pieces of two constituent metals in juxtaposition and heating them to the melting point of the alloy to be formed, which was, in some cases, 150° below the melting point of the more fusible of the two metals. In 1889 Coffin showed, and I have repertedly verified the accuracy of his experiment, that if the freshly fractured surfaces of a steel rod, 9.5 mm. square, be placed together and heated to below redness, they will unite so firmly that it is difficult to pull them apart by hand. The steel is highly carburized and the diffusion of a carbide of iron probably plays an important part in effecting the union. There must have been molecular interpenetration in this case, though the steel was at least 1000° below its melting point. In 1894 Spring‖ proved that if the carefully surfaced ends of cylinders of two metals were strongly pressed together and maintained for eight hours, at temperatures which varied from 180° to 400°, interpenetration would take place, true alloys being formed at the junction of the two metals.

In these experiments, which are of great interest, the temperatures at which the cylinders were maintained were below the melting point of the more fusible of the two metals. Care appears to have been taken to avoid heating them up to the melting point of the *eutectic* alloy, though it was in some cases close to it. The necessity for this precaution will be obvious as the union of the two compressed cylinders might easily be effected by the fusion of an *eutectic* alloy with a relatively low melting point.

I observed in 1887 that an electro-deposit of iron on a clean copper plate will adhere so firmly to it that when they are severed by force a copper film is actually stripped from the copper plate and remains on the iron, thus affording clear evidence of the interpenetration of metals at the ordinary temperature. I found that this interpenetration of copper and iron will take place through films of electro-deposited nickel.¶ Mylius and Fromm have shown that metals interpenetrate and form alloys when they are precipitated by electrolysis from their aqueous solutions.**

The diffusion of metals in each other must be closely connected with the evaporation of solid metals or alloys at temperatures far below their melting points, and it will be well before describing the new experiments on diffusion in solid metals to briefly recall the facts which are already known. It is not necessary to go further back for definite views on the subject than to the time of Boyle,††

* *Revue Universelle des Mines*, Vol. I., 1877, p. 411.

† *Ber. der Deutsch. Chem. Gesell.*, Vol. XV., 1882, p. 595.

‡ *Wied. Ann.*, Vol. XXIV., 1885, p. 1.

§ Communicated to Phil. Soc. of Washington. Feb. 18, 1888; *Zeitschr. Phys. Chem.*, Vol. II., 1888, p. 6, or *Chem. News*, Vol. LXIII., 1891, p. 17.

‖ *Bull. de l'Acad. Roy. de Belgique*, Vol. XXVIII., 1894, p. 23.

¶ *Journ. Iron and Steel Inst.*, Part I., 1887, p. 73.

** *Ber. der Deutsch. Chem. Gesell.*, Vol. XXVII., 1894, p. 630.

†† *Collected Works*, Shaw's Edition, Vol. I., 1738, p. 400.

who thought that "even such (bodies) as are solid may respectively have their little atmospheres," . . . "for" he adds, "no man, I think, has yet tried whether glass, and even gold, may not in length of time lose their weight."

Boyle's opinion was correct, for mercury which has been frozen by extreme cold does, as Merget* showed two centuries later, evaporate into the atmosphere surrounding it; a fact which is of much interest in connection with Gay Lussac's well-known observation that the vapors emitted by ice and by water both at 0°C. are of equal tension. Demarçay† has proved that in vacuo metals evaporate sensibly at lower temperatures than they do at the ordinary atmospheric pressure, and he suggests that even metals of the platinum group will be found to be volatile at comparatively low temperatures.

Thus he finds that cadmium volatilizes at 160°, zinc at 184°, and lead and tin at 360°, and subsequently Spring‡ (1894) demonstrated that zinc is volatile at atmospheric pressure at about 300°, cadmium at about 400°, while even copper is slightly volatile at the latter temperature.

Moissan§ has stated quite recently that the vapor tension of solid silicon enables it to unite with iron and chromium by true "cementation" at a temperature of 1200°, which is much below the fusing point of these metals.

It must be borne in mind that the interesting facts recorded by the various experimenters whose names I have cited hardly come within the prevailing conditions in the ordinary diffusion of liquids, in which the diffusing substance is usually in the presence of a large excess of the solvent, which is supposed to exert but little chemical action on it. This condition has been fully maintained in my own experiments on the diffusion of liquid metals. It must also be remembered that Van't Hoff‖ has made it highly probable that the osmotic pressure of substances existing in a solid solution is analogous to that in liquid solutions and obeys the same laws, and it is also probable that the behavior of a solid mixture, like that of a liquid mixture, would be greatly simplified if the solid solution were very dilute.

Nernst expresses the hope that it may be possible to measure by indirect methods the osmotic pressure of substances existing in solid solutions. I trust that the following experiments will sustain this hope by affording measurements of the results of osmotic pressure in masses of solid metals at the ordinary atmospheric pressure, and at a temperature at which it has hitherto been scarcely possible to detect diffusion in them.

The following experiments constitute, so far as I am aware, the first attempt to actually measure the diffusivity of one solid metal in another. It must be borne in mind that the union of two clean surfaces of metal, and even the interpenetration of two metals to a slight depth below the surfaces, does not necessarily depend on diffusion alone, as the metals become united in a great measure by viscous flow. The nature of welding demands investigation, but the union of metals by welding is effected most energetically when the metals are in the col-

* *Ann. de Chim. et de Phys.*, Vol. XXV., 1872, p. 121.
† *Comptes Rendus*, Vol. XCV., 1882, p. 183.
‡ *Loc. cit.*, 1894, p. 42.
§ *Comptes Rendus*, Vol. CXXI., 1895, p. 621.
‖ *Zeitschr. Phys. Chem.*, Vol. V., 1890, p. 322.

loidal condition, in which true diffusion is least marked. It may be observed that discs of gold and lead, pressed together at the ordinary temperature for three months, were found to have welded together more perfectly than two similar discs kept in contact at 100° for six weeks, although at least ten times more metal had interdiffused in the latter case than in the former.

Diffusion of Gold in Solid Lead.—The attempt was first made to ascertain whether diffusion of gold in solid lead could be measured at a temperature of 250°, that is 75° below the melting point of lead. With this object in view, thin plates of gold were fused on to the end of cylindrical rods of lead, 14 mm. in diameter and 7 cm. long. This could readily be effected by the point of a blow-pipe flame, and, when the cylinder of lead was kept cool by immersion in water to within a few millimeters of its end, the gold rapidly alloyed with the metal, but, as many analyses showed, did not penetrate the cylinder of lead more than a millimeter. Such cylinders were maintained for 31 days in a little iron chamber lined with asbestos, the temperature of which only varied by a degree or two from 250° C. The cylinders were then measured and cut up into sections, and the amount of precious metal in the respective sections was determined by analysis. It should be observed that gold and lead are singularly well adapted for such an investigation, as the amount of precious metal diffusing can, by the aid of a delicate balance, be determined with wonderful precision.

The results of calculations of the diffusivity of gold in lead at 251°, 200°, 165°, and 100° is shown in the following tables, two experiments being given at each temperature:

GOLD IN LEAD AT 251°.

Number of Section (Counting from Base).	Weight of Alloy in Grams.	Weight of Gold in Grams.	Gold Per Cent.	Diffusivity in Square Centimeters Per Diem.
I.—1	26.9	0.0824	0.3064	.....
2	22.2	0.0053	0.0239	.....
3	19.6	0.0015	0.0076	0.023
4	22.3	0.00013	0.0006	.....
5	24.6	0.00002	0.0001	.....
II.—1	2.83	0.1803	0.3700	.....
2	2.55	0.0020	0.0784	.....
3	3.12	0.0013	0.0417	.....
4	2.84	0.0006	0.0211	0.03
5	2.99	0.00026	0.0087	.....
6	5.92	0.00013	0.0022	.....
7	6.03	0.00001	0.0002	.....
8	4.40	.....	.....	.....

GOLD IN LEAD AT 200°.

I.—1	1.50	0.2147	14.3	.....
2	1.05	Lost.	.....	.....
3	2.00	0.00055	0.0275	.....
4	2.14	0.000185	0.0086	0.007
5	2.18	0.000022	0.0010	.....
6	3.95	Trace.	.....	.....
7	4.60	None.	.....	.....
II.—1	0.39	0.00082	0.210	.....
2	0.49	0.000140	0.029	0.008
3	0.82	0.000187	0.023	.....
4	2.12	0.000220	0.010	.....
5	2.50	0.000040	0.002	.....

GOLD IN LEAD AT 165°.

I.—1	0.64	0.00025	0.039	.....
2	2.33	0.00069	0.030	0.005
3	2.02	0.00030	0.015	.....
II.—1	0.80	0.00031	0.039	.....
2	2.06	0.00060	0.029	0.004
3	1.40	0.00027	0.019	.....

GOLD IN LEAD AT 100°.

I.—1	0.22	0.00013	0.059	.....
2	1.19	0.00006	0.005*a*	0.00002
3	1.98	.....	.....	.....
II.—1	0.22	0.00034	0.155	.....
2	1.52	0.00010	0.007	0.00002
3	1.43	.....	.....	.....

(*a*) It may be thought that 0.005% of gold is rather a small quantity to measure with accuracy, but it represents no less than 1½ oz. of gold per ton of lead, an amount of precious metal far in excess of that which assayers are accustomed to deal with in the valuation of auriferous commercial lead.

Gold in Lead at 251°.—The total length of the cylinder was 6.5 cm. in the case of Experiment I., and 7.5 cm. in No. II.; fragments of gold were fused on their lower ends. Time, 31 days.

In calculating the diffusivity in the experiments, the initial concentration of the solid lead-gold alloy from which diffusion starts was deduced from the general trend of the concentration curve as plotted from the preceding figures. The first section in each case, which of course contained pieces of partially alloyed gold, was neglected.

Gold in Lead at 200°.—The next experiments were made at 200° and only lasted 10 days. The results of two of these are given. In No. I. the cylinder of lead was 2.5 cm. long, and had a plate of gold fused on to its end. In No. II. the cylinder was 1 cm. long, and the plate of gold was merely held against the carefully surfaced end by means of a binding screw.

There are many other results of similar experiments at 200°, which closely agree with those given. There is, however, one point that must not be lost sight of. The lead cylinder was solid, but the experiments just described were conducted at a higher temperature than the melting point of the *eutectic* alloy of gold and lead, which melts at nearly 200°. It may be that directly gold diffuses into lead it forms an alloy which is fluid at the temperature of the experiment and, therefore, though the lead itself is solid, it may, nevertheless, contain the gold in the form of a fluid alloy which diffused as such. Great care was consequently taken to ascertain whether gold would still diffuse in solid lead at a temperature well below the melting point of the *eutectic* alloy (200°), and for this purpose the temperature chosen was 165°.

Gold in Lead at 165°.—The length of the cylinder in No. I. was 0.64 cm., and it was maintained at this temperature for 30 days. The length of the cylinder in No. II. was 0.60 cm., and it was heated for 29 days. In each case diffusion started from a 5% alloy of gold and lead, pressed against the end of the lead cylinder,* and there was no need, therefore, to reject the first section of the cylinder.

*This precaution was necessary, as the diffusion must be very slow, and it was important to avoid any complication which might arise if the cylinder were heated by fusing gold on to the lead. At the conclusion of the experiment the gold alloy was simply detached by breaking it off the lead cylinder.

Gold in Lead at 100°.—In the concluding experiment of the series, the lead cylinders, 0.45 cm., were maintained in a water oven at a temperature of 100° for 41 days. Each cylinder had a plate of very pure gold pressed against its surfaced end.

It is remarkable that gold placed at the bottom of a cylinder of *solid* lead, 7 cm. long, should, at 250°, appear in notable quantity at the top of it in less than a month. The diffusivity of gold in *solid* lead is, however, slow when compared with diffusion in the *fluid* metal. The diffusivity of gold in lead at 500° was shown in the first part of this paper to be 3.0. In *solid* lead, on the other hand, the diffusivity at 250° is 0.03. It is also clear that gold will diffuse into solid lead at the very moderate temperature of 100°, which is 225° below its melting point, a fact which must be considered to be remarkable, and one the existence of which has hitherto been unsuspected. The diffusivity is, however, only $\frac{1}{100,000}$ of that which occurs in fluid lead.

Welding of Gold and Lead, and Diffusion of Gold in Lead at Ordinary Temperature.—The fact that two clean surfaces of lead will weld together at the ordinary temperature is well known. It may be well, however, to state in con-

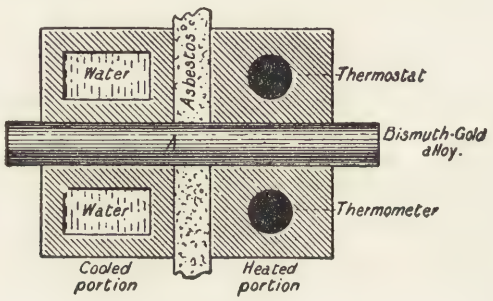


FIG. 5.

nection with the experiments of Demarçay* on the volatilization of metals in vacuo at comparatively low temperatures that if the ends of small bars of gold or silver be surfaced, pressed against lead, and maintained for 4 days in vacuo at a temperature of only 40°, the interpenetration of the two metals will be so complete that their separation can only be effected by the application of a load of 70 lbs. per sq. in. of the sectional area of the bars, or no less than one-third of the breaking strain of lead.

It remains to be seen whether diffusion can be measured in solid lead at the ordinary temperature, and, with this object in view, cylinders have been prepared and set aside for future examination.

In searching for evidence of diffusion in solid metals at the ordinary temperature, it will be well to examine certain alloys used in art metal-work by the Japanese, who often employ an alloy of copper containing a small proportion of gold, called *Shakudo*, which is soldered or welded in alternate layers with pure copper. The gold in the copper enables it to assume a beautiful purple patina when it is treated with suitable pickling solutions, which leave the pure copper of a red color. In this way very singular banded effects are produced. Many of

*Loc. cit.

the specimens are centuries old, and I have attempted, by grinding away the existing patina and re-pickling the surface, to ascertain whether the widening of the colored bands would show that, in the course of time, gold had diffused from the *Shakudo* layers and had passed into the copper. I believe that there is evidence that it does do so, but the inquiry is full of difficulty, and needs training in micrography, of which my friend, M. Osmond, is a master. We propose to study this part of the subject together, and I only allude to it here because, if diffusion occurs in copper, silver, and gold at the ordinary temperature, its results should be revealed in the products of this ancient oriental art.

Diffusion of Gold in Solid Silver.—A short cylinder of silver, containing 20% of gold, was pressed against the carefully surfaced end of a cylinder of pure silver, 1 cm. in diameter and 2.5 cm. long. These cylinders were kept in an annealing furnace for 10 days, at a temperature which never exceeded 800°, and was, therefore, 160° below the melting point of silver (960°). The lowest melting point of the gold-silver series of alloys is 850°. The cylinder was then cut into sections in the usual way, and the amount of diffused gold determined by analysis. As the temperature was intermittent the true diffusivity cannot be taken, but I am satisfied that the diffusivity of solid gold in solid silver at 800° is of the same order as that of gold in lead at 200°. It would appear, therefore, that the melting points of the metals have a dominating influence on the resistance offered to diffusion.

Diffusion from a Hot to a Cold Portion of Solid Metal.—It is well known that, in the liquid diffusion of salt solutions, osmotic pressure will drive molecules from the hot to the cold portions. Professor Thorpe suggested that an experiment of this nature should be tried with solid alloys. The principle of "reflection," which would, of course, be involved in such an experiment, has already been alluded to under the liquid diffusion of metals. The difficulty is to obtain a solid alloy of uniform composition, but, after many experiments, a rod of bismuth was obtained, in which it was believed that 3.75% of gold was uniformly distributed. This was arranged, as shown in Fig. 5, which represents, in sectional plan, a double block of brass, enabling half the rod to be heated to 170°, which is below the melting point of the bismuth-gold *eutectic* alloy, while the other half can be cooled by a stream of water. The heating was maintained for 6 days. The result proved that there was a distinct concentration of 0.1% gold at the point *A*, where the bar entered the cooled chamber. Similar results have been obtained with gold-lead alloys, and, if the bar be maintained at 240°, which is above the melting point of the *eutectic* alloy, the effect is very marked. I offer these statements with some reserve, as they require confirmation.

It will be evident that attention must now be devoted to the measurement of atomic movement in solid metals and alloys, for there can be no question that such movements play an important part in all structures in which metals are employed.

NOTE.—The data now published form but a small portion of the investigation which has long been in progress. The manipulation it involved was singularly difficult and tedious, and it would probably have been far less advanced than it is if I had not had the advantage of the aid of one of my former students at the Royal School of Mines, Mr. Alfred Stansfield, B.Sc. He made the calculations which were necessary to extend Stefan's tables to the range covered by the present experiments, and has shown untiring interest in conducting the work which has been undertaken during the past two years.

THE MICROGRAPHIC ANALYSIS OF METALS.

By J. O. Arnold.

The fact is now generally recognized that metals practically identical in ultimate chemical composition may under varying conditions of thermal treatment possess very different mechanical properties. Ordinary chemical analysis thus failing him, the metallurgist has been compelled to seek for other means of scientifically explaining the observed mechanical phenomena. The variation in the properties of given samples of metal of like composition may be due either to a formation of new compounds by the elements present in an alloy, or to a structural rearrangement of the same compounds. To ascertain the nature of the change first named an endeavor must be made to isolate the elements, not as such, but in the actual compound in which they exist in the metal under examination. Structural changes must be determined by the microscope. Of course both changes may exist together, and the problem often becomes most complicated in its nature.

The science of metallography is not of very recent birth. It was founded 35 years ago by Dr. Sorby of Sheffield, and the whole of his beautiful preparations of steel and iron sections (now in the microscopical laboratory of the Sheffield Technical School) seem to have been completed prior to 1864. The wonderful and patient work of Sorby was received by the world of practical iron and steel metallurgy with indifference if not with contempt, but it finally attracted attention in Germany, and in 1878 it was evident that Professor Martens, a distinguished microscopist, was exploring the new ground, while since 1890 various able papers have proved that metallography has taken root internationally. Sauveur in America, Andrews and Stead in England, Charpy and Osmond in France, Wedding in Germany, and Behrens in Holland are among the most distinguished laborers in this rich field.

Proceeding to a practical consideration of the microscopy of metals, it may be remarked that the science has almost nothing to do with the examination of fractures. All metals are really artificial crystallized igneous rocks, and as such must be polished to as nearly as may be a dead smooth surface, which is then etched so as to throw up as distinct areas the constituents forming the mass. There are many methods of preparing micro-sections, which of course are all examined as opaque objects, and therefore require for such examination special reflectors.

The Microscope and Accessories.—Any microscope may be adapted for examining metals. It is conveniently fitted with two eyepieces, say No. 1 and No. 3;

with two objectives, viz., $\frac{1}{6}$ and 1 in. For the former the illumination which has on the whole given the most satisfactory results is the high-power reflector of Beck, which is merely a short tube provided with a plain glass disk (a half-inch cover-glass serves well); this disk is fitted between the body of the microscope and the objective, the latter acting as its own condenser. When rays of light passing through an opening in the reflector-tube strike the glass disk (which is inclined at an angle of about 45°) they are reflected down through the objective to the surface of the section, up again through objective, disk, and eye-piece to the eye. In this simple manner brilliantly lighted fields are readily secured at a magnification of 600 linear. For low-power work the inch objective is fitted with the Sorby-Beck reflector. This consists of a parabolic mirror giving oblique illumination (under which brilliant constituents seem black) and a plane mirror inclined at an angle of 45° for direct illumination. By means of a little pivot the two kinds of light may be at once changed one for the other. The whole apparatus, including microscope, may be obtained for about £6. Where the conditions are favorable daylight may be employed, but usually a large Argand flame is the best source of illumination.

Preparation of Samples.—Of course in practice sections of various shapes may have to be dealt with, but when possible it is far better to prepare standard sections. Circular plates 1 in. in diameter and $\frac{1}{8}$ or $\frac{1}{10}$ in. thick are convenient. These should be turned or shaped by an experienced mechanic, being finally finished on a dead-smooth file and by means of emery-cloth to an accuracy of $\frac{1}{1000}$ in. This is important, as will be seen later on. The samples are now ready for polishing. There are several ingenious machines for this purpose, notably that of Mr. J. E. Stead; but certainly for standard sections for comparisons in the preparation of which time is not a vital factor, hand-prepared samples are not to be excelled. There are several methods of preparation, and, without claiming it the best, the writer in many years' work has found the following to be a very good one. To carry it out the following articles and materials are required: Three cast-iron taper blocks, 4 in. diameter at top, $4\frac{1}{4}$ in. at bottom, and 2 in. deep. The upper surface is machined to smoothness. Over the blocks slips a cast-iron ring, $\frac{1}{2}$ in. deep by $\frac{3}{4}$ in. wide, tapered inside so as to fall to a depth of $\frac{1}{2}$ in. on the tapered block. Thus on these blocks can be tightly stretched by means of the wedge-rings the various polishing materials. In addition to the blocks a brass plate $\frac{1}{2}$ in. thick, machined on one side and rather larger than a sheet of emery cloth, will be required. On this plate is tightly cemented a sheet of emery cloth. With this by means of *right-line* rubbing the file-marks are removed from the section. The latter will still be covered with a regular series of small scratches from the emery cloth, and there follows a rubbing on No. 1 block covered with 00 emery cloth, which substitutes for the somewhat coarse scratches of the 0 emery cloth those of the finer cloth. The piece under preparation should be from time to time revolved through an angle of 90°, so as to make the second series of scratches at right angles to the first. Passing to No. 2 block, this is covered with 00 emery cloth from which all detachable emery has been removed by means of a steel plate. On this block the small scratches from No. 1 block are removed, still maintaining the right line and never using a circular polishing motion. The piece is now fairly smooth and is ready for No. 3, or the final block.

(Of course where much work is in progress No. 1 block is in course of time promoted to No. 2 block, when it has become quite free from loose emery grains.) Over No. 3 block is stretched the finest selected kid-skin of even thickness and lightly charged with the finest jeweler's rouge. On this block (which must be kept under a bell-jar when not in use) all trace of scratches are removed by *circular* polishing. The section is then ready for etching if necessary. There are, however, certain structures which do not require to be etched, this process having been delicately and automatically effected by the combined influence of the fine rouge and the chemicals used in preparing the kid-skin, traces of which seem to be invariably present.

The foregoing process is very simple in description, but the student will find that it requires a vast amount of patience and no little experience to prepare a really perfect sample. Sometimes on getting to the last block but one, when the section appears all but ready for No. 3 block, the disgusted operator will find that suddenly a series of rather deep scratches appear without any apparent cause. This point will be explained later on.

Etching.—For the process of etching the writer uses very dilute nitric acid. If wrought iron or mild steel be under examination, an etching fluid composed of 49 c.c. of distilled water and 1 c.c. of nitric acid of sp. gr. 1.20 may be employed. If hard steel be the subject of investigation, one-fourth the proportion of acid will be sufficiently energetic as a rule. But here again experience will be required to get the best result. The process of etching is very like the development of a dry photographic plate, and its progress must be carefully watched with a large and powerful hand-lens. The process, it should always be remembered, is not one of corrosion or mere chemical solution, but a *galvanic action* due to the various crystalline constituents assuming opposite electro-positions. Substances acting as cathodes are practically unacted upon, while anode constituents are rapidly dissolved, their areas forming depressions distinct from the apparently raised cathode areas. Thus, regarding the section as a positive photographic plate and the dilute acid as a developer, constituents unacted upon (cathodes) become "high lights;" substances most violently acted upon (anodes) become "shadows;" while often constituents slightly dissolved represent the "half-tones." In carrying out the etching process *finger marks* on the polished surface of the section are quite as fatal as on a photographic plate.

A convenient etching dish is of glass $\frac{1}{2}$ in. deep, 3 in. long, and 1 in. broad. In this the polished sample is placed, and is well flooded and washed under a jet-tap with water to remove dust and air-bells. The piece should never again come into contact with air during the etching process, but be kept covered with a little water. The latter is next displaced and substituted by 50 c.c. of the etching fluid, rapidly poured in. When it is judged that the etching is complete the acid is displaced and the sample well washed by a stream of water from the tap. The washed piece is then quickly transferred to another etching trough containing strong methylated spirit. After soaking for a few minutes it is dried by pressing it face downward on a flat folded handkerchief of old linen. It is sometimes advantageous to rub the dry section very gently for a few turns on the kid-skin block, but it should always be remembered that this treatment causes fine particles of rouge to settle in the depressed portions of the sections, so that the student

should learn to recognize such deposits and be careful not to describe them as structure.

Mounting.—For mounting there will be required glass slips 1 by 3 in., cover-glasses .006 by 1 in., Canada balsam in chloroform, and brass cells 0.1 by 1 in. The last are best machined exactly to size from a piece of cold-drawn brass tube in the lathe. A good turner will make 100 accurate to $\frac{1}{1000}$ in. in a day. Lastly, a pair of knife-compasses will be required to cut out paper labels which just drop into the annular space between the section and the inside of the brass cell.

The brass cells are mounted on the glass slips with the balsam, and the latter should be allowed to dry slowly without the aid of heat, otherwise the cement may become brittle and a sudden shock cause the cell to drop off. The cover-glass is also put on with balsam, and just clears the section so that with very high powers the section can be brought into focus. On the label are written the distinctive marks of the steel and also its treatment; for example, 201 A. T. E. means ingot No. 201, annealed, transverse section, etched. A brief code of marks will readily suggest itself, the details being of course kept in the record-book.

MICRO-CONSTITUENTS OF IRON AND STEEL.

The micro-constituents of steel the existence of which is generally recognized by metallographists are: (1) Pure iron (ferrite*); (2) carbide of iron, Fe_3C (cementite*); (3) iron containing suspended granules of Fe_3C (pearlite*); (4) iron containing laminated scales of Fe_3C (pearlite*); (5) iron containing hardening carbon (hardenite,* martensite,† sub-carbide of iron‡); (6) graphite or free carbon (temper-carbon §). Much difference of opinion exists among steel microscopists as to the nature of these constituents, and the student must refer to the various papers of the authorities quoted, wade through the heated controversy, and from the arguments draw his own conclusions. The writer's opinions on nomenclature are set forth in a criticism of a paper on "The Microstructure of Steel," by Mr. Albert Sauveur, recently read before the American Institute of Mining Engineers.

It will be observed that the constituents named have reference only to carbon and iron alloys, and it is necessary to study these in the practical absence of other elements to obtain a true base upon which to build up the science of steel metallography. The neglect of this vital precaution by certain workers has already led to deplorable confusion. In order to lead the student to a systematic course of study a few concrete directions on the examination of well-known commercial products will be given so as to practically illustrate the nature and distinctive features of the above constituents.

Swedish Lancashire Hearth Bar-iron.—Certain brands of Lancashire bars as made in Sweden (*e.g.*, DU, s) will represent the purest commercial iron, containing about 99.8% of Fe. Sections taken from ½ by 3 in. bars both transversely and longitudinally will provide most interesting preliminary studies. In portions of the section it is probable that crystals of almost chemically pure iron will be found. These crystals will be pure white, and their facets and angles will be very clearly marked by dark intercrystalline junction-lines. In other parts of the section,

* Howe. † Osmond. ‡ Writer. § Ledebur.

however, the iron crystals will be small and irregular, being mixed with dark granules of steel. It will thus be seen that though the mean carbon of the bar is perhaps only 0.05%, it is nevertheless not evenly distributed, the mixed areas often containing as much as 0.2% of carbon. The iron crystals in both longitudinal and transverse sections present the same characters, but a most remarkable difference appears in connection with the involved slag. When it is remembered that the total silicon in such bars, including both that involved in the slag and that in combination with the metal as silicide of iron, is only as a rule about 0.02 or 0.03%, the amount of slag seems strangely large and at once indicates the possible fallacy of planimeter measurements. The slag in the longitudinal section occurs in elongated bars varying in breadth and often having a semicrystalline appearance. In the transverse section, however, the end view appears as dots and dashes. It is therefore at once evident that while the slag-globules originally present in the bloom have been drawn out in the direction of the rolling, the iron itself has crystallized after leaving the rolls, otherwise the crystals would have an elongated appearance in the longitudinal section. The bars of slag give to wrought iron its fibrous-looking fracture from the fact that the junctions of the slag and iron form planes of rupture. A careful microscopical examination of a given sample of iron of this class correlated with a mechanical test will at once overthrow the idea generally held by engineers that the iron itself is fibrous. As a matter of fact, often the most crystalline iron will give the greatest elongation and present a very fibrous fracture. A crystalline fracture is due to faulty *intercrystalline* cohesion, the mass breaking along such lines of weakness before the crystals themselves have begun to elongate.

Steel Castings.—Sections prepared from a steel casting containing about 0.3% of carbon will teach very important lessons to the student of metallography. A glance at the unannealed section will at once dispose of the notion that mild steel is a homogeneous material. The structure will consist of about two-thirds white irregular iron crystals mixed with about one-third darker steel granules. The dark color of these areas is due to a deposition of carbonaceous coloring matter liberated by the action of the nitric acid. When examined at high powers (say 500 diams.) the steel areas will be found to be compound in their nature, consisting in fact of iron through which is held in suspension very small granules and irregular plates of Fe_3C. True (unhardened) steel, therefore, consists of an intimate mixture of iron and normal carbide of iron, the proportion of the latter (in iron and carbon steels) being about 13%. When the carbon present reaches about 0.9% the whole mass is true steel. The microscope therefore teaches an important theoretical lesson (which is found in practical work to be true), viz., that a casting containing 0.45% carbon consists of an intimate mixture of equal parts of irregular crystals of pure iron and true steel. It will be evident that a casting well under 0.45% C will be ductile, because it practically consists of iron in which grains of steel are suspended. Well over 0.45%, however, the structure consists of steel in which grains of iron are suspended.

A series of mechanical tests will show that the decrease of ductility noted between steels varying from 0.35 to 0.55% is altogether out of proportion to the difference in carbon. In fact the influence of structure is here shown with remarkable clearness. The mechanical distribution of sulphide of iron is often

remarkable in unannealed steel castings, and contributes to their brittleness. This question cannot now be discussed, because a brief mention of results, however decisive, would only lead to hasty and forensic criticism. It may, however, be remarked as a guide to students that the mysterious series of scratches which often suddenly appear during the polishing of a section are due to the falling out of particles of sulphide of iron from the section upon the block. The hard minute fragments then of course are rolled under the sample, scoring it sometimes very deeply.

On examining the annealed casting the change in structure is very remarkable. The dimensions of the crystalline grains of both the iron and the steel are found to be much larger than in the unannealed sample. The lines of demarcation are also much less sharply defined; in fact the steel areas are now more or less dovetailed into the iron because the Fe_3C in the steel areas is now distinctly laminated, each hard plate being separated by a broader interspace of iron. In a mild casting the rearrangement is accompanied by a marked increase in ductility; but it is a curious fact that above 0.65% of carbon annealing often *reduces* the ductility as measured by the elongation, owing to the carbide of iron plates being so thick and large that they form a series of lines along which rupture takes place before elongation sets in. The structure of an annealed steel casting is often best seen when the sample is unetched. In such a case the carbide of iron plates present themselves as brilliant white lines on a blue background of iron, the latter having upon it a film of oxide due to the action of the kid-skin on the last block.

Hard Tool Steel.—A very beautiful section may be prepared from a bar of pure tool-steel containing about 1.3% carbon. The structure will be found to consist of cells of "steel" in which the carbide of iron shows a decided tendency to form in fine laminæ rather than in grains. But round each cell of steel (and often in streaks in the cell) will be found well-marked investing cell-walls of "cementite" or surplus carbide over and above that necessary to saturate the iron and form true steel. In other words, the mass which was entirely steel at 0.9% of carbon has now begun to assume the characteristics of "white iron." It will then be evident why turning-tool steel is so much more "cold-short" than, say, sett steel. The latter is comparatively homogeneous; but hard steel consisting of cells cemented together with brittle Fe_3C has naturally a tendency to rupture under a sudden shock along the junction-lines of the walls and cells. The foregoing micromechanical fact is best exemplified in the converted bar before melting or shearing. Every steel metallurgist knows how readily hard converted bar is broken into small pieces for charging by the blows of a hand-hammer. A section of such material consists of steel cells so large and of cell-walls of "cementite" so thick that a very low-power objective, indeed often a hand-lens, is quite sufficient to show the structure very distinctly.

Mottled Iron.—A sample of pure mottled iron will form a valuable and striking standard sample. In it may be readily recognized certainly four and sometimes five of the constituents already tabulated. The graphite will show in long narrow dark scales, often arc-shaped. In the vicinity of the graphite will be found iron or "ferrite," formerly in combination with the now free carbon as Fe_3C or "cementite." The undecomposed portion of the latter being frequently in juxtaposition to the ferrite is readily distinguished with a little practice by its more

brilliant appearance. In it often occur, in dots and dashes forming a peculiar pattern, little dark areas of "steel" in which the carbide is quite granular. In addition large granulated or laminated areas of " steel " are always present.

Malleable Iron.—A bar of white iron (say ½ in. round) after annealing in ore (or indeed after going through the ordinary flame process) makes an interesting microsection. The circumference will be found to consist almost altogether of crystals of ferrite, the iron of course being impure owing to the presence of silicon. On passing through the zone of ferrite into the interior of the bar the structure will be still built up of impure ferrite crystals, but among these, in the form of irregular dots, will be involved numerous areas of graphite or " temper-carbon." Often in such sections " steel " areas are practically absent, the carbide of iron having almost totally decomposed into iron and free carbon. This decomposition is of course much facilitated by the silicon usually present; but as an example of how complete this dissociation may be even when silicon is absent, the following case may be quoted: A hard steel containing 1.78% of combined carbon and only 0.28% of total impurities other than carbon was found after a thorough flame annealing to contain 1.70% of graphite and only 0.08% of combined carbon. As might be expected, the change in the mechanical properties of the material was startling, as will be seen from the subjoined figures.

	Before Annealing.	After Annealing.
Maximum stress, tons per square inch	53.20	18.46
Elongation per cent in 2 inches	3.50	32.50

MICROSTRUCTURE OF GOLD.

Melt half an ounce of pure gold in an annealing-cup inside a larger covered graphite crucible. When thoroughly fused remove the whole from the furnace and allow the button to cool *in situ*. When cold very gently machine the metal into the standard size and shape previously described. The section is then polished as usual, but it is not necessary to get the very soft surface quite free from scratches. Place a mixture of 5 c.c. of *aqua regia* and 5 c.c. of distilled water in a 3-in. porcelain basin, and bring the liquid to boiling. Remove from the lamp and drop the gold section, polished face upward, into the hot acid. A brisk action at once takes place, and when the crystalline structure of the metal seems fully developed the acid is displaced by a jet of cold water, the section is well washed and dried, and mounted as usual. The large primary crystals may be examined at a very low power, and seem to vary from almost black to gold color. This variation is purely optical. If the section be examined at high powers, it will be found that each primary crystal is built up of a very large number of small secondary crystals, and that the orientation of the secondary crystals is different for each primary crystal Hence the faces of the small crystals may be at such an angle as to reflect the light almost entirely outside the objective; therefore such an area seems almost black On the other hand, the angle may be such as to reflect the whole of the light into the microscope, when of course the area appears in its natural color. A bar of pure gold can be bent backward and forward through an angle of 180° many times without fracture, so intensely ductile is the metal.

If, however, a similar bar of gold contain, say, 0.2% of bismuth, it will be found to be very brittle and will not bend at all, but snap short. A button of gold

containing 0.2% of bismuth may be prepared as before, and on examination will explain most clearly the extraordinary mechanical effect of the small portion of impurity added. It will be found that primary crystallization has been prevented, and that the gold is now in small irregular grains, very different from the large geometrical crystals formed on cooling the pure metal. But the brittleness of the mass is due to the fact that each primary grain is surrounded by an investing wall of a laminated brittle gold-bismuth alloy. The structure of the latter is not readily observed, because it is dissolved out by the acid, leaving out-of-focus channels, at the bottom of which, however, it is sometimes possible to observe the striæ of the alloy. Similar effects may be obtained with all metals of low fusion point, such as potassium, lead, and tellurium. If, however, 0.2% of silver be alloyed with the gold, the structure and mechanical properties cannot be distinguished from those of the pure metal. A third type of structure may be obtained by alloying gold with 0.2% of pure crystallized silicon, which must be added to the molten gold at a high temperature, the mixture being again strongly heated above the melting point. Such a section will reveal the fact that the silicide of gold does not exclusively arrange itself between the primary grains, but is also found permeating the interior of the grains forming peculiar antimacassar-like patterns.

MICROSTRUCTURE OF COPPER.

Pure copper, like gold, consists of primary and secondary crystals, and the effect of small quantities of impurity on the metal is also very interesting, and is moreover of great practical importance. It is known that a small quantity of bismuth not only renders copper very brittle, but also enormously lowers its electrical conductivity. A button of pure electrotype copper melted out of contact with air with, say, 0.5% of pure bismuth, yields a most remarkable microsection, which is prepared in the same manner as that described previously for alloys of gold. In the present case, however, the bismuth does not seem to form any alloy with the copper, but merely to dissolve in the molten metal, and on solidification segregates into investing membranes or sectional meshes around the primary grains of copper. On etching the section (unlike the gold alloy) the bismuth meshes are left in relief and at low powers appear as a network over the copper. At high powers, however, the remarkable fact will be evident that each cell-wall of bismuth presents near its center either a cleavage plane or a contraction fissure.

The few examples of the microstructure of metals briefly described in this article will render evident the rich fields of research waiting for the labors of the metallographist; but it cannot be too strongly stated that the microstructure of a metal teaches a lesson of very limited value unless it is correlated with the chemical composition and mechanical properties of the sample under examination.

PROGRESS IN ORE DRESSING IN 1896.

By Robert H. Richards.

The author has for some years been collecting material for a book upon ore dressing; making experiments, visiting dressing works, and searching the literature of ore dressing at home and abroad. He now embodies in the following paper a few of the ideas and conclusions which have been gained in this work and which will be more elaborately treated in his book shortly to be published.

We are now able to say that there are three conditions of settling in water which must be taken into account in the planning of mill work. They are "free settling," "hindered settling," and "agitation."*

Free settling occurs when particles of ore, for example galena and quartz, are settling together in water with ample space around them. The author has found that for small sizes, 10-mesh and less, particles will settle uniformly when the diameter of the quartz is from $2\frac{1}{4}$ to $3\frac{3}{4}$ times that of the galena, according as the quartz is 10-mesh or 60-mesh in size respectively. The pointed box acts very imperfectly upon this principle.

Hindered settling takes place when quartz and galena, for example, are falling through a rising current, but with the particles so close together that they collide and interfere with each other. Here, quartz and galena will be in equilibrium, that is, will settle together when the quartz has about 5.8 times the diameter of the galena. The jig-bed during the act of pulsion, or period of upward current, is a very perfect instance of this. The ideal separator is a perfect instance of this condition; the practical separator, however, in order to produce commercial results is generally run at such speed as to depart from the principle to some extent.

Agitation.—This word seems best to express the work of the vanner and of the keeve; both these machines certainly give a balance of grains, when in equilibrium, that has a greater ratio than in hindered settling, although neither the one nor the other is a perfect separator. For example, the vanner loses fine material in the tailings, and for a well-adjusted vanner there is doubtless a definite ratio in the tailings between the diameter of the largest quartz grain and the largest grain of galena. These ratios have not yet been studied, and therefore no figures can be stated.

* *Trans. Am. Inst. Min. Eng.*, Vol. XXVI., "Cycle of the Plunger Jig."

Transient Bins.—At Waverly, Nova Scotia, a bin-house with four large bins is placed between the mine and the mill. The total cost of this operation is roughly estimated as follows: Interest and depreciation on $500, invested at 10%, and 30 tons per day, per ton, $0.005; one extra man at $1 per day and 30 tons, per ton, $0.034; interest on one car and repairs on building and car at $50 per year, per ton, $0.005—total cost per ton, $0.044.

In shaft sinking, level driving, or stoping, the ore from each face of work is dumped into its own bin until this is full. Ten stamps are then set to work upon this batch and accurate mill returns are kept, including tons of ore milled; ounces of gold obtained; balance of profit or loss. In this way every piece of ground is tested by mill run, recorded on the mine map, and the latter so filled becomes an efficient means of laying out the month's work.

Coarse Crushing.—At some of the Lake Superior mines coarse preliminary Blake crushers have been adopted in addition to the finer crusher formerly used. Again, in certain gold mills a large sized Gates, or Comet, crusher has been adopted to take the place of a smaller-sized Blake crusher, the object sought being to substitute the smaller cost of crushing with jaw or spindle crusher for the expensive hand breaking by sledges.

Rolls.—Geared rolls seem to still hold their own for use as slow-moving roughing rolls. The swing-box for the movable roll is being received favorably for the fine high-speed belted rolls. Belted rolls with large pulley for direct belt and small for crossed belt are meeting with favor. The plan of having the fine and roughing rolls of the same size, so that the roll shells can be partly worn down on fine work and then sent to the roughing rolls until worn out, is favored. There seems to be a marked tendency in modern milling for increasing the number of rolls and consequently the gradation of the crushing; the slight increase in expense is much more than compensated for by the increased capacity of the mill and the reduction of slimes by the graded crushing. Rolls still hold their own as crushers of middlings, but where the minerals can only be severed or unlocked by fine comminution, the Huntington and Bryan mills and the gravity stamps are still competing for supremacy.

Hand Sorting.—In regions where labor is not expensive hand sorting with sledging may be resorted to with great advantage, saving the expense of crushing and washing the richer ore, and avoiding the losses in slimes. Again, in Western regions where the ore is rich and concentration by machines unavailable for certain reasons, hand sorting by sledging is usual. On the other hand, in localities where the price of labor is high, but mill work available, the tendency is to abolish the sledge entirely and to use larger crushers, but the advantages of hand picking may still be retained. The ore in its transit from the bin to the Blake crusher, or from the Blake crusher to the rolls, may pass through a chute or conveyor and from it different kinds of ore may be taken out. This form of hand picking costs so little per ton of picked ore produced that it is commending itself to mill men. For example, if mill expenses are 80c. per ton and a diligent man can assort enough good ore at 50c. per ton to earn wages more remunerative than day wages, the advantage to the mill of getting the smelting ore at this low figure, and of avoiding cost of crushing and the loss due to slimes, is so great that it needs no argument to support it. For example, suppose the

milling cost per ton of raw ore is $0.80 and the reduction by milling is two tons into one, then the milling expense on one ton concentrates is $1.60, while the cost on one ton of picked ore is $0.50. It would probably pay to pick even up to a cost of $1.60 per ton of assorted ore to make the other savings.

The Grizzly calls for attention in two ways, namely, its construction and its use. The bars used for grizzlies must necessarily become too weak to withstand the shocks to which they are subjected before the metal is half worn away. There are therefore two parts of the bar which should be considered separately, to-wit, the head which receives the wear, and the web which furnishes the strength. To some extent grizzly bars have been designed to waste the least metal in the worn-out bars per ton of ore treated, by having such special shapes as the thicker head and thinner web. A very ingenious bar is used at the Quincy and Tamarack rock-houses of Lake Superior; the head is made either of wrought or cast iron and the web of wood. It is found that a spruce web will easily last as long as an iron head, and when the bar is worn out there is therefore little left to be thrown away. One might go still further and have a permanent iron web and a replaceable head, but this has not yet been tried. All of these devices for economy may be dispensed with, however, when waste-iron bars are at hand, such as old stamp-rods, as at Angels' Camp, old rails, etc.

The grizzly is used to prepare for the crusher. There does not seem to be much need of it to relieve the crusher of the fines, for it is everywhere economy to have a crusher of greater capacity than the mill. It is an advantage, however, where the tendency to form slimes is to be avoided, and it saves some wear on the jaws.

The mills employing rolls, trommels, and jigs will be found to have generally adopted the preliminary grizzly, while those having stamps, plate amalgamation, and vanners are about equally divided. In these the insertion of a grizzly gives another apparatus to tend and repair, and at the same time it probably does not materially lessen the quantity of final slimes produced in the mill, but simply saves the wear on the jaw-plates.

Sieve Scale.—The discussion in regard to the sizes of the holes in the trommels has left mill men undecided as to what policy to adopt, but there appears to be a decided tendency toward close sizing before jigging. The practice appears to be to put in a sieve scale perhaps not quite so close as Rittinger's, namely, 32, 22.6, 16, 11.3, 8, 5.6, 4, 2.8, 2, 1.4, and 1 mm., but not nearly so far apart as the half of the above, namely, 32, 16, 8, 4, 2, and 1 mm. The mill men have been governed somewhat by the quantity of stuff passing at a certain point. If one jig was getting too much to treat, the simplest relief might be found in respacing the trommel sieve scale, making a smaller gap, and hence less material would be delivered at the point named. Theoretically considered, there can be no doubt, if there are no flattened and included grains, that with certain ores in which the difference in specific gravity between the valuable mineral and the gangue is great, there is no need of close sizing,* after the slimes have been set aside, further than as convenience may suggest. For instance, it may be convenient to put nut size, pea size, and sand size each on its own jig, and sieves might then be provided accordingly. On the other hand, included and flattened grains

* *Trans. Am. Inst. Min. Eng.*, Vol. XXIV., p. 485.

are found almost everywhere. We may say, therefore, that the chances are that a heavy ore like galena will be better concentrated after close sizing, and that a lighter ore like chalcopyrite or blende will certainly be so.

Again, from a theoretical standpoint only, we may say that if a 16-mm. grain of middlings with 50% quartz, 50% chalcopyrite is equal settling with a 22.6-mm. grain of pure quartz, it will go into the tailings with it. The same thing will be true with a 11.3-mm. of half and half grain, as compared with a 16-mm. grain of quartz, also with an 8-mm. grain of half and half, as compared with an 11.3-mm. grain of quartz, and so on down the whole of Rittinger's scale. This line of argument, the truth of which cannot be doubted, points to a regular sieve scale, but in practice we find that the mills of Colorado, Montana, and Idaho introduce another element which modifies the sieve scale and enables the mills to do good work even when the coarser sieves vary from each other by what is apparently much too great a difference, for they recrush and re-treat all the tailings, and so the included grains are not lost.

In practice, the zinc-concentrating mills of Joplin, Mo., depart more or less from close sizing. In some of these mills the ore is simply crushed to 15 mm. in size and without any preliminary sizing is put directly upon the jig; in others the sizing is as close as that of Colorado, Montana, and Idaho, but no attempt is made to send the tailings to be recrushed, because there is an extremely small amount of included grains present, a fact due to the large free crystallization of the blende. Finally, little or no attempt is made to save the fine sizes, partly from sharp competition between the miners, and partly from the buyers of ores, who discourage the saving of fines.

The Finest Trommel.—The size of the hole in the finest trommel has increased in diameter to 2, 3, and even 4 mm., owing probably to the influence upon Western mills of the Lake Superior copper mills, which crush to $\frac{3}{16}$ in. or 4.68 mm. A certain difference between the two methods, however, deserves to be discussed. The Lake Superior copper mills take the four spigots of their separators to their four jigs, with two sieves each, and run these beds sufficiently soft and loose to have their tailings practically cleaned from free copper. The hutches of these jigs are then taken to a lower floor where they can be treated slowly on 3-sieve jigs, and yield two rich copper hutches and one sand hutch, which is run a little freer to clean the tailings. These latter jigs are called finishing jigs. In the Western mills these finishing jigs are entirely omitted, and the spigots of the hydraulic separators are sent to 4-sieve Harz jigs, with the expectation of making up by the two extra sieves on the upper floor, which are rushed with work, for the three Lake Superior sieves on the lower floor, which are gently worked. It should also be added that the Western jigs have all the sizes of quartz on the additional sieves, while the larger grains at Lake Superior have gone into the tailings of the upper-floor jigs, and are therefore not present on the finishing jigs to hinder the separation of copper. For the three reasons, therefore, firstly, because of the smaller sieve surface, next, because the extra sieves are rushed, and finally, because they are clogged with the larger sizes of sand, the author is of the opinion that these mills have gone too far in the direction of enlarging the hole in the last trommel. It may also be said that the separator, next to the pointed box, is the weakest tool in the mill, because up to date no separator has

been built that does not give in its spigots a quality of rich mineral so fine that it will find its way into the tailings of the jig which treats it. This quality of ore would be saved if sent to the vanner or to tables where it belongs.

By adding one or two more trommels and bringing the last hole down to 1.5 mm., which is Linkenbach's limit, a considerable portion of the fine ore above referred to would be saved.

These fine trommels are, it is true, more troublesome to manage than the coarser ones, but the saving will more than compensate for the additional expense, and it may pay to go even finer than 1.5 mm. for the finest trommel. Another point gained will be the diminution of the hydraulic water of the separator, hence a lessening of the slime water and an increase of its chance of treatment. As further argument we may say that in Wetherell's dry magnetic concentrator he uses five trommels with 10, 16, 24, 30, and 50-mesh sieves respectively, all of which are below the usual Western sizes, and the calcite (2.6 sp. gr.) is jigged from the willemite (4 sp. gr.) of all these sizes, proving both the sizing and the jigging to be feasible. In his case, if he had jigged those sizes together, the result would have been a poor separation, while now it is a good one, and no special difficulty has been found in running these fine trommels. In certain Western mills the tailings of the separator jigs are treated as middlings and recrushed. This is only to be recommended as a method for unlocking the included grains, but the fine free ore in these tailings, which finer trommels would have diminished in amount, is more or less slimed and lost by this recrushing.

The Hydraulic Separator.—This instrument is fed with grains from the last trommel, varying from 2 to 4 mm. in size, or from the steam stamp with its $\frac{3}{16}$-in. or 4.68-mm. round holes, and in one instance $\frac{3}{16}$ by $\frac{3}{8}$-in. slots (4.68 mm. by 9.37 mm.).

A great deal of ingenuity has been exercised in devising separators, and a great variety of forms have been proposed. A few general principles may be laid down as follows: All forms have two currents of water. (1) The carrying current, which carries the grains rejected by one pocket forward to the next, and carries off the final overflow to the settling tank. This current is well provided for in all the separators. (2) The hydraulic or upward current, which weighs the sands and selects only those entitled by their settling power to enter the particular pocket or spigot in question. In regard to these currents certain definite general principles may be laid down. They all have a contracted area or hydraulic column in which the weighing is done, and it is true that the narrower these hydraulic columns are the better will be the work of weighing. On the other hand, a sufficient area will be required to discharge the sand. Hydraulic columns with large area for the weighing are very harmful; they lead to down currents in one part and up currents in another, to overcome which the mill man will use extraordinary quantities of water, thereby greatly diluting his fine slimes.

The hydraulic separator works on the principle of hindered settling.* If the hydraulic space in which the particles are weighed against each other is sufficiently high and narrow, so that down currents may be avoided, it may be made

* *Trans. Am. Inst. Min. Eng.*, Vol. XXVI., "Cycle of the Plunger Jig."

to rank as a fairly accurate scientific instrument, the diameter of grains of heavy mineral bearing an approximately definite ratio to those of the quartz. But in the hurry and rush of mill work the best separator will fall short of this condition, and more fine, heavy mineral will be found in the earlier spigots than belongs there, making the subsequent jigging more difficult.

The hydraulic separator is also a serious diluter of the fine slimes, rendering the handling of this material more difficult. It is also one of the weakest parts of the mills whose work is based upon trommels. On all of these accounts its range of sizes should be diminished to a minimum by inserting one or two more trommels.

Pointed Boxes and Distributing Tanks.—If the slimes are to be treated upon vanners, it is a common practice to use a long distributing tank with spigots at intervals to suit the vanners. The slimes are fed to this tank at intervals along its length in such manner as to feed all the vanners with similar material and to avoid grading the same; the overflow of this tank may go to another, and its spigots to more vanners, or else to circular tables. The final tank of this series will aim to rob the water completely of its sediment. If the slimes are treated upon circular tables, a series of pointed boxes will generally be used and have the advantage of giving a series of slimes which are roughly graded from coarser to finer, and in the past have been considered to be true sorted products, suitable for the series of tables. The last box will aim to remove all slime. This whole question of slime treatment is in its infancy and the near future should see great advances in it.

Coarse Jigs.—Jigs are used upon coarse sizes as large as 2 in. (50 mm.) diameter. These jigs have a height of tail-boards or a depth of the beds of from 6 in. down to 4 in., make from 80 to 120 pulsations per minute, and discharge the concentrates under side gates or gates on the tail-boards. They are driven by the plain Harz eccentric or by one of the accelerated motions given by the sliding block or elbow joint, but the plain eccentric is used much more extensively than the others. As a rule the coarse jigs have one and sometimes two sieves, and are adapted simply to quickly take out the pure ore and send their tailings back to the rolls, doing the work of hand sorting mechanically. An instance of the use of the Collom jig for concentrating pyrites which pass through a ¾-in. hole (19.05 mm.) and rest on a ½-in. hole (12.7 mm.) is found in the Province of Quebec. A special form of spring is utilized and lasts six months. The whole of the concentrates pass into the hutch, which is discharged periodically by opening a 6-in. gate.

Medium Jigs.—The trommel mills all employ eccentric or accelerated jigs, the former very much predominating. Side gates are generally used. Occasionally the hutch alone is relied upon for concentrates; in one instance lead bullets are utilized for bed material. These jigs are provided with two, three, or four sieves, more being used for the finer sizes. Their tail-boards are all from 4 to 5 in. high, the coarser sizes being the higher. The speed of pulsation is from 100 to 250 per minute. The sizes of the sieves range from 15 by 28 in. to 24 by 38 in. In the Joplin zinc district jigs with five or six sieves are used. The hutches are rejigged on another similar jig with finer mesh than the roughing jig.

Fine Jigs.—Separator jigs and jigs for working the recrushed middlings are of

two principal kinds: Mills employing trommels use the Harz jig or the accelerated jig, and, in one instance, the Evans jig, and the Collom jig is used in mills with steam stamps. The Harz jigs have beds or tail-boards about 4 in. high, and are run at speeds varying from 150 up to 500 pulsations per minute, while the Collom jigs have tail-boards 2½ to 3 in. high. The sieves, of which there are two, are generally 22 by 34 in., net size, and make from 120 to 150 pulsations per minute; they are run loose, catching the coarser concentrates upon the sieve, while the hutches all go to finishing jigs of the same pattern, only with three consecutive sieves. Of these, the first two are run tight for pure copper, the third a little looser, so as to free the tailings from copper. Certain Montana and Michigan mills employ grinders, rolls or stamps for recrushing the middlings. In some Michigan mills the concentrates from the second sieve of those jigs which treat the first and second spigots of the separator are skimmed off by hand and carried up to the stamp on the supposition that the steam stamp will comminute the particles still more and liberate the included grains of copper.

What has been said of the jigs which treat the products from the separator also applies to fine jigs for treating recrushed middlings.

Slime Treatment.—The mills which employ rolls and trommels use either vanners or circular convex slime tables for the treatment of the slimes, but the majority use the vanners, and some mills treat the middlings of the tables on vanners. The mills with steam stamps use the circular tables altogether. The opinion prevails on the Pacific coast that for coarser sizes the side shake is preferable, while for finer the end shake does better work; in Colorado, however, the end-shake vanner is employed for the coarser sizes. The corrugated belt has not been adopted, but a modification of it is finding great favor; it has a band 1 or 2 in. wide, of fine corrugations placed from 2 to 3 ft. apart, and running from edge to edge. This belt strengthens the claim of the side-shake vanner, as it does not seem logical to mount it upon one with the end shake.

Circular tables are made from 16 to 18 ft. in diameter and with distributors varying from the Evans 6-ft. wide fixed distributor down to the conical 3-ft. distributors rotating with the table. The slopes of the table vary from ¾ to 1½-in. fall per ft. Most of the tables revolve, although a few fixed tables with rotating launders are to be met with, and occasionally concave tables above convex tables are also found. Tables mounted two high on one shaft are common, and sometimes tables mounted three high are used, and in one instance a three-high group consisting of a concave rotating table above a convex rotating in the middle, and a convex fixed below is used.

In the author's opinion, this whole branch of concentration, including the pointed boxes or distributing tanks, the vanners and tables, is in a primitive condition. The only sources we have for information upon pointed boxes are Rittinger, their inventor, and Linkenbach; the former considered it to be a vast improvement over previous methods, but the latter finds it to be an unsatisfactory apparatus, while the author coincides with both these opinions, the fact being that Rittinger made a great step forward when he designed his "spitzkasten," but other parts of ore dressing have now advanced to so great a state of perfection that the slime methods appear very crude. Investigations are at present in progress to determine the right slope for a table, the proper quantity of water to be used, the

best method of feeding a pointed box, and the behavior of the currents in it. In a general way they show what a very imperfect instrument the pointed box is, even with the best design and feeding, and they also show the need of a new classification of slimes. We have Rittinger's *Mehl*, 1 to 0.25 mm.; and *Staub*, 0.25 to 0 mm. Kunhardt designates them as meal and pulp.

The author believes that the time has now come for a systematic division of slimes in vanner mills, and that the suggestion is offered by the work of the Kennedy mill and Gates' blanket plant of Jackson, Cal. The process is briefly as follows: The stamp sands, passing through a 30-mesh wire screen (probably about 0.262 in. or 0.6654 mm. in diameter), are well treated on a vanner; the tailings of the vanner next go to a hydraulic separator and the spigot is thrown away while the overflow goes to blanket tables. The concentrates of the blanket tables are finished on a little end-shake vanner. The measurements of the mill vanner are: Travel of belt, 55 to 63 in. per minute; slope, 3.45 to 4.19 in. in 12 ft.; side shakes, 206 to 216 per minute; belts, 4 ft. smooth. The pulp film on the belt is 0.2 in. thick. The hydraulic classifier is a short V trough, 18 in. wide at the top, sloping 60° to a lengthwise edge; below it is an extension downward, like the keel of a boat, 4 in. thick, which is also of a V shape, sloping 60° to a crosswise edge; at the foot of this again is a 2½-in. pipe 8-in. long and inside a hydraulic water-pipe ½ in. in diameter, with four vertical rows of ⅛-in. holes looking in four directions, the holes being 1 in. apart vertically; the spigot is at the bottom and is ⅜ in. in diameter. Four of these separators handle the tailings of the 100-ton mill. The blanket tables are 12 ft. wide, 10 ft. long, and slope 1½ in. per ft. The concentrates are hosed off once each hour. The little finishing vanner is 4 ft. wide and 8 ft. long; has smooth belt with end shake and little rollers every 6 in.; it slopes 1½ in. to the foot, and shakes ½ in. 240 times per minute; the belt travels 85 in. per minute, and the pulp film is 0.076 in. thick. The tailings of the Kennedy mill, amounting to 100 tons per day, assay $1 per ton, the blanket tables heads assay $30 per ton, while the finishing vanner concentrates assay $100 per ton, and weigh 1000 to 1200 lbs. per day.

Measurements of the diameters of the grains of final pyrites are now being made, and they indicate that the tailings of a well-run vanner will be found to have an approximately constant ratio between the diameters of the largest quartz grains, and of the largest ruling grains of pyrite (thowing out the few scattering larger grains). The author prepared a set of figures on this ratio for insertion in this article, but was obliged to withdraw them at the last moment because their reliability was questioned.

It is well to recur to the classifications suggested at the beginning of this paper, viz., "free settling," "hindered settling," and "agitation." The hydraulic classifier works on the principle of hindered settling. The principle of agitation is here suggested for the work of the vanner, and the reasons for this suggestion may be stated as follows: If the vanner on the one hand made tailings in which the ratio of the diameter of maximum pyrites grains to maximum quartz grains was no larger than the hindered settling ratio, then the spigot of Mr. Gates' hydraulic separator would throw away a large quantity of valuable pyrites, which, however, does not occur. If the vanner, on the other hand, saved the

finest slime pyrites, there would be nothing left for Mr. Gates' plant to do, a proposition quite as absurd as the former. Only the middle course then remains, namely, that the vanner when well run has a ratio between its ruling maximum pyrites and quartz that is much larger than that of hindered settling.

The Kennedy vanners and the Gates' blankets thus introduce an entirely new practice in the concentration of slimes, and one that needs a new classification, as they show us that there is a balance of equilibrium in the vanner treatment, and that a well-run vanner can save almost all the pyrites down to a certain fraction of the diameter of the quartz, but is powerless to save particles below that size. Are we not, therefore, justly entitled to make a new classification of slimes into coarse slimes, which can be saved upon a well-run Frue vanner;, and fine slimes which require blanket tables and finishing vanner for their treatment?

Furthermore, it appears the vanner is not the perfect instrument it has often been claimed to be, but has a limit at which equilibrium is reached between the largest grain of quartz and the largest grain of pyrites, at which point these two grains behave alike; this being so, we need a name for it, and the author suggests "Agitation."

These statements can, however, only be considered a first contribution to the subject and many measurements will have to be made before a final conclusion can be arrived at. But in mills with distributing tanks and vanners the classification into coarse and fine slimes can be made as above stated.

This classification cannot be made in mills employing slime tables and pointed boxes since there is no agitation under these conditions. But there are serious losses in slime table work which need careful consideration. An investigation of the best conditions of sorting slimes, and for treating the sorted slimes upon slime tables, is now being made by the author. Perhaps the most serious defect found in existing practice is in the fine ore which goes to waste in the tailings on the feed side of the slime table. Theoretically the pointed box delivers from its first spigot or box the coarser grains of quartz and galena, and each succeeding spigot yields finer sizes, the relation of the diameter of the quartz to the galena in each product being that of free-settling particles;* the table takes these particles, rolls the larger quartz to the circumference, retaining the smaller grains of galena until the wash-off jet carries them into the concentrates tank. On the other hand, practically, the pointed box and table depart very considerably from the above. Experiments with the former show simply that the maximum diameters of the minerals in the first product are larger than those in the second, the second than the third, and so on, each product having all sizes of galena and of quartz below their maximum diameters. Experiments upon the table show that definite angles of slope and definite quantities of water are required to wash definite sizes of quartz and galena to the best advantage, and if all sizes are put together on a table with one slope and one water quantity, the mill man can only aim to make an average treatment, which will save the most ore under the circumstances, and let the rest go.

In considering the means for bettering the practice we must distinguish between mills already in operation and new mills about to be constructed. In the

* *Trans. Am. Inst. Min. Eng.*, Vol. XXVI., "Cycle of the Plunger Jig."

former, if the mineral is of high value, a paying saving may result if the tailings from the feed side of the slime table, which contain this finest grade of ore, be fed to a Gates' classifier, the overflow to Gates' blanket tables, and the concentrates of these to a Gates' vanner. In these final concentrates will be found the missing fine ore. A much more satisfactory way for new plants would be to substitute for the very imperfect pointed box the efficient Gates' hydraulic classifier, making two spigot products and an overflow, and to feed these perfectly sorted products each to a table with adjustments suited to its size.

The quantity of water will be increased by this classifier, instead of being diminished by the spigots of the pointed boxes, as is the case in existing practice; hence, we must provide either an ample settling tank, to take out the last of the floating mineral, and send the products of its spigots to blanket tables, or, better still, a blanket plant large enough to handle the whole overflow of the classifier.

The plant to handle the slimes of the Kennedy mill, near Jackson, Cal., which stamps 100 tons gold ore and treats it upon 24 vanners, cost $2000.* A plant to handle the slimes of a 400-ton mill, using rolls and trommels, probably would not cost over $6000, and if the mineral is rich in precious metal the savings effected would probably pay for it in a very short time.

Settling Tanks.—The feeding and cross sections of settling tanks are very important and are not well understood. If the velocity of feeding gives a surface current it will continue more or less to the end of the tank, and the bottom will be stagnant, and might just as well be made of wood with holes to let the settlings out as of stagnant water. If the stream is directed vertically downward, near the feed end of the tank, it will hit the bottom, forming an inverted mushroom shape; part will follow the bottom, widening upward as it goes toward the end, part backing and rising up at the feed end, following the top out roughly to the end, and leaving a stagnant middle wedge. If a tank is long and narrow, with a noticeable current in it, it will not permit the finest particles to settle at all, for the bottom water comes to the top after having traveled a certain distance. All of these conditions call for square tanks, of medium depth, with a very perfect distribution from top to bottom of the feed end, with baffle boards to break up the currents, and perforated board, or upright slats, for the final equalizing of the whole current. The square form shortens the tank about as much as is safe on account of the difficulty of perfect distribution; a still shorter would be better if perfect distribution were possible. A depth of 2½ ft. is sufficient for a tank 10 ft. square, while for one 40 ft. square 4 ft. is enough. The shallow depth gives the particles a short distance to go to find repose. If the tank is to have spigots and discharge continuously, a set of 60° V's in the bottom, running across the current, will do this; the V's can have spigots of ⅛-in. pipe and so discharge a thickened pulp. Where ⅛-in. spigots are employed the water must be sifted through a 20-mesh sieve or a plate screen punched with 1-mm. holes to remove fiber, wood, and coarse fragments, in order to maintain free spigots.

The suggestions contained in this paper may be thus summed up: Increase hand sorting without the hammer where labor is high and the mineral breaks clean.

* *Mining and Scientific Press*, Oct. 3, 1896, p. 277.

Add more trommels and diminish the coarse separator or classifier work for saving losses now made in the tailings of the jigs treating separator products, and to diminish the water in the slimes.

Substitute an efficient, fine hydraulic classifier for the wasteful pointed box, for grading the slimes according to the law of free-settling particles.

Adopt tables which by slope, quantity of water, and rate of rotation, are suited to treat the carefully graded slimes with which they are fed.

Adopt an extensive blanket plan for the overflow of this fine classifier, with special, steep, rapid end-shake vanners for bringing up their concentrates to a high quality.

For vanner tailings, where the mineral is rich, and the vanner practice good, and the vanner is fed with whole pulp, adopt an after treatment, consisting of hydraulic classifiers to remove the larger quartz from the fine silt; of blanket tables to partially concentrate the fine silt, and of special vanners, as indicated above, to improve the quality of the concentrates.

Pay special attention to the feeding and construction of settling tanks.

TECHNICAL LITERATURE OF THE YEAR.

*Davis Crushing Rolls.**—These rolls are of massive design, the two side frames being parts of one casting. They retain all the good points of Cornish rolls, and are well housed and protected from dust. The adjustable roll is carried by swinging arms, which turn on a heavy steel pin held firmly in the frame; the part of the pin carrying the arm is eccentric with the part in the frame, and a partial revolution of the pin raises or lowers the arm, thus providing a perfect vertical adjustment for the roll shaft. The bearings are all of the ball and socket type, which allows perfect adjustment under all conditions. The arms of the adjustable roll are held in place by the tie-rods that connect with the springs, and the distance between the rolls is adjusted by means of right and left nuts on these rods. Upon the heavy forged steel shaft is securely mounted a tapering roll center which forms a solid support for the full width of the shell, which is of open-hearth steel, forged and rolled to shape, and held in place by draw-bolts. For longitudinal adjustment and wear there are placed on the roll-shaft, between each bearing and the center, two or more steel rings, which may be changed from one side to the other to shift the roll as desired. These rolls are built in four sizes, from 12 in. face by 12 in. diameter to 16 in. face by 38 in. diameter.

H. Van F. Furman† contributes an article upon crushing and pulverizing ores, which also discusses the qualities of the Blake, Dodge, and Comet crushers, as well as the Schranz mill, the Cornish, Rogers, and Cazin rolls, and stamps, and gives a good idea of the machines and of the work they do.

Effect of Forcing Rolls.‡—A great record in fine crushing has been made by the Metallic Extraction Company at Florence, Colo. For six months the average crushing was over 125 tons per day, with a record of 160 tons at times, crushed to pass a 40-mesh steel wire screen. The crushing was done by Gates crusher, Blake multiple-jaw crushers, and Krom rolls with rolled steel tires, making 100 revolutions per minute. The ore was a hard igneous rock, chiefly andesitic

* *Engineering and Mining Journal*, Feb. 15, 1896, p. 159.
† *Colliery Engineer*, November, 1896, p. 154.
‡ *Engineering and Mining Journal*, Aug. 15, 1896, p. 146.

breccia and phonolite, from Cripple Creek, with an average sp. gr. of about 2.5. This volume of rock crushed is extraordinary.

A curious fact was observed concerning a pair of rolls having a 14-in. face, driven at 100 revolutions per minute. After being overtaxed with grinding hard rock for several weeks they burred out ½ in. at each edge, that is, the faces were spread out to 15 in. wide, yet remained true.

The lack of data on the comparative hardness and toughness, or, better said, the strength or resistance to compression, the specific gravity, and the friability of ores, is deplored. Such data on typical ores would be of as much value to the metallurgist as data concerning flow of water in pipes is to the hydraulic engineer.

Stamp Batteries.—John Darlington,* of London, designed a wooden frame stamp battery constructed of small sections adaptable for mule transportation. Each stamp-head weighs 750 lbs., but no other piece weighs more than 400 lbs. The frames are of pitch-pine planks from 3 to 4 in. thick, with thin plates of mild steel between them, and similar plates on the outside, the whole construction being firmly bolted together. The mortar blocks are of square pieces of pine bolted together without steel plates. The cam-shaft pulleys are of wood 5 ft. in diameter and 10 in. face.

Edwin A. Sperry describes the use of the Tremain steam stamp with amalgamation.† It has two stamps in one battery, operated directly from the boiler, the steam being admitted beneath the piston to lift the stamp; the exhaust from this is admitted above the piston, and acting by expansion forces the stamp down to strike the blow. The piston stem and shoe weigh 300 lbs., but owing to the force of the steam they strike a blow equal to a 1000-lb. stamp, with a drop of 5 to 8 in. The screens are on three sides of the battery. The capacity is 12 tons in 24 hours. Of the amalgam 95% is caught on outside plates. The discharge is brought by chuck-block to 6 in. above the die. About 0.2 cord of wood is burned for one ton crushed. The shoes and dies wear off about ½ in. each for each 100 tons stamped, and the loss of mercury is 1 oz. per ton of ore treated.

A two-stamp battery with discharge on three sides is made‡ for prospecting and development work by the Joshua Hendy Machine Works. The stamps weigh 850 lbs., and drop 6 in. 110 times per minute. By having a discharge screen on three sides the discharge area is nearly equal to that of an ordinary five-stamp battery.

John Ross, Jr.,§ has given detailed costs of mining and milling at the Wildman and the Hector Mines, Sutter Creek, Amador County, Cal. His final figures are as follows:

	Tons Mined.	Cost of Mining.	Tons Milled.	Cost of Milling.
Wildman, 1895	31,591	$2.02	31,591	$0.364
Wildman, year ending Sept. 30, 1896	33,497	2.17	33,512	0.408
Hector, year ending Sept. 30, 1896	41,846	1.55	41,771	0.318

These figures include all expenses. The operation of the rock breakers and delivery to the mills is charged to mining.

* *Engineering and Mining Journal*, June, 1896, p. 541.
† *Trans. Am. Inst. Min. Eng.*, September, 1896; also *Colliery Engineer*, November, 1896, p. 158.
‡ *Mining and Scientific Press*, Nov. 28, 1896. § *Ibid.*, Dec. 26, 1896, p. 523.

Common bar soap is recommended* for lubricating stamp stems. Unlike oil or grease it would be beneficial rather than harmful in the case of amalgamation.

The Kinkead Waste-rock Experiment.†—J. H. Kinkead has erected a mill to work the waste dumps of the Comstock lode, at Virginia, Nev. The mill is a concentrating plant, with no new features, except that the Kinkead mill is used instead of stamps. This machine consists of an iron pan 5 ft. in diameter, with a convex bottom declining outwardly from the center, with vertical sides 6 in. high at right angles to the bottom. The muller weighs 2000 lbs.; a conical shoe, concave on the under side, is bolted to it; its concavity is slightly less than the convexity of the die. A shaft 6 ft. long, attached to the muller, is made to gyrate by a crank at its top on a short vertical shaft in the timber above the machine. The crank has a 10-in. throw, so that the muller gyrates around on the conical surface of the die. The rock, which must pass a ½-in. screen, is fed through the center of the muller, and is first broken up by the central part of the shoe which is in the form of inverted steps; the rock is then ground by the gyratory motion of the muller and discharged through a 40-mesh screen around the sides. The present experimental mill consists of two Kinkead mills, amalgamating plates, sizing boxes, and two 6-ft. plain-belt Frue vanners. Each mill will crush 15 tons of waste rock in 24 hours. A No. 2 Pelton motor (12 in.) with ¼-in. nozzle and 220 lbs. pressure actuates the whole mill.

Friedeberg Mill for the Fine Grinding of Coal and Other Material.‡—In the Friedeberg dry grinder a vertical shaft, passing through the floor of the grinding space, carries two radial arms, opposite to one another. Each of these arms is jointed in the middle and carries on its outer end three rolls with vertical axes, two of them being attached to the main arm by means of short side arms. When the shaft revolves the rolls are forced against a steel grinding ring. On the upper end of the shaft is a cone, which distributes the coal or ore from the feed-hopper to the rolls. If any very hard substance gets between the ring and rolls, the latter pass over it without harm on account of the joints in the main arms. The rolls also act as a fan to drive the pulverized ore up and through the screen above. With a 60-cm. ring the capacity is 200 to 300 kgms. of coal per hour through a 40-mesh screen, and with an 80-cm. ring about double that quantity, using 4 to 5 horse power per hour.

Size of Wire Cloth.§—Much difference exists in the wire gauges used by different manufacturers; but when, in addition to this, wire cloth of the same mesh is made with a variety of gauges of the same system, there is a decided lack of uniformity in screens of the same mesh. The aperture of 20-mesh cloth made of No. 24 wire (Roebling gauge) is 0.027 in. diameter. A 24-mesh with No. 25 wire has an aperture of 0.0217 in., but with No. 33 wire it is 0.0287 in. That is, 24-mesh wire cloth made of the finest wire used commercially, for this grade has greater capacity than 20-mesh cloth made of the coarsest wire.

The only rational method of rating wire cloth is according to the diameter of aperture, as is already the custom with punched steel plate. This could best be expressed in millimeters. The metric system of measurement should be adopted.

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* *Colliery Engineer*, December, 1896, p. 219.
† *Mining and Scientific Press*, Jan. 25, 1896, p. 61.
‡ *Colliery Guardian*, May 1, 1896. From a communication to Hanover Mining Congress.
§ *Engineering and Mining Jounal*, Sept. 12, 1896, p. 242.

The question of gauges,* which has been discussed for 20 years, was answered by the Railway Master Mechanics' Association in 1894, when they adopted the decimal system. As a result the decimal gauge has been adopted by most of the railroads of the country, and the American Steel Manufacturers' Association has endorsed it and recommended its general use.

The Gates Iron Works catalogue, No. 5, lists screen cloth by the actual measure of the opening, giving also the diameter of the wire. This is an important step toward helping the mill man to get what he wants.

Stamp Mill Screens.†—T. A. Rickard says, in his article on "Stamp Milling," that the tendency to use punched plates to the neglect of woven-wire screens seems a curious and unfortunate retrogression in milling practice. Nevertheless woven wire is destined to supersede punched or perforated iron plate, for they give conditions promoting uniformity of pulverization and a high crushing capacity, having a smaller area of solid metal for holes of equal diameters; it is urged, however, against their use that they are more easily choked and weakened by pulp and chips, and must oftener be removed for cleaning and repair, but as the ordinary cost for screens is only about 1c. per ton of ore this is a weak objection.

The Kreiss Transporting Trough.‡—This trough is of semicircular section and transports mineral products either horizontally or up an incline, which may be as steep as 12°, by a jerking motion. The forward motion is rapid while the return is slow, and the revolutions are 380 to 470 per minute. A trough 7 meters long and 0.38 meter wide, with a rise of 0.85 meter, moved 75,000 kgms. of tailings of 15 to 35 mm. diameter in 10 hours. A smooth bottom, especially when wet, prevents good work, and therefore the bottom is made with indentations 0.30 mm. long and 5 mm. deep.

"Vibromotor" Screen.§—The "vibromotor" screen is intended to do away with the jerking of vibrating screens operated by cams or cranks and with the consequent general jarring. The screen is hung from four ropes, instead of rods, and receives its motion from an unbalanced horizontal flywheel upon a vertical shaft set in the upper end of the screen frame. The flywheel is driven by a jointed rod and beveled gears from the general shafting.

Middle Product Jig.‖—E. G. Tuttle describes a jig for obtaining three products—heads, middlings, and tailings—from a single sieve. His sieve is 36 in. long, 36 in. wide, with a tail-board 13 in. high. His heading dam is 2 in., and his middling dam 7 in. high; they are adjustable, however, and are both 36 in. long—that is, the whole width of the sieve—and there are adjustable gates in front of both the dams, moving downward or upward to cut off or let on the discharge respectively.

Another paper, "Eccentric Jig with Adjustable and Automatic Lower Discharge Arranged for the Full Width of the Bed and for One or More Compartments," describes a jig with two sieves, with one automatic discharge on each.

The Woodbury side-shake vanner¶ is equipped with a Woodbury crimped-

* *Engineering and Mining Journal*, Oct. 31, 1896, p. 410.

† THE MINERAL INDUSTRY, Vol. IV., p. 324.

‡ *Berg- und Hüttenmännische Zeitung*, Nov. 27, 1896, p. 397.

§ *Engineering and Mining Journal*, March 21, 1896, p. 278.

‖ "With Adjustable and Automatic Discharges for the Middle and Lower Products," *Trans. Am. Inst. Min. Eng.*, September, 1896.

¶ *Mining and Scientific Press*, Sept. 19, 1896, p. 233.

edge rubber belt, pulp distributor, and adjustable cranks. The shaking frame is supported by four upright springs; and the side shake is given by adjustable cranks. The distributor consists of a slowly revolving bowl, discharging even quantities of sand and water through distributing channels to the whole surface of belt.

*Union Ore Concentrator.**—The Union ore concentrator is a side-shake belt machine having an iron frame, supported from above by four hangers. The shake is given by adjustable cranks, and the forward motion of the belt by cone pulleys and a chain. At the head end is a wooden roller intended to remove most of the heads into a small box before the belt passes into the wash-box.

The Ellis Concentrator.†—This resembles the ordinary traveling-belt machine; but the characteristic point is that the belt is filled with small batea or pan-like depressions, which the gyratory motion given to the belt causes to resemble a miner's pan. In practical tests, the 4-ft. machine has disposed of from 15 to 20 tons of ordinary pulp in 24 hours, and the 6-ft. belt twice as much. In a test of pulp carrying 30% zinc and 10% lead from a Utah mine, the galena was thoroughly separated from the zinc.

The Spadone vanner belt‡ has flanges on its two margins which are connected with the belt by flexible joints. These flanges stand erect except when the belt is passing over the rolls, when they for the time bend outward.

Fraser & Chalmers' pamphlets on "Browne Sizer" and "Frue Vanner" give instances where vanner concentration was improved to a marked degree by classifying the pulp previous to feeding it upon the vanner.

The common method of conveying water in cold countries§ by means of underground pipes, to avoid freezing, has been adopted at "O. K.'s" ten-stamp mill.

The pulp from the battery plates is treated by hydraulic classifier preparatory to feeding it upon the vanners.

The Accumulation of Amalgam on Copper Plates.‖—R. T. Bayliss describes briefly the treatment of a gold and silver ore at the "combination" mill of the Drum Lummon Mine, Marysville, Mont., and gives a more special description of amalgamating plates and routine. An apron-plate, rolled copper electroplated with 1 oz. of silver per sq. ft., after being in continual service for 3 years and 10 months, was recently taken out and the amalgam scale removed by striking the back and front of the plate with a hammer. Although this buckles the plate and makes it unfit for use, it is the best method found here to remove the accumulated scale, which is then retorted, after which the plate is melted to a bar, the gold and copper contents of which amount to more than twice the value of a new plate. One plate yielded $8340.54 in bullion, another more than $11,000. A remarkable feature of these observations is that the average fineness of bullion obtained from daily clean-ups was gold 541.5, silver 443.9, total 985.4; whereas the fineness of bullion from the accumulated amalgam was gold 431.4, silver 562.5, total 993.9; the total of the latter being greater, yet the gold fineness 110.1 less and the silver fineness 118.6 more.

* *Engineering and Mining Journal*, July 11, 1896, p. 29; *Mining and Scientific Press*, March 21, 1896, p. 221.
† *Ibid.*, Sept. 26, 1896, p. 297.
‡ *Mining and Scientific Press*, Dec. 5, 1896, p. 457.
§ *Canadian Mining Review*, December, 1896, p. 259.
‖ *Trans. Am. Inst. Min. Eng.*, February, 1896.

In the discussion of this paper results corroborating Mr. Bayliss' experience are given, but the use of the steel scraper for removing the amalgam scale is advocated. Mr. Bayliss replies that steel scrapers cannot be used at the Drum Lummon mill without injuring the plates, because of the hardness of the scale. He submits that we must look to chemical, and not to mechanical, causes for an explanation of the excessive accumulation of scale and of the variation in the relative value of gold and silver in the amalgam taken off the plates from day to day, as compared with that remaining on them in the form of scale.

*Concentration of Low Grade Iron Ores.**—Wm. B. Phillips briefly reviews his process of roasting for magnetization, described in *Trans. Am. Inst. Min. Eng.*, Vol. XXV., p. 399, and shows that complete magnetization can only be obtained with a bright red heat, and that from failure to get this or other causes he did not reach commercial results.

Dr. H. Wedding† contributes an article discussing the roasting of iron ores, both for direct use in smelting and preparatory to magnetic separation.

Magnetic Concentration of Iron Ore.‡—The ores treated were Clinton red fossiliferous soft and hard ore; the example cited below is the soft. The machine used was the Wetherill magnetic concentrator of the inclined pattern.§ The machine consists of two straight bar magnets, BB, B_2B_2 (Fig. 2), placed side by side and parallel to each other; they have cores of soft iron, 8.6 in. long, 10.75 in. wide, and 2.5 in. thick, wound with 950 coils each, A, A_2 (Fig. 2), of wire carrying 6

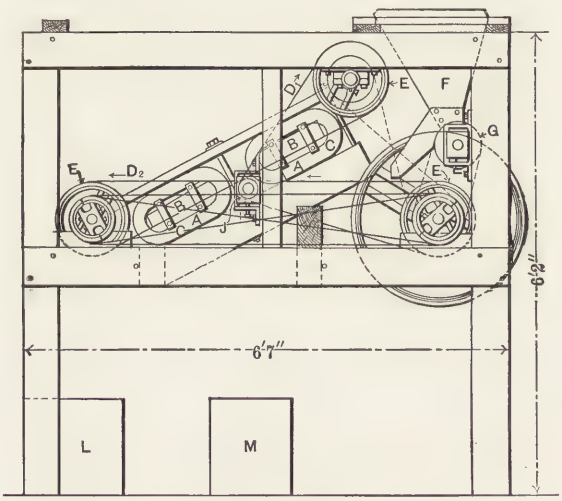


FIG. 1.

to 8 ampères with 52 volts per magnet. The pole pieces, BB, B_2B_2 (Fig. 2), or extensions of the cores, are 15.2 in. long, 10.75 in. wide, and 2.5 in. thick; the excess of width over that of the core, $RRRR$ (Fig. 2), being all on one side in the form of a lateral extension. The two lateral extensions of the pole pieces of

* *Engineering and Mining Journal*, July 25, 1896.

† *Journal Iron and Steel Institute*, September, 1896; *Stahl und Eisen*, Oct. 1, 1896, p. 771; *Engineering*, Oct. 9, 1896, p. 476.

‡ Wm. B. Phillips, *Engineering and Mining Journal*, Aug. 1, 8, 15. 1896, and notes by the author.

§ THE MINERAL INDUSTRY, Vol. IV., p. 733.

one magnet are placed adjacent to those of the other magnet, leaving spaces of 0.92 in. between them at each end, and the windings are arranged to bring the north and south poles together at each end, thus making a complete magnetic circuit with two air gaps of 0.92 in. in length, and an area of 0.92 by 10.8 sq. in. of intense magnetic field at each end of the pair of magnets, in which areas all

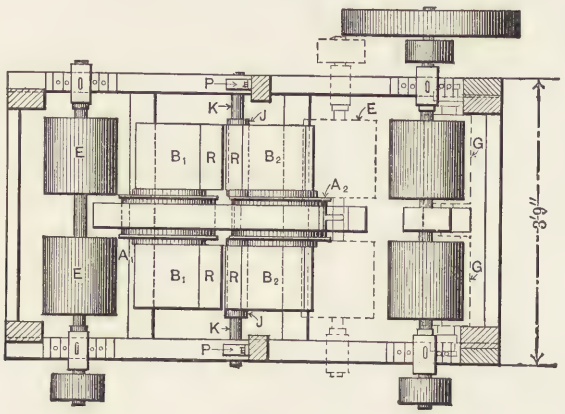


FIG. 2.

the work is done. The distance of 0.92 in. is adjustable and may be increased or decreased to give the best effect, by adjusting bolts, *CC* (Fig. 3), which admit a movement of the two pole pieces, *BB* (Fig. 3). The plane of each of the magnets in this machine is tilted at an angle of 27°, on a hinge, the line of which is parallel to the length of the magnets, and the plane of the upper magnet is 1.2 in. above the plane of the lower magnet. The machine has three belts of thin

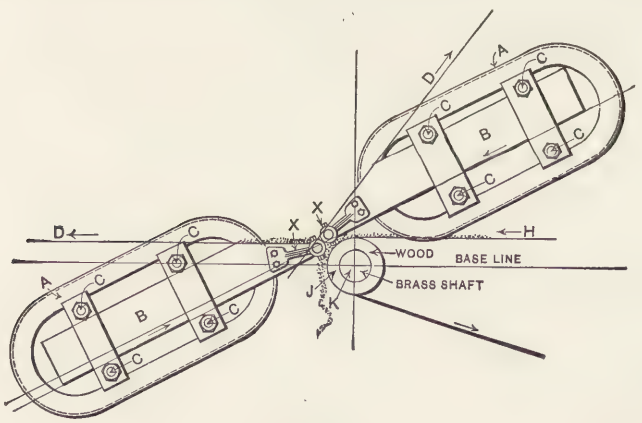


FIG. 3.

canvas at each end of the pair of magnets, *HD*, D_2 (Fig. 1), for the treatment of ore. The three have different duties to perform. No. 1 belt, *H* (Fig. 1), brings the ore in horizontally under the pole pieces of the upper magnet, and passing over a wooden roller, delivers it as nearly as possible to the field of greatest magnetic activity. The roll is adjustable, up and down, for obtaining

greatest effects. No. 2 belt enters into action along the under surface of the upper pole piece, rounds the tips, moves up the bevel, *D* (Fig. 1), of its wedge (which here has an angle of 54° with the horizontal), bringing with it magnetic particles, which, as they get above the magnetic field, tumble back down the slope, the belt moving on to its driving pully and returning again to its work. No. 3 belt enters its field of action on the under surface of the lower pole-piece, and rounding the tip, passes off in a horizontal direction, D_2 (Fig. 1), carrying with it all its own concentrates and those which have tumbled off No. 2 belt. Then rounding its driving pulley it delivers up its load of concentrates, and returns again to its work.

The ore treated was crushed through 15-mesh, and that portion which rested on 40-mesh was used for magnetic concentration. The portion which passed through 40-mesh contained 49.4% iron, and was considered sufficiently rich. The material through 15 on 40-mesh was subjected to three consecutive treatments—with 10, with 8, and with 6 ampères respectively. The heads of the 10 ampères went to the 8 ampères, the heads of the 8 ampères went to the 6; the tailings of the 8 and 6 ampère treatments are called final middlings, and the tailings of the 10 ampère treatment are called final tailings in the table.

	Weight.	Fe.	Insol.
	Per Cent.	Per Cent.	Per Cent.
Raw ore	100.0	39.20	40.16
Heads of 10 ampère	59.3	54.10	18.80
Heads of 8 ampère	55.0	54.10	18.70
Heads of 6 ampère	52.4	56.40	17.10
Final middlings	6.9	38.85	41.35
Final tailings	40.7	16.70	74.10

The shrinkage in weight by the 8 and 6 ampère treatments lowered the value more than the increased percentage of iron gained, and hence it was decided that the single 10-ampère treatment was the best commercial method. The hard Clinton ore was treated with almost similar results.

*The Concentration of Ores in the Butte District, Montana.**—Chas. W. Goodale gives his conclusions derived from 15 years of milling. He classifies the ores of the Butte district and gives the history of concentration there, which began in 1881. Descriptions of the following mills are given: Anaconda Mill of 1883 to 1886, Anaconda Concentrator of 1886 to date, The Butte Reduction Works, The Parrott Concentrator of the Parrott Silver and Copper Company, The Butte & Boston Concentrator, and the Colorado Concentrater of the Colorado Smelting and Mining Company, the latter being the latest in the district is described more particularly.

The figures quoted are strongly in favor of rolls over Huntington mills for crushing middlings, to-wit: At one mill the cost per ton of treating tailings by the Huntington mill is 8.8c., at another the cost by Cornish rolls is 4.6c.; the cost of power is not included in either case. Tests on the comparative working of Frue vanners and circular tables on Gagnon ore showed that vanners had the advantage. The merits of various belt machines, the best material for roll shells, and the cost of concentrating are discussed.

*Chas. W. Goodale, *Trans. Am. Inst Min. Eng.*, September, 1896.

In the discussion of this paper regarding "banks" on vanner belts, results of tests in favor of the Frue vanner, as compared with the Tulloch, are given. In sampling tailings John Carkeek strongly recommends that the whole stream of the tail-race should be diverted for a short interval of time to a settling tank and allowed to settle till the next sample is taken. The accumulated settlings make the sample.

*Kavan-Rittinger Table.**—C. Blömeke says that the Luhrig table is better adapted to treat the finer slimes than the Kavan table, and also has a greater capacity. A Luhrig table treated 213.3 kgms. (dry weight) of slime, middle product, in an hour, while a double Kavan table treated only 155 kgms. of fine sand in an hour. The Luhrig combination of four tables occupies 40% more space than the Kavan combination of three double tables, but its capacity is 37% greater, and, moreover, it is decidedly more accessible for adjustments.

Kirschner† says that Blömeke's conclusions are open to question because the tests of the two machines were not made with the same material. He says that at Pribram, on comparative tests using equal quantities of ore, the Kavan-Rittinger table recovered 39.4% of the silver and 50.7% of the lead, treating 158 kgms. per hour, while the Luhrig table recovered only 22.2% of the silver and 41% of the lead, treating 166 kgms. per hour.

Luther Wagoner‡ gives a short abstract of R. H. Richards' paper, "Close Sizing Before Jigging," in *Trans. Am. Inst. Min. Eng.*, Vol. XXIV., and examines facts about grains under 0.1 mm. diameter.

Mr. Wagoner§ has also deduced new formulæ for the velocity of grains falling freely in water based on the results obtained by Richards,‖ together with those by Pernolet,¶ and by the author himself.

If v = velocity, x = diameter of grain k, a and b = constants for each mineral, the final equation is

$$v = k \frac{x^{\frac{3}{2}}}{\sqrt{a x^2 + v + b}}$$

Making x large, b may be omitted and $v = k\sqrt{x}$ which is the ordinary equation; making x small, ax^2 may be omitted and $v = k\, x^{\frac{3}{2}}$. The author gives a table of the above constants for eight minerals, and from it he finds that the mean value of $a + b = 1.4156$, which is almost the square root of two, viz., 1.4142. He believes it to be an important fact that the value of $(a + b)$ is practically constant, though he has not yet discovered the significance, and he therefore urges further investigation of this question.

*The Cycle of the Plunger-Jig.***—Professor R. H. Richards has designed a jig indicator to give curves showing the behavior of the plunger, the water, the quartz sand layer and the heavy ore layer, during the act of jigging. On the indi-

* C. Blömeke, *Berg- und Hüttenmännische Zeitung*, 1896, p. 13.

† *Oesterreichische Zeitschrift*, July 18, 1896, p. 369.

‡ "The Conditions Necessary for Equality of Velocity in Particles Settling Through Liquids," *Journal Associated English Societies*, Vol. XVII., August, 1896, p. 73.

§ *Ibid.*

‖ *Trans. Am. Inst. Min. Eng.*, Vol. XXIV., p. 414.

¶ *Annales des Mines*, 1853, p. 144.

** *Trans. Am. Inst. Min. Eng.*, February, 1896.

cator card the vertical movement of the pencil point represents rise and fall, while the horizontal movement represents time. As a result of tests upon 40 jigs in different mining districts he concludes that the act of jigging is divisible into three operations:

1. Pulsion or upward movement. The water and quartz curves diverge, because the water is moving up faster than the quartz. Here the law of hindered settling is acting to bring the coarse galena below the quartz, and the fine galena adjacent to, and in equilibrium with, the proper size of quartz.

2. Return downward, which is divided into three periods: (*a*) A moment when the quartz and water-curves are converging, because the water is moving down faster than the quartz. Here the fine grains of galena acquire their maximum velocity downward before the coarse grains of quartz; and hence Rittinger's law of acceleration is probably at work, helping the fine galena to get below the coarse quartz. (*b*) A moment when the two lines of quartz and water are parallel, that is, a moment of relative idleness as to separation. (*c*) A moment when the curves of quartz and water are diverging. The sand is falling faster than the water; hence the law of hindered settling is again at work.

3. Suction. Here the sand reposes upon the sieve. The water curve is converging rapidly toward the horizontal quartz-line, therefore the water is passing down through the sand at a high rate of speed, and suction comes in to draw downward through the interstices the small particles of galena which the law of hindered settling has placed adjacent to, and in equilibrium with, the larger particles of quartz.

*Dressing Quicksilver Ore at Idria, Austria.**—In his article on "The Quicksilver Reduction Works at Idria," Professor R. Helmhacker describes the simple method of dry ore dressing used there. The hand-picked ore is crushed in dry rock breakers, repeatedly hand picked and separated on gratings and sieves of several styles into lumps of 28 to 120 mm., assaying 0.35% quicksilver; coarse ore of 12 to 28 mm. assaying 0.5%, and fine ore up to 12 mm. assaying 0.9%. During the last year recorded the undressed ore was raised from 0.6% quicksilver to an average of 0.85% by the dressing.

Washing Graphite.†—In an article by Ernst Faber on making lead pencils, he says that attempts for chemically purifying the graphite have been given up. The finely ground mineral is mixed with water and thoroughly stirred and disintegrated by a stirring apparatus and by blowing in steam. When the mixing is complete the heavier and coarser particles settle and the rest flows through a series of settling boxes, until they are all filled. The contents of the last box are then pumped through a filter press, the last box yielding the purest product.

Suggestions as to Building a Mill.‡—The location of a mill is generally determined by its water requirements. The advantages of water power are so great that it is often worth while to carry ore to the river banks. The water motor should be low enough to get the full benefit of the fall, but the mill should be well above flood level. There should be ample height for the tailings' dump, and if tailings are worth future treatment proper retaining dams should be had.

* THE MINERAL INDUSTRY, Vol. IV., p. 531.
† Glaser's *Ann. f. Gewerbe u. Bauwesen*, Vol. XXXVII., p. 43.
‡ *Australian Mining Standard*, Nov. 30, 1895, p. 643.

Water for batteries should be carried to reservoirs giving at least 20 to 30 ft. head and holding at least 24 hours' supply for the whole mill. If the water is foul from any cause settling tanks should be provided. In cold climates there should be special tanks for heating the mill water by live steam, but not by waste steam from the engine. If the mill water must be used over, there should be tailings reservoirs where the slimes would settle. Mine water should be purified before using in the mill. A side-hill location is always desirable. The average height of a mill, with rock-breaker floor above ore bins, is about 50 to 55 ft. from the level of the car track to that of the concentrating floor, the length over all the profile of the mill being rather over 100 ft.; hence a slope of one in two is the most suitable for a millsite.

*The Gates Canvas Plant.**—W. S. Hutchinson describes the Gates canvas plant of the Kennedy mill, near Jackson, Cal., for the treatment of the tailings from stamping, amalgamating, and vanners, which contain about $1 per ton in gold, mostly in the form of finely divided rich sulphides, which are too fine to settle on the belts of the vanners. This plant has been entirely rebuilt, with many improvements, since its description by W. H. Storms in *Bulletin 6* of the California State Mining Bureau. The Kennedy mill has 8 batteries of 5 stamps each; and 24 4-ft. vanners. The coarse part of the pulp from the mill, which is not worth further treatment, is separated from the fine in two double hydraulic separators. The overflow from these goes by troughs to 24 canvas tables in two rows of 12 each, facing one another, and sloping to the same tails launder. Each canvas table is 12 by 10 ft. There are two extra tables to provide for the pulp diverted from any two tables while their concentrates are being hosed off. The concentrates go to a settling tank and thence to a small vanner, while the overflow of the settling tank and the tails of the vanner are treated on a canvas table and thence to two other tables in succession. The Kennedy mill crushes about 100 tons of quartz per day, and the canvas plant saves 12 to 16 tons of concentrates per month, worth about $100 per ton, which are shipped to the smelter. The plant, including everything, cost $2000.

Slime Washing.†—C. Blömeke continues his discussion on slime washing (*Berg- und Hüttenmännische Zeitung*, 26 and 28, 1895, Vol. LIV., referred to in THE MINERAL INDUSTRY, Vol. IV., p. 728), with special reference to the Bartsch and Stein slime tables.

The Dry Concentration System of Pape Henneberg.‡—C. Blömeke describes this process and gives the progress it has made up to the present time. A plant for treating 50 tons of hard quartz rock in 20 hours consists of 2 stone breakers, 6 ball mills, 4 centrifugal separators, 4 exhaust fans, 50 sq. meters of sieves, 12 Stein vanning tables, requiring 80 to 90 horse power, two-thirds of which is for the crushing. The cost is 3 marks ($0.75) per ton treated. Plants have been erected in Sweden, the Oural Mountains, and in China.

Briefly described, the centrifugal concentrator is a rapidly whirling horizontal disc 450 mm. in diameter, making 3000 revolutions per minute. The mixed sands fed upon the center of this disc are thrown with great velocity horizontally

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* *Mining and Scientific Press*, Oct. 3, 1896, p. 277.
† *Berg- und Hüttenmännische Zeitung*, June 26, 1896, pp. 209–212.
‡ *Ibid.*, July and August, 1896, pp. 227–261.

in all directions. The fine dust is taken out by suction applied at the circumference of the disc and in a direction toward the center of the same, the air and dust passing off underneath the disc to a settling chamber. The coarser particles, which overcome the force of the air, are thrown out into a series of concentric launders, according to their momentum, the heaviest going the furthest. A radial revolving sweep with a separate scraper for each launder collects and discharges the several products through the bottom into their proper spouts. These several products are graded according to settling power, and are analogous to a series of water-sorted products. In the process the sorted products are sent to sieves and yield heads, middlings, and tailings. The middlings are recrushed and join the dust. The dust is treated with water on Stein tables. A pyritous hornblende yielded 90% of the total gold, while in the usual way only 61% was obtained. A hornblende and quartz with pyrites and gold yielded 96.5% of the gold against 50% by the ordinary method, no mention being made, however, as to what is meant by the ordinary process.

*The method of dressing zinc ore** at the works at Friedensville, Pa., together with a short description of the process and of some of the machines, has been described by H. K. Landis. The screening is done by bar screens, the hand picking upon perforated iron plates. The jigging is by moving sieve jigs operated by power, and the fines are washed on a hand buddle.

Improvements in the Dressing of Gold Ores.†—F. Hille discusses the necessity of better work in preventing loss in tailings in gold milling. He describes the system of gold milling which the Australian commissioners recommended to their government and to the mining world; it is known as the "Luhrig," the "Bilharz," and the "Krupp" systems. The author gives the plan and arrangement of a mill of 40 tons per day capacity built on this system, for treating a gold ore containing copper and iron pyrites, blende, and galena. From the receiving bin the ore is automatically fed to the crusher, which discharges into bins and is then automatically fed to the three 5-stamp batteries. The stamps weigh 900 lbs. each. The pulp from the stamps passes over three Krupp amalgamating tables, and then to four spitzlutten. The spigots go to four Bilharz tables, the overflow to a spitzkasten. The spigots of the spitzkastens go to two circular slime tables, and the overflow to the tailings launder. Mr. Hille says that the amalgamating tables and Frue vanners now used in their mills are not the right machines for their work.

Mr. Hewitt‡ has devised a new method of dry amalgamation of gold ores. The pulverized ore is fed by a hopper and forced by a strong air-blast through a spiral or worm of pipe 3 in. in diameter, possessing a number of coils, and the axis of which is horizontal. Mercury is placed in each coil in sufficient quantity to catch the gold without seriously diminishing the diameter of the pipe. The gold particles impinge violently upon the mercury, which is agitated by the blast. A large saving is claimed with a small loss of mercury.

H. M. Chance§ describes recent improvements in gold milling which have contributed to the present conditions by lowering the cost and increasing the effi-

* *Colliery Engineer*, September, 1896, p. 62.
† *Journal of the Canadian Mining Institute*, Vol. I., Part I., p. 21.
‡ *Engineering*, London, May 15, 1896, p. 646.
§ "Recent Improvements in Gold Milling," *Engineering Magazine*, June, 1896, pp. 461-468.

ciency of gold-milling processes. He gives a table of assumed data showing that getting the greatest yield from the ore does not necessarily bring the greatest profit. Improvements in mill design are noticed, namely, shifting the main line of shafting back from the stamps, and in other cases raising it up so that the driving belts run nearly horizontally, thus prolonging the life of belts and making parts more accessible. There is a general substitution of gyrating cone crushers for those of the Blake type, on account of reduced cost and increased capacity. The large capacity in present free-milling practice is due partly to narrow mortars, rapid short drop, and partly to the relatively larger portion of the work now done by rock crushers. Improvements in material for wearing parts of stamps, crushers, and rolls have helped to reduce cost. He gives also a general discussion on preparation for chlorination, cyanide, and other processes.

*Milling Machinery Run by Electricity.**—John McGhie, in an article on "Long Distance Transmission of Power by Electricity in the United States," writes of the use of electricity (transmitted long distance) at Gold King Mill, Telluride, Colo., for running crushers and stamps. The electric generator (alternating) is run by Pelton water wheels, the current being carried on bare wire 3 miles. At Silver Lake Mines, Silverton, Colo., the current is carried over 3 miles and drives stamps, crushers, and washers in the mills. The Ontario Mills, Park City, Utah, are driven by electricity transmitted 5½ miles.

Electric Mining in the Rocky Mountain Region.†—Irving Hale recites the advantages of the use of electricity for transmitting power derived from water to the mines and mills. He mentions the use of electricity for running mills at Aspen, Colo.; for running the Virginius Plant, Ouray, Colo.; for supplying power to several mines and mills at the Telluride Plant, Telluride, Colo.; also (3-phase system) for mills, mines, and lights at the Silver Lake Mines Plant, Silverton, Colo.

Corrugated Coal Screen.‡—At Pikeville, Tenn., flat shaking-screens of corrugated metal are used. The corrugations running across the screen are 1 in. high and 6 in. apart, from crown to crown, their purpose being to check the coal and give every chance for complete separation.

Adjustable Coal Screen.§—M. L. Mulholland has devised a shaking coal screen the mesh of which may be quickly varied by the use of two plates having square perforated holes of the same size in both plates, one resting on the other. The upper plate may be set so that its holes coincide with those of the lower plate or so that the opening is made smaller. For example, a screen whose normal opening is 2¾ in. square, with bars 1 in. wide between, may be adjusted so that there shall be four times as many ⅛-in. holes as of 2¾-in. holes.

A new coal screen‖ in use in Iowa is made up of transverse shafts provided with cast-iron wheels with corrugated surfaces, all of which revolve in the same direction, that is, down the slope. The wheels are held apart on the shafts by washers of suitable size for the apertures that are desired. The screen is 9 ft. square and does not require so much slope as the ordinary screens.

* *Cassier's Magazine*, February, 1896, p. 370.
† *Trans. Am. Inst. Min. Eng.*, September, 1896.
‡ *Colliery Engineer*, July, 1896, p. 280.
§ *Colliery Guardian*, Jan. 10, 1896, p. 78.
‖ *Engineering and Mining Journal*, Dec. 12, 1896, p. 561.

The Borgman and Emdesher coal screen* consists of flat bars on edge; above and at right angles to them are rollers driven by sprocket and chain; as the rollers push the lump coal forward the fine particles drop through the interstices.

At the Virden shaft, Illinois,† a shaking screen supported upon wheels is used for screening coal.

The Scaife improved trough washer‡ has, for removing the waste, a swing drop bottom, which saves the time and water for sluicing out the whole length of the trough. It is a cheap and efficient device.

Baum Coal Jig.§—Baum obtains the plunger action by compressed air acting at 40 to 50 lbs. pressure per sq. in. directly upon the surface of the water. The admission and exhaust of air are regulated by suitable valves driven by eccentric, pulley, and belt. A slight pause between the closing of the admission and the opening of the exhaust valves permits the pressure to drop to that of the atmosphere and diminishes clearance losses. The inventor claims a better separation of the coal by this device than by the usual piston mechanism, because the movement of the water is accelerated from the beginning to the end of the pulsion stroke of the former, while in the latter the most rapid movement is at the middle of the stroke. When the air-pressure is exhausted the coal settles quietly upon the jig, avoiding the suction and compacting of the piston jig. A feldspar bed is not used even for the finer sizes.

In the Braceville coal-washing plant‖ of Illinois a somewhat modified Robinson washer with a capacity of 600 tons in 10 hours has been adopted. The cost of washing is 5c. per ton, including interest, but would be reduced one-half if the shoveling of the unwashed coal from the cars could be obviated. The yield of merchantable coal is 66% of the unwashed coal.

The Robinson washer¶ has been adopted at the coal-washing plant of the Tennessee Coal, Iron and Railway Company, and is found to be simple, compact, and economical as regards installation, operation, and use of water. It also does not need close sizing.

*Coal Washing.***—After unsatisfactory attempts to wash fine coal on jigs, Wiesner has reduced the ash of such coal 40 to 50% by passing it through spitzlutten.

Elliott Coal Washer.††—The Elliott coal washer is a trough 60 ft. long, 1½ ft. wide at the bottom, 2½ ft. wide at the top, and 8 in. deep, with a fall of 1 in. in 12. In it are scrapers 4 in. high, drawn up-hill by endless chains; these scrapers are 6 ft. apart and carry the slate and pyrites up to the head, allowing the good coal to be discharged at the foot. The slack coal treated is 1¼ in. in diameter and less. The machine consumes 15 gals. of water per minute and uses the same over again. The washer treats 10 tons of coal an hour, and reduces the ash from 24% in the crude to 4% in the concentrated coal.

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* *Engineering and Mining Journal*, Dec. 26, 1896, p. 608.
† *Ibid.*, Dec. 19, 1896, p. 585.
‡ *Colliery Engineer*, December, 1896, p. 190.
§ *Ibid.*, November, 1896, p. 163.
‖ *Engineering and Mining Journal*, Nov. 28, 1896, p. 511.
¶ *Engineering Magazine*, September, 1896, p. 33.
** *Oesterreichische Zeitschrift*, May 2, 1896, p. 43.
†† *Engineering and Mining Journal*, March 28, 1896, p, 303; THE MINERAL INDUSTRY, Vol. IV., p. 729.

*Coal-washing Plant at Brookwood, Ala.**—F. M. Jackson tells briefly the history of the Brookwood coal-washing plant of the Standard Coal Company. The plan adopted by Mr. W. M. Stein, the designer, after a careful study of plants abroad, are the same as used by Schuchtermann and Cremer of Germany. Though much like the Luhrig system, it differs in this main point, that the Luhrig only crushes to 1½ in. before washing, depending on recrushing the larger sizes and rewashing in many cases, while this system requires the coal to be crushed to ⅝ in. in size in all cases before washing and jigging. This washing plant has proved practically successful. A full description of it is given by Mr. Rudolph Boericke in *Coke*.† A brief description of it was given in THE MINERAL INDUSTRY, Vol. II., p. 830.

The Wunderlich Coal Washer.‡—Professor Rochelt describes the Wunderlich coal washer, which he saw in partial operation at the Barre coal mines in Kladno in 1894. This system of coal washing, which has since been perfected, depends upon the free settling of particles in a stream of water, and is based on laws established by Rittinger: two forces act upon the particles, the horizontal force of the running stream and the vertical force of gravity. The author works out the formulæ above referred to for the different sizes. The system consists of four operations: The first washing, yielding clean coal; the washing of the middlings and fines from the first washer; the settling of the sand; and the settling of the fine coal. The washers are troughs made wholly of iron, and carry a stream of water 400 mm. wide and 200 mm. deep, at a speed of 0.8 meter per second. The mixed coal and slate, 70 mm. and less in diameter, is fed at the surface of the water. Three adjustable openings for carrying off the heavy portion are placed in the bottom of the trough. The latter portion is carried forward to a traveling screen, which sifts out the fines and elevates the coarse, good coal to a delivery chute; the fines and the heavy portion are united and rewashed on a second washer. The plant consists of four first washers, two second washers, and one fine coal settler, and yields in 10 hours 240 tons of clean, washed coal.

The Sopris Coal-washing Plant in Colorado.§—The Colorado Fuel and Iron Company has recently erected a coal-washing plant at its mines near Sopris, Colo. The coal that passes bars 1½ in. apart goes to a Jeffrey coal crusher, is screened, and conveyed to bins. The finest stuff is separated from the buckwheat by an air blast on the way to the bins. The various sizes from the bins are jigged, conveyed to the disintegrator, and then to bins. The plant handles 500 tons of washed coal per day.

A draining device‖ at the coal-washing plant of the Colorado Fuel and Iron Company is described. The troughs of the Jeffrey conveyors have perforations in their bottom for a distance of 9 ft., through which the water drains while the clean coal is being conveyed forward.

Coal Washing.—In Chapter III. of Mr. John Fulton's book on *Coke* is given a description of the various machines for crushing and washing coals, together with the method of grouping them in coal-washing plants. The book contains the

* *Proceedings of Alabama Ind. and Sci. Society*, Vol. VI., Part I., p. 50.
† *Coke*, by John Fulton, p. 65.
‡ *Oesterreichische Zeitschrift*, Feb. 15 and 22, 1896.
§ *Engineering and Mining Journal*, Oct. 24, 1896, p. 391.
‖ *American Manufacturer and Iron World*, Dec. 18, 1896, p. 876.

experience of the author, together with compilations from the literature, and some of the plants are written up by other engineers.

*Arrangement of Coal-washing Plant for Treating Bituminous Coal.**—E. G. Tuttle gives the plans and specifications for a 300-ton coal-washing plant, with estimates of cost of construction and running, and of capacity. The design embodies almost all of the requirements likely to be met with in coal washing. The screening is to be done wet. It is assumed that the coal sent to the washer will not be much larger than 3 in. at its greatest dimension, for larger sizes are hand-picked at mine tipple. The coal passing through the screens at the mine tipple is dumped into the receiving pit at the washer, and is then elevated to a double shaking-screen, giving three sizes, which are disposed of as follows: Coal larger than 1½ in. is delivered to a picking belt and thoroughly hand-picked on the way to storage bins. Coal on the ¾-in. screen is crushed to pass this screen and then joins that which has already passed through. All coal passing the ¾-in. screen goes to a revolving screen at the head of a line of three screens, similar in construction but of various-sized mesh. All the sizes of coal above $\frac{1}{16}$ in. are jigged separately; the coal passing $\frac{1}{16}$-in. perforations is separated by hydraulic classifiers. All the coal on its way from the jigs to the storage bins passes over drainage screens with $\frac{1}{25}$-in. perforations; the water and fine dirt passing through goes to settling tanks, and the water is used over again. The dirt goes to a pit and is removed by scrapers. Details of the various machines, with speed, power, and capacity are given. An estimate of the cost of plant and cost of washing is made.

F. W. Hardwick,† in a lecture at Mansfield, England, describes the different systems of washing and screening coal. He brings the machines for washing under three heads: 1. Those with water flowing down an inclined plane; the trough washers. 2. Those with continuous ascending current; Robinson's and Marsant's washers. 3. Those with an intermittent ascending current; the jigs. The Humboldt, Luhrig, Coppee, and Baum systems are also described.

The Baum system of coal washing,‡ at Herne, Westphalia, consists of screening, classification, washing, and storing the coal. The screen is of the Briart type with moving bars; from the oversize, above 80 mm., the slate is picked on a conveyor by hand while the coal is delivered to the cars. The undersize in a conical revolving screen yields four sizes and fines, all of which are washed on Baum compressed-air piston jigs. The coal pockets to avoid breaking the coal are filled with water, which is drawn off after the bin is full and before the coal is run into the cars. The plant described has a capacity of 150 tons per hour. A similar plant§ at the Friedrich Hohenegge Mine in Karwin has a capacity of 75 tons per hour.

An air blast is used‖ at the Colorado Fuel and Iron Company's washer for blowing the dust away from the unwashed coal, which passes through square holes ½ in. wide. The cleaned buckwheat drops into a bin below.

Ore Sampling.¶—The object of D. W. Brunton's investigations and experi-

* *School of Mines Quarterly*, Vol. XVII., p. 378.
† *Colliery Engineer*, November, 1896, p. 161.
‡ *Colliery Guardian*, Nov. 6, 1896, p. 883.
§ *Oesterreichische Zeitschrift*, Oct. 10, 1896, p. 537.
‖ *American Manufacturer and Iron World*, Dec. 18, 1896, p. 876.
¶ *Trans. Am. Inst. Min. Eng.*, Vol. XXV., p. 826.

ments was to get data for determining the fineness to which crushing must be carried, in sampling gold and silver ores, in order to obtain results within an allowable limit of error. The results of careful experiments are tabulated so as to show the effect of one particle of ore, more or less, in 0.1 of an assay ton of various minerals. From experiments and calculations he concludes that the degree of crushing the sample depends upon: 1. The weight or bulk which the sample is to have; 2. The relative proportion between the value of the richest mineral and the average value of the ore; 3. The specific gravity of the richest mineral; 4. The number of particles of richest mineral which are likely to be in excess or deficit in the sample. Two sets of curves, plotted from data found, are given for determining the fineness of crushing necessary in ore sampling; one diagram is for diameters ranging from 1 to 6 in., and the other for diameters varying from 0.0025 to 1 in. In using the diagram to find out how fine a 500-lb. sample of an ore carrying 50 ozs. of precious metals and containing 2500 ozs. of pyrites per ton, must be crushed, it is found that if we crush to 0.25-in. diameter the limit of error will be about 0.1%, while if we crush to 0.5 in. it will be slightly over 1%. The principal error in sampling arises from the use of screen cloth irregularly spaced, and partially worn, and from cutting samples down too far without recrushing.

*Assay Office Sampler.**—A sampling machine for miscellaneous samples (5 to 500 lbs.) has been adapted from the regular Bridgman sampler. It can sample sizes varying from dust up to ½ in. or more. It divides the sample into two equal portions, and may easily be run at 100 revolutions per minute, thus insuring an even distribution of the contents of the sample. It will sample the ore about as fast as it can pass through a 1-in. pipe.

Notes on Conveying-belts and their Use.†—Thomas Robins, Jr., explains the requirements of a belt-conveyor and the function of the cotton duck and the rubber, and describes a series of experiments to determine the best combination of duck and rubber, and the most durable rubber compound for conveyors. In discussing the four principal methods of supporting conveyor-belts he gives the defects of methods in general use and describes a form of belt support composed of three pulleys, one carrying the middle or bottom of the belt, and one on each side with its axis at an angle of about 45°. This mounting gives an enormous carrying capacity and greatly prolongs the life of the belt. The details of the driving pulleys, supporting pulleys, idlers, return pulleys, power, etc., are well described. A diagram showing method of calculating the working-load of belts, and a table showing the amount of stuff carried by different widths of belts for various speeds, are given. Mr. Robins explains why this form of conveying, which is so cheap and simple, has not been more generally adopted.

Roller-pan Conveyor.‡—The Jeffrey roller-pan conveyor consists of a steel chain to which interlapping pans are bolted; this forms a tight, endless, open trough, to each section of which axles and rollers are attached, serving to support both loaded and return pans. It is used to convey clay and shale from bank to pan.

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* *Engineering and Mining Journal*, June 6, 1896, p. 543.
† *Trans. Am. Inst. Min. Eng.*, February, 1896.
‡ *Clay Worker*, April, 1896, p. 380.

*The Hydraulic Elevator at the Chestatee Mine, Georgia.**—The great obstable in working the placer deposits of the South is lack of drainage. W. R. Crandall describes a method of working an underdrain placer by means of his hydraulic lift or jet pump, worked in connection with hydraulic giant and tailing-flume. The elevator has a 5-in. feed-pipe with 1½-in. nozzle for hydraulic pressure of 60 lbs. It lifts to a height of 18 ft. all the dirt and water from a giant with a 1¼-in. nozzle and 60 lbs. pressure. With this elevator many placer deposits unfavorably situated as to drainage are made available.

* *Trans. Am. Inst. Min. Eng.*, February, 1896.

ELECTRICITY IN MINING.

By Timothy W. Sprague.

The phenomena of the production and utilization of the Röntgen rays have occupied more time and attention during 1896 than any other development in electrical science. Probably never before has a subject proved more widely attractive and received in a comparatively short time closer attention from the most eminent scientists of the world. One cause for this lies, of course, in the practical value in medicine and surgery which the discovery was at once seen to possess. It is safe to say that the possibilities in applications of so-called "X" ray photography and "X" ray effects will prove attractive for a long time to come, and that experiments in these lines and others suggested by them will show further benefits to the human race.

The generation of electricity direct from coal or carbon is another subject which has been prominent during the year. The Jacques process has been most discussed and has seemed to offer most promise of developing into a commercial possibility. The process which is the invention of Dr. William W. Jacques, chief electrician of the American Bell Telephone Company, consists essentially in converting the potential energy of carbon or carbonaceous material into electrical energy by chemically combining oxygen with the carbon or carbonaceous material through an intervening electrolyte. In practice the generating cell is an iron pot, the iron forming one of the elements of the cell. In the pot is placed the electrolyte, caustic soda. Heat is applied externally in such amount that the soda is fused. An iron tube carries air from a pump down to the bottom of the fused soda, whence it bubbles to the surface. Into this bath is thrust a stick of carbon. A portion of the air reaches the surface of the carbon and effects, it was at first claimed, a perfect oxidation without chemical change in the electrolyte. That this is the case is widely disputed at the present time. In Dr. Jacques' first practical tests the "pots" were each 12 in. deep and $1\frac{1}{2}$ in. in diameter. There were 100 of these set in ten rows of ten each and suspended over a grate, air being conveyed from a pump to the bottom of each pot. The carbon in each pot was connected to the rim of the next pot, and the carbon of the last pot and the rim of the first pot constituted the terminals of the generator, from which wires led to switches leading in turn to two banks of 60 lamps each, and to a small motor for driving the air-pump after the process was started. In his later experiments Dr. Jacques claims an efficiency of 32%. The results were as follows: The rating of the generator was 2 horse-power. The average net electrical horse-power was 2.05; carbon consumed in pots per electrical horse-power hour, 0.223 lbs.; coal consumed on grate per electrical horse-power hour, 0.336 lbs.; electricity obtained from one pound of coal, of which 0.4 was consumed in pots and 0.6 on the grate, 1339 watt-hours or, as above stated, 32% of that theoretically obtainable. The efficiency of the process is, therefore, said to be 12 times greater than that of the

average electric light and power plant, which is taken as less than 3%, and 40 times greater than that of ordinary plants of corresponding size. The figures are instructive as showing where the coal or carbon is most largely used, and also that in his calculations of efficiency the coal burned on the grate was taken into account.

The Jacques process is extremely interesting, and its development will be eagerly watched. It is not too much to say, however, that no one making an investigation of the process would on account of it postpone an installation of electrical apparatus using the present systems of generating current through the mediums of steam-boilers and engines. In this connection it is of interest to note the remarks of Prof. Elihu Thomson, who stated recently that "the central-station manager need not, for the present at least, fear having to throw aside his boilers and engines. There does not at present loom upon the electrical horizon any such formidable competitor of the ordinary modes of generating electricity from fuel as need give him uneasiness about the value of his present plant, or give rise to any roseate visions of increased earnings due to the substitution of some new mode of generation."

In arriving at such a conclusion the crudities of the Jacques and other similar processes, and the numerous obstacles in the way of their becoming, for a long time at least, commercially successful are not alone taken into consideration, but also the relatively small proportional importance of expense of fuel in large electric stations, except those operating under extreme conditions, such as, for instance, some isolated mining plants. In many plants the cost of fuel is not more than 12% or 15% of the total cost of electric distribution, and where fuel is cheap the actual fuel cost may be so low that even 100% efficiency of conversion would make very slight difference in the total cost of operation of a large plant. Statistics recently gathered by Charles H. Davis from 58 electric railways operating under various conditions in different parts of the country show that the cost of fuel averages but 5.75% of their total operating expense. By the greatest refinements of triple-compound condensing engines running at full load, less than 1½ lbs. of coal will produce one horse-power hour. Such results would show a total conversion efficiency of about 11.8% against 32% shown in the direct process. Many other conditions, such as first cost, attendance, depreciation, ease of manipulation, regulation, etc., would come into the question, as well as a comparison of total costs between such a system (which without complications must be a direct-current system) and an alternating-current system, particularly where distance enters into the problem.

A consideration of this subject of coal-conversion efficiency in the generation of electricity naturally brings up another method of fuel conversion, namely, that of gas or oil conversion. It is stated on excellent authority that an efficiency of 20% can be obtained by means of gas or oil engines. Even in engines of 30 horse-power and less, one horse-power has been generated from the gas produced by usual methods from less than ¾ lb. of coal. A large amount of time and money is being expended in improving and developing these engines, and the future is likely to show a horse-power by brake measurement produced under most favorable conditions with the expenditure of less than ¾ lb. of coal or oil. One point which must be considered in connection with this form of fuel conversion is that

when the engine is stopped the expenditure of fuel stops immediately. No banking of fires is required, and similarly in starting no preliminary expenditure of coal is necessary. As applied in mining operations the tremor which seems inseparable from this form of power generator would not be an important objection, and the comparative ease of transportation of oil and its high calorific power would seem to strongly recommend the oil-engine as a prime mover for electric plants in isolated and comparatively inaccessible localities.

The use of storage batteries in connection with lighting and power stations has largely increased during the year. This is probably due to improvements in the manufacture of the batteries themselves, making them somewhat cheaper, more lasting and thus more economical, and also to a wider appreciation of their value as a flexible adjunct to the generating capacity of the station. They are charged while the load is light, and discharged when it becomes heavy, and when conditions do not require the operation of the generators themselves. In this way the load on the steam-engines is kept more nearly constant, insuring more economical operation, and the cost of fuel and attendance is cut down when the demand for current can be satisfied by the batteries alone. In mining operations the conditions are unlike those of a lighting plant, or those of the city or railway power station, in which cases the peak load, so called, occurs regularly at certain times and can be counted on with certainty. Except under unusual conditions the load in a mining plant is maintained with uniformity throughout the run, or else fluctuates irregularly. There are certain conditions, however, occurring in mining plants where storage batteries can be employed to advantage with the result of lessening the total horse-power requirements in engines and generators, or in allowing the steam plant to be shut down during a portion of the time when it would otherwise be required to run on light loads.

In power transmission, the end of the year saw the long-looked-for transmission from Niagara into Buffalo, and the utilization there of the electric current generated by the power of Niagara Falls, put through a series of conversions in voltage, phase, and variety before its final use for operating street-cars in Buffalo. The cost for power is showing somewhat higher than was expected, so that the profits are not so great as were looked for, and competition with steam-power generated on the spot is not out of the question where the distance is considerable. The idea of transmitting Niagara's power to New York and Chicago is not to be thought of as a commercial undertaking, for the present at least.

The increase in the number of power-transmission plants has been very marked. The multiphase alternating current, with high voltages for transmission, is used almost exclusively in this work. The principal disadvantage in the use of this form of current lies in the fact that for work requiring direct current, or where direct current is at least preferable, one extra transformation is required, which introduces a slight loss and means a greater first cost. This disadvantage is outweighed completely, however, where the distance of transmission is great and the amount of power transmitted is large. In haulage applications the multiphase current itself has been used in one or two cases, notably at Lugano, Switzerland, where an electric railway is in operation using a three-phase current. It has also been reported that a three-phase locomotive is employed in the copper mines of the Compagnie du Boleo, in Mexico, but this doubtless arises from the fact of a

three-phase motor being there used to operate a drum-hoist from which a rope-haulage plant is operated. The three-phase mining locomotive may be developed later, but the necessity of two trolleys will be a decided disadvantage.

The present status of the power-transmission work of the world is shown in a table prepared by Dr. Louis Duncan, which, although not complete and intended to include only the principal plants of the world, shows an aggregate of about 35,000 horse-power transmitted electrically over distances varying from a few miles up to 35 miles on a commercial basis, and up to 100 miles (including the Lauffen-Frankfort plant) experimentally. Supplying several important omissions from this list (among them the Niagara plant) brings the total horse-power well over 50,000, not including a great many plants which come properly under the head of distributing plants. A majority of the transmission plants are on the three-phase system for the main transmission, and Dr. Duncan expresses himself as feeling that the three-phase system is the best adapted to this work. The voltages of transmission range from a few hundred up to 10,000 and 15,000 on the commercial transmissions, and 30,000 on the Lauffen lines. It is a significant fact that a large number of these plants are already increased over their capacity when first installed, or are being increased at present.

One of the most interesting transmission plants contracted for during the year is that of the Pioneer Electric Company at Ogden, Utah. The distance of transmission is 36 miles, or one mile longer than the Fresno, Cal., plant of the San Joaquin Electric Company, mentioned in THE MINERAL INDUSTRY, Vol. IV., the largest commercial plant at present in operation. The Pioneer Company will at first transmit current from Ogden to Salt Lake City at a voltage of 16,000, but the intention is to extend the lines 30 miles beyond Salt Lake City to supply a number of mining properties. When this is done the voltage will be raised to 36,000. A large number of transmission plants were installed during the year for supplying power to mines or mining operations, and the prediction that electric transmission and distribution of power was destined to confer particular benefits on the mining industry, and open up operations which would otherwise be entirely impracticable, has been fully verified. In the Rocky Mountains alone 52 companies have installed electrical apparatus for mining and metallurgical purposes, most of the installations having been made in the last four years. These plants show a total in generating capacity of over 10,000 horse-power. The opinion which was very general a few years ago that electrical apparatus was unreliable has been entirely removed by the splendid showing of the plants themselves.

In the coal-mining districts of the Middle, Eastern, and Southern States the dull times have affected the installation of improved apparatus. In some cases, however, this same cause has led to the placing of orders for coal-cutting and haulage machinery, a cheapening of cost of production being absolutely necessary in view of close competition. In Ohio and Indiana there has been much activity, particularly in machine-mining plants, and the same is true in Pennsylvania and West Virginia.

In underground haulage the principal advance of the year was in the capacity of single locomotives constructed. The Crozer locomotive described in the *Engineering and Mining Journal*, Vol. LXII., p. 5, which is capable of developing a draw-bar pull of about 10,000 lbs., is a long step in advance in capacity of mine motors.

It weighs 22 tons, is 17 ft. 8 in. long, 5 ft. wide over all (for 44-in. gauge track), and 6 ft. in height. It is the first mining locomotive built by the Baldwin-Westinghouse combination, and is used in the mines of the Crozer Coal and Coke Company at Elkhorn, West Va., in the Pocahontas field. The Southwest Virginia Company, at Pocahontas, Va., in the same field, which has long been a large user of electric power, has during the year more than doubled its generating-station capacity. The principal use of electric power there is for coal-cutting by Jeffrey chain machines.

Another large underground haulage machine was placed during the summer in the mines of the Hillside Coal and Iron Company, at Forest City, Pa., by the General Electric Company. It consists practically of two locomotives run tandem, with controlling devices, etc., all placed on one machine, and a single trolley furnishing current to both machines. It is capable of exerting a draw-bar pull of over 5000 lbs. at a speed of 10 miles per hour. The idea is a novel one and would at first sight appear to have some disadvantages, but except for some defects in the motor windings, which have nothing whatever to do with the type of machine and are easily remedied, the locomotive is proving very efficient. The Hillside Coal and Iron Company shares with the Lykens Valley Coal Company the credit of being the first in the anthracite region, or in fact anywhere in the United States, to appreciate the advantages of electric underground haulage. From a very modest beginning the Hillside Company now has at its Forest City plant a most modern equipment, consisting of two 200 horse-power directly connected generators with complete station equipment and several locomotives, hoists, pumps, drills, etc.

Some figures on the actual economy of electric haulage in existing plants follow. The first plant consists of a 100-horse-power engine, a 75-horse-power generator, 1000 ft. transmission, $1\frac{1}{4}$ miles of underground-trolley road, and a $6\frac{1}{2}$-ton locomotive. The locomotive gathers cars from three different points in the mine, and delivers them at the foot of the slope. The main gangway is very crooked. During 1896 the number of trips pulled per day averaged 20, and 8 cars was the average trip, although the capacity of the machine is above this. The grades on the main road, 3100 ft. long, are ordinarily about 1% in favor of the loaded cars. The grades in one branch 1000 ft. long are 3% for 500 ft. and 1% for 500 ft., both in favor of the loaded trip. On the other branch, 2100 ft. long, the grades are nearly all against the loads, running as high as 7% for 150 ft., $5\frac{1}{3}$% for 350 ft., and $3\frac{1}{2}$ and 2% for half the remainder. The locomotive hauls four and five cars on all these grades. The roof throughout is low for an anthracite mine, and the height of entries averages but little over 4 ft. The plant cost complete $7625.18. Depreciation at 5% on the entire plant, and charged against 200 working days, amounts to $1.90 per day. Fuel is not charged, as the steam is taken from a large battery of boilers and no increase over former fuel-consumption is noticed. The cost of haulage, then, shows as follows:

Power-house engineer	$1.75	Depreciation as above	$1.90
Motorman	1.75	Oil and waste (actual)	.20
Helper	1.60		
Repairs (actual)	.76	Total	$7.96

Coal hauled per day 288 tons, making, thus, the cost of haulage 2.76c. per ton. To haul the same amount by mules under the same conditions would require, according to careful estimate:

Seventeen mules at 50c. per day	$8.50	Four boys at $1 per day	$4.00
Three drivers at $1.45 per day	4.35		
Three drivers at $1.25 per day	3.75	Total	$20.60

costing 7.15c. per ton for 288 tons. The saving here shown by mechanical haulage is 4.39c. per ton, or on the regular output, 200 days in the year, $2528. If from this is deducted 6% interest on the investment, or $457.51, the net saving per year is $2070.49 on an investment of $7625.18. This plant operated at the capacity given would therefore pay for itself in less than four years. The locomotive averages 30 miles per day, or 12,700 miles since installed. The repairs amount to a little more than 2c. per mile.

Another haulage plant in an anthracite mine, with less than a mile of total haul, cost $6103. It shows a saving on 250 tons per day of 3.21c. per ton. The locomotive makes about 32 miles per day, and the cost of repairs is 2.7c. per mile.

Still another plant of about the size and capacity of the first two shows a saving per year on 600 tons per day basis, 200 days per year, of $1113.66, the first cost of plant having been $7039. In the latter two cases the time of an electrician is charged in the cost of locomotive haulage, one-half to each plant.

One more example shows a larger saving on an investment of $7880, namely, $1375.20 on 550 tons per day basis, 200 days per year. In the latter cases interest at the rate of 6% on the investment has been taken into account in calculating the net saving.

In coal-cutting, as already stated, there has been much activity in the installation of new plants and enlarging existing plants. The chain type of machine, such as is now manufactured by all the leading companies, is used to the entire exclusion of others. The use of three-phase induction-motors for operating coal undercutters has increased largely, and the motor has shown good results in this trying work. A demand exists for a single-blow pick-machine similar to the pneumatic machines now on the market, for use in connection with electric haulage apparatus or electric chain coal-cutters, but no such machine, commercially successful, is on the market.

The subject of coal-cutting brings up the question of voltage and the danger limit, as one or two fatal accidents have occurred during the year in connection with coal-cutting or haulage plants using 500 volts potential, either direct or alternating. The subject is one which demands close attention. It is, perhaps, not generally understood that while the voltage on a line affects the violence of shock, it does so in an indirect way, so to speak, *i.e.*, the voltage determines, under equal conditions, how much current will pass through any object which becomes part of the circuit. If a man's body is so placed that it becomes part of the circuit, and the voltage of the current is varied under these conditions, the amount of current passing and the amount of injury he would sustain would vary proportionally. If, on the other hand, the voltage were maintained at a constant point, and the contact made with the man's body were increased or diminished, the same changes in effect would take place. The changes in contact would increase or diminish the amount of current flowing, just as was done in the previous case by the change in voltage when the contacts remained the same. In other words, by broad or, as the expression is, " good " contacts with a low voltage line, equal injury might or would be done as with a much poorer or smaller contact with a high voltage line. As an instance of this there is on record an authentic

case of death from electric shock on a circuit carrying only 220 volts potential direct current, where one contact was made through an open bleeding cut on the victim's hand. This, of course, gave a very low resistance to the flow of current, which was, consequently, large even at this low voltage. It was also reported in this case that the victim was a sufferer from heart-disease. This statement is usually made whenever a similar accident occurs; and, while it may be true, it is also true that a majority of people seemingly are afflicted in the same way, and no effort should, therefore, be spared to prevent such accidents to weak-hearted people as well as to those in perfect health.

Another source of danger to employees, which is common to any method of mechanical haulage by locomotives, is that of collisions where two or more locomotives are run upon the same stretches of track. At least one fatal accident of this sort occurred during the year. The conditions so nearly approach steam-railroad conditions that the same caution should be exercised as in the operation of any single-track railroad, and the availability of the electric current for signal-lights would seem to simplify the solution of the matter. Not alone the danger of life and limb, but the damage to property and loss of valuable time, will force attention to this subject.

One development in electric haulage during the year has been an increase in the number of mines operating entirely without mules. The employment of small locomotives for gathering coal from the rooms to the main partings and returning the empty cars to the rooms again, generally in low seams, saves enormously in yardage in main headings, and also, to a large extent, in minor headings and entries.

In coal-cutting the development of the year has been principally in the speed of undercutting. The chain cutters now produced cut at a higher rate of speed than was formerly possible, and this without an undue increase of weight and power consumption. This is due partially to the advantage in this type of machine over the older forms, and also to a better mechanical design in both the frame and running gear by which greater strength is obtained and friction reduced. The chain cutter manufactured by the Morgan-Gardner Company of Chicago has met with much success, particularly in Ohio and Indiana. The Jeffrey, Independent, and General Electric chain machines have also been installed in soft-coal fields throughout the country, and in one or two instances exported. In some cases where in wide rooms 40 to 60 ft. face can be obtained, the chain breast machine has shown as high as 250 lineal feet of face undercut to usual depth during a single shift of 9 hours.

In electric percussion drilling there has been no marked development during the year. The Marvin Drill Company, which is practically alone in the manufacture of electric percussion drills, has installed several plants in the West, and added to the equipment of other plants, notably that of the Solvay Process Company, at Syracuse, N. Y.

The rotary drill is being employed more and more in anthracite mines, for both coal-drilling and rockwork. By means of power drills low and poor veins can be worked and hard-drilling coal mined at a profit. The modern drill weighs about 125 lbs. without standard, which varies in weight according to thickness of vein. The time ordinarily taken for putting a 6-ft. hole 2 in. in diameter in coal is 5 minutes, which includes time for changing bits. Making due allowance for time

taken in moving and resetting standard and for all delays, from 12 to 15 minutes should be allowed for each hole. Two men are required for each drill, and the economy of the method over hand-work, including interest and depreciation, is from 20 to 40%, according to conditions. Mr. F. J. Platt in a recent paper gives the actual results from several plants near Scranton. The Sterrick Coal Company, of Peckville, has been running drills since December, 1893, in one of its veins, which was too poor to be economically worked by hand. When the machine plant was first installed one man was given a contract for driving the heading, airway, and eight rooms. He was paid $1.20 per day. During 1895 this contractor produced 5685 cars or about 11,370 tons of coal in 137.4 working days, or an average of 43 cars per day. The cost of repairs on the drills during that year was 0.6c. per ton produced, which cost has since been reduced. The Sterrick Company is reported as considering that working this vein with electric drills is saving 50c. per car, which would mean a saving for the year of $2843.

The same paper gives interesting figures of the operation of an electric drill used by the Dolph Coal Company, at Winton, Pa. The cost of making a rock-cut 12 ft. high by 14 ft. wide and extending 150 lineal feet was $786.90, or $15.74 per lineal yard. This cut was made in 11 days of three shifts each, or 33 shifts. The saving in time was valuable besides the large saving in cost. A contract is now let to a miner to take down 2½ ft. of rock in 16-ft. headings of the same company. The contractor mines the coal by hand and furnishes labor to blow down rock for $3 per yard. The company drills all holes in the rock with an electric drill, at its own expense, and furnishes powder for the rock-work. Two men with an electric drill bore, for the shooting down of 25 lineal feet of rock, 8⅓ yards in one day. The total cost of mining these headings 8⅓ yards per day is $31.13, including the $3 per yard to the contractor, or about $3.74 per yard. This is made up of $25 paid the contractor, $1.98 paid the drill-runner, $1.50 for the helper, 70c. for sharpening tools, and $1.95 for 15 lbs. giant powder. The cost of driving this heading by hand entirely is $6 per yard, showing a saving, leaving depreciation out of consideration, of $2.26 per yard.

The New York & Scranton Coal Company is using an electric boring-drill for driving a heading and an airway through a vein averaging 7 ft. 5 in. in height, but made up of two strips of coal 23 in. in thickness separated by 43 in. of rock. The cost of this driving using a drill is $2.85 per yard; the cost by hand is $5 per yard. During September, 1896, one drill made 109 yards advance, 58 yards in the heading and 51 yards in the airway, again showing a decided advantage in speed of advance over hand-work, as well as economy in cost.

The cost of the rotary drill is from $150 to $200, and when running at its full capacity it requires about 4 horse-power of electrical energy.

In pumping and hoisting there has been an increase in the size of units employed and a marked increase in efficiency. The largest electric hoist in America is at the shaft of the Free Silver mine at Aspen, Colo., and is equipped with one motor of 150 horse-power and an auxiliary motor of 60 horse-power, giving it a lifting capacity overbalanced of 10,000 lbs. at a speed of 1000 ft. per minute. Orders have just been placed in America with Webster, Camp & Lane for two hoists, each to be equipped with a 250 horse-power induction motor for export to South Africa.

A test just completed on a triplex 6½ by 8 in. Knowles pump, operated by a 50

horse-power General Electric induction motor wound for 110 volts three-phase current, and run under the equivalent of 550 ft. head, delivering 259 gals. per minute, showed a total efficiency of 77%. The motor, which ran at a speed of 450 revolutions per minute, showed an efficiency of 89.2%, and the pump 86%. This pump was one of a lot of twelve of the same size which are also being shipped to South Africa. Included in the same South African plant are six horizontal 5 by 6 in. triplex pumps and fourteen vertical 4 by 6 in. triplex pumps, which are designed for sinking pumps, all fitted with three-phase motors.

To illustrate the quality and extent of electrical mining applications in South Africa, a few further details of this plant follow: The initial installation consists of three Allis-Corliss cross-compound condensing engines, 24 by 44 by 48 in., each directly connected to a 500-Kw. 3300-volt 60-cycle three-phase generator made by the Brown-Boveri Company of Baden, Switzerland. The engines run at 90 revolutions per minute, their fly-wheels being 20 ft. in diameter and weighing, with pole-pieces, 90,000 lbs. each. The engines were made in Milwaukee, Wis., the generators in Switzerland, and the fly-wheels in Blackburn, Eng. The power station, which is just outside Johannisburg, contains space for a fourth unit of the same size. With each generator is a 24-Kw. 75-volt Elwell-Parker exciter, directly connected to a Chandler "Silent" double-crank vertical compound condensing engine. These are manufactured at Wolverhampton, England. The switch-board consists of 15 panels of white marble, carrying brass-finished instruments, the whole mounted on an ornamental iron platform. The panels were made in Baden, and the iron platform in England.

The overhead lines are carried on wrought-iron tapered poles set in cast-iron bases, the latter being fixed with sole-plates on rubble foundations. The cross-arms are of wood supported by malleable-iron brackets. The insulators are of heavy porcelain cemented on galvanized-iron pins. All this material goes from England. Bare wire is used.

Among the motors at first installed are three of 100 horse-power each, wound for 3100 volts, belted to water-pumps, three of 30 horse-power wound for 110 volts, belted to slime-pumps. The other uses of power are to operate four or five electric locomotives for hauling on the different levels. These locomotives are run on an 18-inch gauge and are of 500 lbs. drawbar pull, each weighing about 6000 lbs. The locomotives and stationary motors are of General Electric manufacture. The current is carried in four directions from the central station. The lines provide for the transmission of 1600 Kw. 14,900 ft. in one direction, 1400 Kw. 11,450 ft. in an opposite direction, 350 Kw. 2800 ft. and 250 Kw. 6550 ft. in opposite directions 90° from the first two. These four distributing lines provide for sixteen shafts and two surface pumping-stations. The shafts are 2500 ft. deep, and the 6½ by 8 in. pumps above spoken of will be used in series of five, each raising the water 500 ft. For carrying the current down the shafts special cables are provided, made by the Henleys Company in England, each cable being specially insulated and having a high tensile strength. The service in this plant is severe, calling for continuous running under full loads, and none of the apparatus will be stopped oftener than once in two weeks, or thereabouts, and then only for examination or repairs. The General Electric Company has supplied a plant of about

800 horse-power capacity to another South African mine for the operating of similar apparatus on continuous service.

There has been some development during the year in the establishment of central stations in mining districts. The advantage of such concentration of generating capacity is recognized, but the question of business arrangements between the operators to be benefited seems a stumbling-block. There seems to be lacking in such cases the mutual confidence in each other which is requisite for the successful carrying out of such a plan. As a rule, too, there is not sufficient attraction in the scheme for outside capital unless the companies to be benefited show their interest and readiness to co-operate.

Where several operations belonging to the same company can be supplied from a central station, the plan shows its advantages at once. Such a station, established two years ago by the Youghiogheny River Coal Company, at Scott Haven, Pa., has been largely increased during the year by the addition of a fourth generator, another 10-ton locomotive, portable pump, and electric ventilating apparatus. With these additions the Youghiogheny plant is the largest central-station mining plant in this country, with the exception of that of the Roaring Fork Electric Light and Power Company, at Aspen, Colo. The Davis Coal & Coke Company has installed at Thomas, W. Va., a central station which will eventually consist of three units of generation of 135 horse-power each, and will supply power for coal-cutting, haulage, pumping, and ventilating in several operations within a radius of from 3 to 5 miles. Another use is already made of electric power at this plant in running slack elevators and conveyors. Five motors are now used for this purpose, and the saving effected over the old method of steam-engines and rope-drives is very marked, while the convenience of being able to start the machinery at any time without waiting for steam is also an advantage.

One or two patent decisions of the year are of interest. In the suit of the Independent Electric Company, now controlled by the Link Belt Machinery Company of Chicago, against the Jeffrey Manufacturing Company of Columbus, Ohio, for infringement of patent covering the small auxiliary cutter which is adapted to cut a channel immediately above the kerf, and thus furnish a holding projection engaging in this channel and preventing a side or lateral movement of the traveling frame, Judge Sage, of the United States Circuit Court in the Southern District of Ohio, rendered a decision on November 7, 1896, favorable to the Link Belt Company and restraining the Jeffrey Company from using the device. Other companies are at the present time using another device which accomplishes practically the same purpose, and probably the courts only can decide whether this also is not an infringement on the same patent.

Another decision more far-reaching is that sustaining the Van Depoele patents on underrunning trolleys and overhead construction material, now held by the General Electric Company, which claims for itself, and, under the recent patent agreement, for the Westinghouse Company, the exclusive control of the usual form of underrunning trolley, and of the frogs, switches, etc., in most common use with it. There is no record of any decisions bearing on this point rendered in connection with any mining plant.

THE MINERAL STATISTICS BY COUNTRIES.

In the following tables we group in a condensed form the mineral production, imports, and exports of the different countries. Many of these production statistics are given in the preceding pages under the headings of the different substances, but we give the figures here also for convenience of reference, and because it is well to be able to see what the production of any given country is as a whole and to compare it with others. The following tables, therefore, present a summary of the mineral industry of each country in a condensed form, and also show how large a share that industry contributes to its foreign trade. For the details of the industry, the articles on the different metals and minerals should be consulted.

The statistics in the following pages are nearly all from official reports, and are made up to the latest date to which those reports have been compiled. We give the figures usually for a period of five years, and the statistics for the preceding years, with much detailed information concerning the mineral resources of the several countries, can be found in the earlier volumes of The Mineral Industry. The object is to bring the history of mineral production up to the latest possible date, and at the same time to avoid unnecessary repetition. In the first and second volumes of this series the statistics were given from the earliest period from which they could be compiled, but in the later volumes it has been deemed unnecessary to repeat the earlier figures, since they were already upon record in this accessible form. A period of five years is sufficient for any ordinary comparison; and the searcher after the statistical history of the industry can refer to one of the volumes named. In all cases where new figures have been obtained which were not given in previous editions they are presented in full.

The different countries are arranged in alphabetical and not in geographical order, as that is considered the more convenient for reference.

We must again urge the importance of adopting some uniform system for the collection and arrangement of mineral statistics and for the valuation of mineral products. Something has been done in this direction, but very much remains to be accomplished. In calling attention to this point and in securing the coöperation of the authorities in different countries, The Mineral Industry has already, we believe, been of essential service; and its editor will continue to labor to that end without ceasing.

ARGENTINA.

THE mineral resources of Argentina according to all accounts are important, but up to the present time they have not received the development of which they are capable. The official publication made in Buenos Ayres in 1889, entitled *Memoria General y Especial, Sobre las Minas Metalurgia, etc., de la Republica Argentina,* by H. D. Hoskold, inspector-general of mines of the Republic, contains full details of the mining industry up to that year. Since that time little or nothing on the subject has been published.

The principal mines of the Republic are situated in the Provinces of San Luis, Mendoza, La Rioja, San Juan, and Jujuy. In San Luis and La Rioja gold, silver, copper, and lead are found. In San Juan, which is perhaps the most important mining province, gold, silver, copper, iron, and coal are mined, while in Jujuy there are gold, silver, copper, quicksilver, and coal mines. Certain of the gold mines in the last province are said to be very rich, and with the expenditure of money for their development ought to become large producers. The veins near the towns of La Rinconada, Santa Catalina, and Timon Cruz are said to be especially promising.

Various companies are working gold, silver, lead, and copper ores in the provinces of La Rioja, San Juan, and Mendoza, attention having been devoted especially of late to the production of copper. Certain copper mines in La Rioja are making good returns for the money invested in them, notwithstanding their remoteness from railway lines. The lack of railway communication indeed is the great drawback at present to mining in this part of the Republic.

The gold resources of Argentina have lately been attracting attention, and several new companies have been organized to exploit mines in San Juan and Mendoza; also in the territories of Neuquen and Chubut. The entire Cordillera bounding the Republic on the west is probably more or less rich in mineral, but there has not yet been any scientific exploration of its slopes, and in the absence of this and the lack of railway communication it is improbable that the mining industry will assume any great development in the near future.

There are no regular official statistics of the mineral production of the Republic. Under the date of Buenos Ayres, October 9, 1896, Minister Buchanan reports to the State Department, as per information received from the Director of the Department of Mines and Geology that, according to the most reliable data his department had been able to secure, there were taken from mines in Argentina in 1895 47 kgms. of gold, 10,210 kgms. silver, and 389,900 kgms. of copper.

AUSTRALASIA.

THE latest official statistics of the mineral production of the various colonies of Australia, Tasmania, and New Zealand are for the year 1895. These are embodied in the subjoined tables. Later official statistics of the production of the precious metals and the more important base metals will be found under their respective captions in the preceding part of this book.

MINERAL PRODUCTION OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Alunite.		Antimony Ore.		Bismuth		Coal.		Coke.		Cobalt Ore.		Copper Ingots.		Copper Ore and Regulus	
1891....	715	$9,440	930	$110,285	40	$2,500	4,102,940	$8,713,978	30,798	$172,365	1	$2,350	3,922	$959,390	676	$66,075
1892....	834	16,420	740	73,400	14	5,400	3,841,842	7,311,942	8,026	44,260	77	5,550	3,592	802,365	1,320	136,165
1893....	834	16,420	1,802	125,460	..	.....	3,331,764	5,858,610	18,146	101,165	26	1,525	1,068	221,175	1,032	70,955
1894....	876	17,240	1,270	93,720	..	.....	3,730,829	5,777,865	35,009	166,045	3	50	1,582	305,170	590	62,235
1895....	845	16,640	489	36,255	..	.....	3,798,406	5,476,635	28,072	123,415	6	130	2,838	596,500	1,075	107,925

Year.	Fireclay		Gold—Kgms.		Iron.		Iron Oxide.		Lead, Argentiferous.				Limestone Flux.		Manganese.	
									Ore.	Metal (*b*)	Value.					
1891....	17	$275	4,769.4	$2,791,528	4,192	$180,505	233	$2,170	93,862	56,476	$17,433,820		75,249	$326,785	140	$1,700
1892....	36	400	4,879.1	2,845,889	2,827	113,025	460	4,345	88,905	46,656	12,108,390		105,032	465,155	16	235
1893....	21	230	5,576.5	3,256,430	2,226	73,930	1,280	7,630	158,343	59,775	14,788,970		132,764	555,205	...	.....
1894....	24	300	10,102.2	5,783,585	2,406	85,850	440	3,350	140,018	43,225	10,977,995		91,430	346,445	14	220
1895....	20	275	11,202.6	6,579,646	2,442	78,100	155	1,740	193,236	30,182	7,805,050		105,861	340,800	3	50

Year.	Opal—Kgms.		Shale Oil.		Silver—Kgms.		Tin Ingots.		Tin Ore.		Zinc.		Sundry Minerals.
1891....	.....	.....	40,995	$390,800	22,693	$674,250	2,988	$1,308,845	207	$48,215	222	$13,125	$16,085
1892....	19.5	$10,000	75,384	680,397	10,907	284,420	3,305	1,507,705	243	62,865	452	25,275	5,790
1893....	206.8	61,575	56,551	506,103	16,546	390,655	2,679	1,115,695	150	33,020	....	.....	2,785
1894....	91.2	28,420	21,510	158,906	26,339	470,750	2,653	897,225	193	38,760	....	.....	4,460
1895....	153.3	30,000	60,377	376,095	17,112	409,290	2,235	680,400	78	12,715	....	.....	23,185

(*a*) From the *Annual Report of the Department of Mines and Agriculture*, New South Wales, 1895.
(*b*) Includes a very small quantity of soft lead, the remainder of the metal and the ore representing export figures.

MINERAL IMPORTS OF NEW SOUTH WALES. (*a*) (*b*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal.		Coke.		Copper.		Copper Ore.		Hardware.	Iron and Steel.	Machinery.	Tin Ingots and Ore.
1891....	30,885	$348,885	77,767	$1,865,835	3	$960	14,836	$669,780	$3,326,465	$7,207,940	$2,721,640	$1,104,125
1892....	722	2,575	62,842	857,005	10	1,860	10,202	509,855	3,173,445	5,394,110	1,544,555	1,186,225
1893....	475	1,865	62,864	322,975	92	16,070	1,851	90,320	1,748,730	2,986,005	1,114,205	1,054,200
1894....	380	1,225	48,347	183,420	354	71,040	679	32,535	892,035	2,884,765	1,107,310	893,240
1895....	380	1,870	43,206	134,160	247	50,480	380	20,620	1,007,020	2,889,335	1,358,680	543,680

MINERAL EXPORTS OF NEW SOUTH WALES. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal and Coke.		Copper.				Hardware.	Tin Ingots. (*c*)		Tin Ore. (*c*)	
			Ingots. (*c*)		Ore and Regulus. (*c*)						
1891.....	2,560,515	$6,569,305	3,922	$959,360	676	$66,075	$261,825	2,988	$1,308,845	207	$48,215
1892.....	2,223,293	5,151,975	3,592	802,365	1,320	136,165	263,490	3,305	1,507,705	242	62,865
1893.....	1,870,885	4,104,020	1,068	221,175	1,032	70,955	212,455	2,679	1,115,695	150	33,020
1894.....	2,170,985	4,077,175	1,582	305,170	590	62,235	178,490	2,653	897,225	193	38,760
1895.....	2,209,838	3,912,395	2,838	596,500	1,075	107,925	159,925	2,235	680,400	78	12,715

(*a*) From *British Statistical Abstracts* and *Annual Report of the Department of Mines and Agriculture*, New South Wales. (*b*) Inclusive of imports overland. (*c*) Representing the entire production.

MINERAL PRODUCTION OF NEW ZEALAND. (*a*) (*b*) (IN METRIC TONS; £1 = $5.)

Year.	Antimony Ore.		Coal.		Coke.		Copper Ore.		Gold—Kgms.		Kauri Gum.	
1891....	420	$24,750	679,495	$1,898,690	2,585	$18,290	¼	$20	7.838	$5,037,440	8,523	$2,185,280
1892....	370	24,500	684,088	1,887,135	4,375	28,455	....	.....	7,405	4,773,720	8,845	3,588,390
1893....	336	17,335	702,613	3,366,575	52	265	....	.....	7.054	4,565,690	8,450	2,553,875
1894....	45	3,805	731,059	1,878,545	109	800	....	.....	6,893	4,439,195	8,471	2,022,835
1895....	55	7,430	752.731	2,053,810	292	3,575	....	.....	9,129	5,810,820	7.544	2,093,830

MINERAL PRODUCTION—*Continued.*

Year.	Manganese Ore.		Mixed Ores.	Silver—Kgms	
1891....	1,172	$13,170	$30	872	$25,755
1892....	529	6,195	3,155	686	19,980
1893....	324	4,715	3,250	1,961	48,715
1894....	543	5,780	1,765	1,685	33,485
1895....	213	2,625	4,400	2,644	53,395

MINERAL IMPORTS. (*c*) (IN METRIC TONS; £1 = $5.)

Year.	Coal.		Iron.	Machinery.	Railway Materials	Tools and Implements.
1891....	127,325	$602,110	$2,829,635	$1,019,620	$142,905	$245,870
1892....	127,460	582,765	3,605,650	1,025,485	120,050	253,390
1893....	117,444	544,890	2,040,680	1,060,220	272,705	337,900
1894....	113,768	525,955	1,898,210	889,980	213,985	309,685
1895....	109,929	488,105	1,739,640	770,045	234,940	299,325

(*a*) From *New Zealand Mines Statements*, by the Hon. A. J. Cadman, Minister of Mines, Wellington.
(*b*) The exports are stated to be identical with the production, except in the case of coal, of which substance the exports were as follows: in 1891, 93,131 metric tons, value $455,865; in 1892, 80,174 tons, value $401,125; in 1893, 70,242 tons, value $363,495; in 1894, 76,204 tons, value $367,190; in 1895, 86,063 tons, value $416,710.
(*c*) From *British Statistical Abstracts.*

MINERAL PRODUCTION OF QUEENSLAND. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Antimony Ore.		Bismuth Ore.		Building Stone.		Coal.		Copper Ore.		Gems.	Gold—Kgms.		Opal.		Mundic Ore.	
1891.....	220	$18,125	119	$55,350	.....	.....	275,949	$640,990	100	$4,325	.....	17.920	$11,528,780	$50,000	..	.....	
1892.....	26	1,390	132	80,000	.....	.....	269,354	616.540	82	12,305	$22,185	19,146	12,311,160	50,000	..	.....	
1893.....	30	1,440	75	51,960	27,510	$21,240	267,091	626,700	302	19,110	22,500	19,187	10,796.450	.....	7	$44	
1894.....	28	1,400	66	31,350	42,995	53,115	275.136	572,965	415	47.910	12,085	17,437	11,587,541	60,000	7	380	
1895.....	...	.....	60	16,290	52,206	53,305	328,252	662,650	441	65,485	29,575	15,987	10,623,863	163,750	..	.....	

PRODUCTION—*Continued.*

Year.	Silver—Kgms		Tin Ore.	
1891.....	889	$109,395	2,272	$581,935
1892.....	6,992	182,180	2,427	615,490
1893.....	10,541	212,040	2,522	534,665
1894.....	5,697	110,385	2,916	511,385
1895.....	6,999	150,210	2.148	340,665

MINERAL IMPORTS. (*b*)

Year.	Hardware and Ironmongery	Iron and Steel.	Machinery.
1891.....	$524,010	$2,032,240	$659,830
1892.....	413.650	1,094,040	689,675
1893.....	288,980	749,070	526,010
1894.....	384,425	1,020,590	593,655
1895.....	526,770	1,468,785	875,480

MINERAL EXPORTS. (*b*)

Year.	Copper Ore.		Copper, Smelted.		Tin Ore.		Tin, Smelted.	
1891.....	36	$2,550	87	$17,770	1,800	$506,525	193	$97,000
1892.....	54	4,470	254	57,445	1,876	177,145	426	198.875
1893.....	107	6,420	29	6,475	1,745	444,520	246	119,615
1894.....	64	4,900	135	25,400	1,783	365,725	473	173,400
1895.....	102	10,840	315	62,695	1,180	224,175	459	146,760

(*a*) From the *Annual Reports of the Under Secretary for Mines*, Queensland. (*b*) From *British Statistical Abstracts.*

MINERAL EXPORTS OF SOUTH AUSTRALIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Copper.		Copper Ore.		Gold—Kgms.		Lead Ore.		Silver-Lead Ore		Manganese Ore.	
1891	3,609	$910,710	13,245	$265,875	6,904	$136,900	130	$8,935	.....	.....	861	$8,625
1892	2,769	660,200	10,674	217,425	6,569	130,485	75	2,605	.....	.....	715	7,630
1893	4,389	1,044,835	1,276	28,268	3,262	62,805	35	2,100	.....	.....	2,467	30,940
1894	5,023	1,043,195	305	9,815	8,904	167,005	.....	.....	.....	.....	176	2,585
1895	5,251	1,132,470	223	8,035	7,473	130,300	.....	.....	2	$55	49	780

Year.	Regulus.		Salt.		Slate, Roofing. Number.		Spelter.		Tin Ore.		Various.	
1891	41	$2,500	7,628	$49,695	170,000	$7,100	16	$1,135	10	$340	.....	*b*$11,495
1892	....	.....	6,475	34,220	60,000	1,155	26	1,835	.....	.....	15	240
1893	....	.....	7,229	45,463	.....	.....	73	4,507	.....	.....	21	925
1894	....	.....	7,747	55,235	.....	.....	46	2,895	.....	.....	.....	2,155
1895	....	.....	.....	.....	.....	.....	26	1,610	70	9,075	6	360

MINERAL IMPORTS OF SOUTH AUSTRALIA. (*c*) (IN METRIC TONS; £1 = $5.)

Year.	Coal, Coke, and Other Fuel.		Hardware.	Iron—Bar, Sheet, etc.		Iron—Galvanized, Plain, and Corrugated.		Implements Not Agricultural.	Machinery, Agricultural.
1891	402,552	$2,008,190	$238,395	8,874	$343,875	7,431	$641,875	$490,635	$153,390
1892	324,954	1,560,880	184,335	8,472	318,385	9,610	605,315	408,875	132,930
1893	313,300	1,973,520	163,260	5,394	194,475	4,831	347,680	283,235	140,495
1894	377,231	1,520,195	163,880	4,848	175,620	6,236	450,995	207,580	70,070
1895	352,411	1,207,325	157,880	5,158	163,205	4,153	286,970	190,975	55,535

(*a*) From the *Second Annual General Report on the Mineral Industry of the United Kingdom* and *British Statistical Abstracts.* (*b*) Includes mica, value $6045, and precious stones, $1490. (*c*) From *British Statistical Abstracts.*

MINERAL PRODUCTION OF TASMANIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Coal.		Gold—Kgms.		Limestone.		Silver Ore.		Stones, Building. Cubic Feet.		Tin Ore.		Tin.	
1891	46,252	$227,620	1,219	$749,080	5,132	$9,245	4,887	$261,420	93,838	$22,595	4,391	$1,309,575	3,329	$1,465,850
1892	36,240	178,345	1,403	870,350	2,253	5,750	9,476	227,510	89,050	21,120	4,737	1,280,415	3,246	1,453,970
1893	34,587	170,210	1,158	729,375	1,546	3,965	14,530	993,050	37,470	3,965	4,781	1,032,055	3,179	1,301,095
1894	30,987	121,995	1,800	1,085,120	.....	.....	*b*19,980	983,250	.....	.....	*b*4,304	1,376,700	3,195	1,348,000
1895	33,221	130,795	1,709	1,030,575	.....	.....	*b*21,702	1,068,000	.....	.....	*b*4,265	1,301,350	.....	.....

(*a*) Figures for 1891 to 1893, inclusive, from the *Statistics of Tasmania, Part V.*, "Production;" for 1894 and 1895, from the *Report of the Secretary for Mines*, Tasmania, 1896. (*b*) Fiscal years ending June 30.

MINERAL IMPORTS OF TASMANIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Coal.		Ironmongery and Hardware.	Railway Material.	Copper and Lead.	Iron and Tin.	Earthen and Glass Ware.
1891	40,688	$200,710	$809,110	$177,685	$9,730	$185,260	$111,440
1892	43,852	227,940	599,765	31,055	8,875	128,875	93,355
1893	32,252	121,395	320,690	57,000	8,205	121,100	51,175
1894	40,832	59,760	291,840	26,745	.....	.....	.....
1895	35,968	57,755	327,755	58,465	.....	.....	.....

MINERAL EXPORTS OF TASMANIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Gold—Kgms.		Tin and Tin Ore.		Silver and Silver Ore.		Copper and Copper Ore.	
1891	1,064	$665,065	3,344	$1,465,850	2,524	$310,990	.....	.....
1892	1,164	728,685	3,254	1,453,970	4,440	396,765	.....	.....
1893	939	655,520	3,318	1,330,795	13,200	769,260	184	$229,830
1894	1,602	1,064,650	3,102	1,012,270	.....	.....	.....	.....
1895	1,521	1,011,135	2,784	838,770	.....	.....	.....	.....

(*a*) From the *Report of the Secretary for Mines* and Appendix A to *Statistics of Tasmania for 1893*, excepting the years 1894 and 1895, which are from the *British Statistical Abstracts.*

MINERAL PRODUCTION OF VICTORIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Antimony Ore.		Calcite and Silicate of Aluminum.		Coal.		Lignite.		Copper Ore		Gold—Kgms.		Lead Ore.	
1891	67	$5,940	102	$25	23,202	$98,655	6,424	$8,365	61	$1,080	17,928	$11,527,980	9	$45
1892	299	11,390	.....	.....	23,739	100,220	6,706	18,625	.....	.....	20,356	13,089,120	.....	.....
1893	90	2,155	.....	.....	93,194	245,835	4,573	11,035	.....	.....	20,874	13,414,000	.....	.....
1894	35	875	.....	.....	174,415	474,995	3,566	10,040	492	73,810	19,697	13,089,464	.....	.....
1895	.....	.....	.....	.....	197,343	886,715	1,987	5,240	7	1,050	21,638	14,379,726	20	500

MINERAL PRODUCTION—*Continued.*

Year.	Silver—Kgms.		Slate and Flagging.		Tin Ore.	
1891	.....	$1,050	2,311	$2,675	1,807	$25,460
1892	16	400	626	900	414	5,195
1893	1,234	29,915	171	225	54	10,850
1894	.....	.....	315	425	62	11,430
1895	.....	.....	366	450	75	15,660

MINERAL IMPORTS OF VICTORIA. (*b*) (METRIC TONS.)

Year.	Coal.		Iron and Steel.		Iron-mongery.	Rails.	Machinery.
1891	861,826	$4,189,050	88,106	$5,233,515	$785,250	$52,945	$1,283,970
1892	751,538	3,375,235	61,827	3,687,405	581,065	20,960	776,846
1893	611,826	2,092,420	23,141	1,753,310	340,575	.....	625,000
1894	550,709	977,075	22,795	1,073,520	403,935	.....	472,330
1895	553,343	1,005,235	42,788	1,422,400	429,290	.....	606,855

(*a*) From *Annual Reports of the Secretary for Mines of the Colony.* (*b*) From *British Statistical Abstracts.*

MINERAL IMPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Copper	Coal.		Iron, Pig and Sheet.		Iron Manufactures.	Lead.		Tin.	Zinc.	Soda Compounds.
1891	$3,160	11,858	$60,390	.....	.....	.....	31	$2,310	$14,460	$2,395	.....
1892	1,625	25,607	69,200	2,189	$125,190	$19,453	78	5,820	9,160	2,240	$5,170
1893	1,215	19,116	76,010	3,068	145,255	20,854	67	4,875	6,340	3,445	3,890
1894	.....	26,348	108,910	.....	.....	.....	.....	.....	.....	.....	.....
1895	.....	.....	137,395	.....	.....	.....	.....	.....	.....	.....	.....

MINERAL EXPORTS OF WESTERN AUSTRALIA. (*a*) (IN METRIC TONS; £1 = $5.)

Year.	Copper Ore.		Gold—Kgms.		Lead Ore.		Pearls.	Tin Ore.	
1890	8	$680	709.36	$433,320	217	$10,675	$200,000	68	$27,025
1891	266	22,310	942.79	575,910	25	1,250	200,000	207	51,000
1892	576	43,480	1,852.18	1,131,420	29	750	200,000	270	69,215
1893	51	3,030	3,449.00	2,106,925	.....	.....	150,000	232	55,670
1894	.....	.....	6,443.00	3,935,495	.....	.....	125,000	396	76,380
1895	.....	.....	7,201.00	4,398,740	.....	.....	100,000	.....	.....

(*a*) From the Blue Books of Western Australia, British Blue Books, and Western Australian Year-Books. The exports represent the production, as practically all ores are exported.

AUSTRIA-HUNGARY.

THE mineral and metallurgical production of the Austro-Hungarian Empire is represented by separate statistics for each division, and the production of the attached Province of Bosnia is also stated apart. The latest official statistics for these countries are for the year 1895, except for Bosnia for which there are figures for 1896.

In the case of many of the important metals, however, statistics for 1896 have been collected privately. These will be found under the headings of substances in the preceding part of this volume.

MINERAL PRODUCTION OF AUSTRIA. (*a*) (IN METRIC TONS AND FLORINS.)

(1 florin = 48 cents.)

Year.	Alum and Alum Shale.		Antimony Ore.		Arsenic Ore.		Asphalt.		Bismuth Ore.		Coal.		Lignite.	
1891..	34,394	20,750	333	42,574	4	300	180	2,808	1,083	25,476	9,192,885	32,684,692	16,183,076	30,769,056
1892..	20,480	12,863	97	11,765	....	...	78	1,407	856	22,308	9,241,126	31,680,036	16,190,273	30,096,891
1893..	13,870	10,862	441	50,708	...	..	88	1,301	797	21,818	9,732,651	33,549,863	16,815,955	34,048,957
1894..	10,854	13,495	696	68,632	2	100	116	1,907	570	15,194	9,572,952	33,182,691	17,332,538	32,290,005
1895..	5,716	9,955	695	55,600	*nil*	..	404	8,118	185	.....	9,722,679	34,104,407	18,389,147	34,923,528

Year.	Copper Ore		Gold Ore.		Graphite.		Iron Ore.		Iron Pyrites.		Lead Ore.		Manganese Ore.	
1891..	9,316	354,574	440	14,440	21,346	693,326	1,231,244	2,854,888	2,831	48,522	13,361	1,068,512	5,279	70,742
1892..	8,636	329,824	161	14,886	20,978	637,012	993,290	2,325,088	1,210	25,292	13,265	922,260	4,558	54,830
1893..	8,576	316,232	477	43,787	23,807	637,870	1,109,111	2,482,191	1,221	25,199	10,696	849,869	5,411	58,556
1894..	7,235	278 849	86	9,907	24,121	881,980	1,214,763	2,676,114	...	.....	12,060	836,744	5,055	55,417
1895 .	7,435	286,897	104	38.997	28,443	985,771	1,384,911	2,971,384	....	.....	12,919	883,244	4,352	41,600

Year	Nickel and Cobalt Ore.		Petroleum.		Quicksilver Ore.		Salt.		Silver Ore.		Sulphur, Crude Rock.		Tin Ore.	
1891..	.....	.....	122,006	492,150	70,633	1,035,560	301,422	22,639,125	14,538	3,180,882	3,088	37,204	720	3,600
1892..	0 3	.....	120,000	482,000	79,447	1,007,829	288,424	20,796,030	14,171	2,672,606	1,804	20,987	33	2,900
1893..	.....	.....	ö122,000	488,000	76,215	735,493	305,586	20,440,426	18,018	3,415,704	1,950	20,485	26	2,741
1894..	0.55	.....	131,930	527,720	84,127	988,754	311,597	24,281,350	18,338	3,000,171	2,435	34,484	24	2,465
1895..	*nil*	... ..	....	.....	86,683	797,218	278,875	23,901,782	18,113	2,294,043	830	8,293	24	2,556

Year.	Uranium Ore.		Wolfram Ore.		Zinc Ore.		METALLURGICAL PRODUCTION. (*a*)							
							Alum.		Antimony.		Bismuth.	Copper.		Copperas.
1891..	22	19,314	57	21,380	28,878	575,557	1,127	74,340	115	45,118	0.68	5,606	1,033	584,720	1,184	30 158
1892..	18	24,889	72	19,806	33,944	580,365	1,096	69,902	114	44,489	0.55	4,765	837	502,593	1,085	31,698
1893..	21	28,647	43	10,066	30,531	465,230	837	54,344	175	74,090	0.58	5,327	944	568,221	1,220	25,199
1894..	26	59,404	40	10,908	28,491	439,780	1,150	73,656	279	107,341	0.02	2,023	1.341	744,772	1,480	27,800
1895..	31	51,761	35	9,154	25,862	384,330	885	54,935	296	92,031	*nil*	..	865	460,990	160	3,593

(*a*) From *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums ; Bergwerksbetrieb Oesterreichs.*
(*b*) Estimated.

Year.	Copper Sulphate.		Gold—Kilos.		Iron, Pig.		Lead.		Litharge.		Nickel and Cobalt Products.		Mineral Paint.	
1891..	198	42,286	14	19,272	617,145	24,881,094	7,583	1,206,104	2,267	253,058	1.5	450	838	25,350
1892..	133	24,587	13	17,580	630,790	24,417,266	7,252	1,125,493	2,520	393,356	0.15	196	2,022	57,429
1893..	177	35,275	35	52,506	663,345	24,186,083	7,212	1,103,840	2,411	352,648	0.12	163	3,020	75,329
1894..	140	28,832	61	96,779	742,372	26,750,603	7,570	1,113,088	2,057	301,514	0.1	156	3,002	76,379
1895..	246	49,231	75	116,753	660,549	22,858,237	8,085	1,204,980	2,034	297,708	*nil*	.....	3,164	73,632

Year.	Quicksilver.		Silver—Kilos.		Sulphur.		Sulphuric Acid.		Uranium Salts.		Tin.		Zinc.	
1891..	570	1,383,682	36,037	3,219,048	45	3,416	12,268	374,576	4	45,244	56	63,718	5,006	1,375,076
1892..	542	1,148,320	36,658	3,293,746	53	4,338	11,038	315,433	2	28,298	72	85,184	5,237	1,264,587
1893..	512	1,068,515	37,344	3,330,265	44	3,613	10,248	336,509	5	74,267	66	76,717	5,870	1,212,709
1894..	519	1,056,718	38,246	3,031,631	76	536	9,938	264,280	5	69,742	80	80,860	6,810	1,268,940
1895..	535	1,168,512	40,081	2,524,993	*nil*	.....	7,431	204,485	4.5	64,009	60	50,536	6,456	1,096,008

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*) (IN METRIC TONS AND FLORINS.)
(1 florin = 48 cents.)

Year.	Antimony Ore. (*b*)		Asphaltum and Asphalt Oil.		Alum Ore.		Coal.		Lignite.		Briquettes.		Coke.	
1891..	1,535	140,500	7,649	9,518	1,407	2,060	1,019,352	4,990,356	2,427,126	7,713,443	35,446	275,086	17,124	167,298
1892..	853	72,788	.....	200,000	1,069	1,560	1,052,214	5,174,772	2,741,391	8,085,417	34,882	232,663	2,129	18,951
1893..	881	67,000	.....	.....	934	1,400	982,798	5,161,900	2,917,899	9,394,300	34,190	269,800	3,188	29,300
1894..	1,265	68,733	4,432	223,791	634	1,141	1,037,322	5,447,201	3,181,071	10,301,700	30,057	233,247	10,250	88,767
1895..	1,240	66,330	4,370	186,150	634	1,141	1,068,046	5,640,514	3,517,901	11,244,391	29,421	234,784	12,033	11,302

Year.	Galena.		Iron Pyrites		Manganese Ore.		Mineral Wax.		Nickel and Cobalt Ore.		Petroleum.		Realgar		Salt.	
1891..	213	36,200	57,715	250,349	128	2,357	1,656	94,767	..	.....	726	23,800	...	...	162,891	14,169,212
1892..	238	40,500	56,050	236,097	1,304	8,340	.....	.....	340	29,240	20	690	0.2	....	176,368	15,706,027
1893	213	40,500	68,189	277,900	1,238	1,800	.....	.....	202	18,700	14	690	1.0	...	167,209	14,605,00
1894..	224	38,700	74,619	320,970	3,748	6,700	.....	... ..	25	8,106	2,061	66,100	.	....	169,282	14,903,000
1895..	80	13,600	66,683	249,947	3,381	1,500	.....	.....	55	21,182	2.083	70,300	*nil*	...	169,395	13,451,000

Year.	Zinc Blende. (*b*)		Sulphuric Acid.		Antimony.		Copper.		Copper Sulphate.		Gold—Kilos.	
					Crude. (*c*)	Regulus.						
1891..	127	2,650	2,230	60,387	1,370	268,745	247	156,552	... ...	.....	2,183.92	3,046,440
1892..	116	2,825	3,340	54,983	343	138,003	317	165,215	5.0	1,871	2,246.77	3,134,437
1893..	77	1,050	2,337	36,700	361	144,700	343	174,400	2.0	300	2,000.00	2,790,200
1894..	.....	.....	4,018	57,046	385	124,642	271	135,367	1.4	350	2,687.07	4,497,627
1895..	.....	... ..	4,223	91,500	465	141,922	286	144,220	*nil*	.....	3,187.27	4,869,959

Year.	Iron, Pig.		Copperas.		Lead.		Litharge.		Nickel and Cobalt Products.	
1891..	304,701	11,525,357	540	10,212	2,173	350,230	351	60,668	58	29,240
1892..	309,494	11,690,952	595	10,233	2,335	412,558	507	93,884	58	20,272
1893..	319,362	12,103,200	762	12,300	2,514	348,600	227	39,600	202	18,700
1894..	312,148	11,216,865	795	7,950	2,113	295,015	689	117,512	23	9,374
1895..	322,206	11,802,398	521	2,086	2,277	323,174	615	119,592	18	5,429

Year.	Mineral Paints.		Quicksilver—Kilos.		Silver—Kilos.		Sulphur.		Carbon Bisulphide.	
1891..	221	8,820	8,631	19,385	16,736 6	1,506,294	40	3,392	174	31,329
1892..	263	10,508	7,853	15,641	18,423.8	1,658,143	42	3,773	116	20,208
1893..	321	8,600	2,500	4,800	23,631.0	2,120,000	70	5,900	249	44,800
1894 .	608	13,853	1,837	4,699	20.155 0	1,217,186	93	7,809	248	44,568
1895..	371	11,211	1,129	2,400	20,432.3	1,231,584	102	7,826	237	42,714

(*a*) From *Magyar Statisztikai Evkönyv.*
(*b*) This does not include the ore produced in Hungary from which the metallurgical products given in the table were derived. It gives only the amount of ore which was sold, and presumably was not smelted in the country.
(*c*) Pure sulphide of antimony prepared by smelting.

MINERAL IMPORTS AND EXPORTS OF AUSTRIA-HUNGARY. (*a*) (IN METRIC TONS.)

IMPORTS.

Year.	Aluminum, Alloys, and Manufactures	Antimony Regulus.	Asbestos. Crude.	Asbestos. M'f'd.	Asphalt. Raw.	Asphalt. Refined	Barytes.	Borax.	Brass Manufactures.	Coal.	Coke.	Charcoal.	Lignite	Chrome Ore.	Clays. Crude	Clays. M'f'd.
1891	42.8	0.4	101	37	1,082	2,022	3,585	1,535	2,269	3,670,196	248,193	1,150	15,174	10	24,115	97,539
1892	49.3	2.1	15	96	1,822	1,442	4,584	852	1,731	3,352,129	254,000	1,023	18,338	10	27,776	121,632
1893	41.2	. . .	97	99	2.144	1,012	4,413	1,451	1,750	3,840,874	309,561	853	20,111	570	25,149	141,143
1894	28.0	.....	1.89	3.66	3,052	865	4.801	1,320	1,955	4,098,266	437,259	1,317	16,700	1,081	24,961	158,961
1895	42.8	.....	2.02	2.29	1,661	748	5,063	1,908	2,246	4,503,002	533,402	1,700	16,797	1,827	27,493	194,468
1896	47.3	... ..	127	...	4,088	1,621	.....	1,439	2.417	5,174,321	.....	.....	.....	1,891	.....	.....

EXPORTS.

1891	6.3	276.4	15	27	51	35	192	27	1,138	713,069	73,453	20,238	900,255	157	28,447	47,974
1892	22.9	305.9	9	69	287	...	33	30	1,234	653,667	87,745	16,181	6,748,844	293	30,882	39,448
1893	28.1	.....	47	187	223	22	35	68	998	642,424	108,576	14,838	6,763,186	327	31,307	44,881
1894	31.5	.....	21	92	295	80	61	55	1,006	624,670	110,559	12,360	6,902,393	240	33,007	49,300
1895	7.0	... ..	44	78	134	43	264	45	1,115	639,762	119,050	10,673	7,110,365	385	37,666	51,727
1896	.....	.....	17	...	101	. ....	.....	72	1,065	658,637	.....	.....	.....	142	.....	.....

IMPORTS.

Year.	Cobalt and Nickel Ore.	Copper. Mf'es	Copper. Ore	Packfong and Manufactures.	Tomback & M'f'res.	Cryolite Crude.	Cryolite M'f'd.	Fluorspar. Crude.	Fluorspar. M'f'd.	Glass and Glassware.	Gold. Coin.	Gold. Metal	Gold. Ore	Graphite. Crude	Graphite. Refined.	Gypsum, Crude	Hydrochloric Acid.	Iron Manufactures.
1891	249	9,951	909	349	1	79	52	3,900	444	4,147	23.6	20.7	0.5	691	181	1,657	499	47,623
1892	459	8,337	909	321	5	128	73	2,837	275	3,652	29.2	30.8	2.3	638	93	1,441	460	49,237
1893	971	11,822	638	292	41	131	99	2,782	228	3,599	46.9	62.1	3.1	782	89	862	438	58,791
1894	1,393	13,383	25	317	16	119	89	3,294	324	4,359	17.4	61.5	3.8	655	77	1,375	498	67,096
1895	1,020	11,746	31	481	15	113	115	3,187	341	4,269	29.2	44.6	5.2	480	160	850	467	96,195
1896	719	13,763	.7	376	324	134	...	3,433	.....	.....	.. ...	13.0	...	620	.....	.....	.....	... .

EXPORTS.

1891	233	590	10	488	24	10	2.0	35,524	15	42,788	5.3	65.5	0.3	6,490	4,887	855	1,540	36,502
1892	194	619	19	435	26	14	..	35,187	22	44,093	8.5	41.1	66.4	7,074	4,911	970	1,876	31,782
1893	305	683	14	448	4	2	1.0	159	63	45,087	9.4	45.5	73.2	7,253	4.284	838	1,841	19,518
1894	129	710	15	576	7	14	0.9	64	6	47,457	15.1	38.0	..	5,115	6,420	901	1,908	20,872
1895	139	410	17	495	4	8	2.7	33	10	50,726	15.3	117.0	1.1	3,629	8,254	1,496	1,460	18,849
1896	113	.....	383	237	176	.6	....	22	.....	.....	.....	.7	45.	3,991	.....	.....	.....	.....

IMPORTS.

Year.	Iron. Ore.	Iron. Pig.	Iron. Rails.	Iron. Spiegel.	Iron Pyrites.	Iron Sulphate	Kaolin. Crude	Kaolin. M'f'd	Lead. Lead, Alloys, M'f'es.	Lead. Litharge.	Lead. Ore.	Lime. Crude	Lime. Hydraulic.	Chloride of Lime.	Manganese Ore.	Marble.
1891	68,121	40,743	705	104	30,770	745	2,827	1,221	4,561	250	283	7,536	16,744	867	425	1,494
1892	71,679	45,843	333	920	32,261	1,424	2,154	1,622	7,348	316	355	8,558	18,931	1,016	519	1,939
1893	73,248	57,733	927	697	45,530	800	2,273	1,874	6,286	316	299	9,115	17,098	1,559	354	1,816
1894	88,212	105,799	100	511	59,685	866	2,985	2,808	8,842	200	290	9,410	17,155	1,966	2,445	1,888
1895	117,600	133,595	236	323	54,609	621	2.589	3,343	8,974	208	416	11,982	7,743	2,358	2,772	1,886
1896	107,018	.. ...	.....	.....	50,691	.....	. ..	.....	7,436	233	540	.....	.. ...	1,988	7,371	...

(*a*) From *Statistische Uebersichten betreffend den auswärtigen Handel des Oesterreichisch-ungarischen Zollgebiets*, Wien. The figures for 1896 are from *Chemiker Zeitung*.

EXPORTS.

1891	88,059	8,394	253	294	853	747	5,811	34,881	255	1,007	3,658	19,924	58,750	311	8.364	3,041
1892	108,120	7,932	161	1,153	683	417	16,988	30,185	202	1,334	3,784	14,432	59,784	38	3,768	2,389
1893	106,259	9,274	20	371	304	476	23,624	26,192	270	1,311	1,608	28,019	66,543	216	3,697	2,638
1894	144,309	9,017	16	82	597	871	28,618	25,020	242	1,273	1,232	38,318	63,526	314	2,099	2,922
1895	165,402	7,334	52	45	383	1,039	24,037	32,166	208	1,294	3,758	33,479	34,698	267	425	2,737
1896	254,390	.....	.....	.....	341	.....	... ..	.....	272	597	3,076	.. ..	.....	114	701	.. ...

IMPORTS.

Year.	Oil, Mineral.		Paint, Mineral.		Nickel and Manufactures	Peat.	Phosphates, Phos. Acid.	Platinum and Other Prec. Metals	Potassium.	Chloride of Potassium.	Nitrate of Potassium.	Quicksilver.	Salt.	Silver.			Slag.
	Crude	Ref. or Partly Refined	Crude	Ref.										Coin.	Metal.	Ore.	
1891	122,790	16,735	1,718	2,409	52	2,485	217	0.05	767	1,917	27,199	17	26,801	49	79	22.0	426
1892	135,823	14,479	1,295	2,165	46	2,935	200	2.10	164	1,945	28,071	2	27,836	60	88	0.5	3,621
1893	139,168	16,764	1 680	3.147	51	2,228	238	.....	....	1,869	37,626	3	26,609	59	58	0.1	1,659
1894	126,159	17,019	1,567	2.636	139	1.649	346	0.99	5	2,405	34,033	11	32,821	73	56	..	1,065
1895	120,479	16,876	1,637	2,605	167	1,993	206	0.70	20	2,678	29,792	4	40.396	49	4	...	981
1896	85,703	... ...	.....	.. ..	165	.....	224	0.33	.	2,474	.....	1.3	53,652	..	143	...	240

EXPORTS.

1891	443	2,188	1,138	1,605	83	2,712	1.0	0.70	243	315	440	488	4,861	31	120	10.0	84,484
1892	576	1,873	1,169	1,232	237	5,101	1.0	3 30	259	925	479	465	10,088	79	127	23.0	49,647
1893	613	3,767	922	1,409	79	5,441	6.0	1 20	99	1,118	388	414	7,678	87	131	14.0	45,478
1894	975	5,053	860	1,720	378	3,823	2.0	2.05	63	627	401	509	6,809	102	153	0.1	58,219
1895	1,454	11,605	661	1,583	55	3,753	0.3	1.90	183	1,074	418	493	6,554	47	91	.	52,688
1896	24,921	.. ....	.....	... ..	437	.....	0.1	0.27	..	1,026	....	569	.....	..	3.27	16.	43,187

IMPORTS.

Year.	Soda.			Glauber Salt	Rock Quartz, etc.	Stoneware.	Stucco.	Sulphur.	Sulphuric Acid	Tin and Manufactures.	Various Minerals.		Various Non-Prec. Metals and M'f'es.	Various Ores.	Zinc.	
	Calcined	Caustic.	Crude & Crys.								Crude.	Ground or Broken.			Ore.	Mf'es of Zinc.
1891	1,391	690	911	5,484	115,893	42,711	7,544	14,939	2,689	2.378	49,704	38,840	1,484	327	104	11,715
1892	807	1,316	197	5,389	117,773	55,477	7,925	13,900	1,909	2,406	46,506	40.266	1,417	432	3,481	14.724
1893	953	1,446	95	6,688	155,571	70,394	8,074	13.090	2,090	2,921	50,722	45.548	1,422	401	3.739	15.529
1894	1,026	1,656	258	8,503	152,606	76,161	10,619	14.410	1.530	3.133	70,177	43,989	1,609	271	6,760	15.314
1895	550	1,155	40	6,617	114,501	46,590	10,916	14,709	1,565	3.038	74,231	52,439	1,785	195	7.691	17.156
1896	1,332	835	57	4,678	.....	.....	....	15,221	3,522	3,377	.....	... ...	... ...	1,882	.....	18,082

EXPORTS.

1891	1,063	626	806	78	315,857	37,731	1,814	352	3,953	125	25.773	34,085	3.380	1,283	7.766	1,184
1892	1,048	202	271	77	260.946	36,943	1,766	474	2,855	144	21.598	35.405	3,400	905	12.578	1,442
1893	608	194	323	570	272.239	36,743	.....	1,508	3.906	150	11,866	33,642	. ....	619	7.578	1,939
1894	497	378	217	693	376,310	12,607	1,529	1,232	8.210	182	13.242	28.172	3,678	1.040	6.979	1,648
1895	1,501	92	278	661	289,304	11,123	1,439	989	6,465	143	19,721	58,853	3,867	1,008	7,491	1,661
1896	2,482	143	437	2,291	.....	.....	.....	1,231	5,912	208	.. ..	.....	.....	703	. ..	2,335

MINERAL PRODUCTION OF BOSNIA. (*a*) (IN METRIC TONS AND FLORINS.)

Year.	Chrome Ore.		Lead-Silver Ore.		Lignite.		Manganese Ore.		Salt.		Zinc Ore.		Antimony, Crude.		Copper.		Pig Iron.		Quicksilver.	
1891	918	10,030	109	2,280	77,266	66.160	8,847	58,000	5.970	167,660	47	188	27	2,720	160	32,000	987	11,200	5.2	5,560
1892	1.296	14.411	41	1,680	85,449	76.049	7,944	51,203	8,006	224,000	16	64	91	6,240	140	26,602	3,173	41,360	0.3	281
1893	965	19,996	...	.....	118,263	113,434	7,403	44,265	8,517	238,476	.	..	3	198	101	19,796	2.816	36,613	0.2	600
1894	1,808	30,890	...	... .	160,800	414,100	6,588	104,085	10,250	779,000	.	...	..	.....	270	135,197	3.287	107,477	...	...
1895	707	... ..	...	.....	195,442	...	8,145	...	12,758	892,983	.	...	..	.....	105	...	2.569	...	0.6	..
1896	443	13,566	...	....	222,724	469,674	68,211	112,548	137,199	.....	..	...	..	.. ..	206	.. ...	10,120	295,158	...	.....

(*a*) From *Oesterr. Zeits. für Berg-, Hütten-. und Salinenwesen* Besides the substances specified in the table there was also produced: Antimony ore, in 1891, 108 tons; 1892, 276 tons; quicksilver ore, in 1891, 88 tons; 1892, 3 5 tons. In 1893: Quicksilver ore, 0.886 tons; copper ore, 2365 tons; iron ore, 6874 tons; antimony ore, 0.089 tons; fahl ore, 0.318 tons; iron pyrites, 0.700 tons; coal, 122,396 tons; iron (pig), 45.16 tons; and salt, 85.51 tons.

Gold ore is mined in Austria in Bohemia, Salzburg, Silesia, and Carinthia. The most important mines are in Salzburg. Silver ores are mined in Bohemia and Bukowina, the chief mines being those of Pribram. The entire production of quicksilver in Austria comes from Krain, where there were three mines in operation in 1895. By far the more part of the output of quicksilver comes from the fiscal works at Idria.

The most important copper mines of the empire are in Salzburg, and are operated by the Mitterberger Kupfergewerkschaft. This company has mines at Mühlbach and Einöden, and smelting-works at Ausserfelden. The fiscal works in the Tyrol produce about 1000 tons of ore per annum. The largest part of the pig-iron production of the empire comes from Bohemia, Moravia, Bukowina, and Styria, the same provinces being the largest producers of iron ore.

The most important producers of lead ore are the mines at Raibl, in Carinthia, and after them those of the zinc district in Galicia. The chief mines of zinc ore are at Raibl, in Carinthia, but Galicia is a zinc-producing province of almost equal importance, the great deposits of Upper Silesia extending across the Austrian frontier into Galicia. There are in the empire seven works reducing zinc ore, of which three are in Galicia and one each in Bohemia, Styria, the Tyrol, and Krain.

Tin ore is produced only in Bohemia in the vicinity of Graupen, but these mines are no longer of much more than mineralogical interest. They are worked now more for their wolfram ore than for tin. The production of antimony ore and metal comes from Schönberg, Proutkowitz, Mileschau, and Prícow, in Bohemia. Uranium ore is mined at Joachinsthal, in Bohemia and uranium salts are prepared from it at the same place. The chrome-ore mines of Kraubath have not been operated since 1881. Manganese ore is mined in Bukowina and Krain. Graphite is produced in Bohemia, Moravia, and Styria. The Austrian graphite is of very high quality, and is used chiefly for the manufacture of lead pencils.

The most important mines of brown coal are in Bohemia, which in 1895 produced 14,939,681 tons out of a total of 18,389,147. Of the total production of stone coal in 1895 (9,722,678 tons) 3,864,107 tons were from Bohemia and 3,608,751 tons from Silesia. The salt industry in Austria and Hungary is described in the article "Salt in Austria," by Professor Helmhacker, elsewhere in this volume.

In Hungary there are in operation 62 gold mines, 23 silver, and 13 silver-gold mines. Iron mining is making great progress, 107 mines being in operation. There are only a few deposits of coal in the kingdom, the most important being that at Fünfkirchen, which is of Jurassic age. Only the Oravicza seams, in the Banat, are carboniferous. Brown coal and lignite are widely distributed through Hungary.

The ore deposits of Nagybanya were described by G. Szellemy in a paper read at the Budapest Mining Congress in 1896. The veins, which in some cases extend for many miles, are characterized by great width. The vein stuff is for the most part quartzose. Galena and zinc blende predominate in the northern and western part of the district. The occurrence of auriferous iron and copper pyrites and rich silver ore is general, the richest in gold being those of the Rothwasser Valley, near Nagybanya, the veins of Felsobanya, and the 17 veins in the Guttin Mountains, near Kapnik. These mines were worked in Roman times.

BELGIUM.

THE most important mineral industries of Belgium are the coal, iron, and zinc, which are treated under those captions respectively, where the statistics of production for 1896 are given. With respect to the Belgium coal industry special reference should be made to the paper by Messrs. Le Bel and Schneider.

MINERAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND DOLLARS; 5 f. = $1.)

Year.	Blende.		Calamine.		Iron Ore.		Iron Pyrites		Lead Ore.		Manganese Ore		Totals.
1891	10,200	$163,280	4,080	$47,410	202,204	$234,540	1,990	$3,820	70	$1,620	18,498	$50,920	$501,580
1892	8,250	126,880	4,010	49,440	209,943	218,620	2,570	5,480	60	1,640	16,775	41,666	463,720
1893	7,300	83,720	4,010	43,440	284,465	295,580	6,301	9,800	67	1,520	16,820	40,100	474,160
1894	7,570	76,000	4,015	39,700	311,222	316,440	3,050	5,980	160	3,380	22,048	55,540	497,040
1895	8,080	72,510	4,150	40,340	312,637	296,090	3,510	7,230	220	5,100	22,478	57,254	478,524

COAL PRODUCTION OF BELGIUM.

Year.	Metric Tons.	Value.	Per Ton.	Profit per Ton.	Number of Workmen.	Annual Wages.
1891	19,675,664	$49,490,800	$2.52	$0.36	118,983	$217
1892	19,583,173	40,257,600	2.06	.12	118,578	191
1893	19,410,519	36,281,180	1.87	.07	116,861	177
1894	20,534,501	38,258,420	1.86	.08	117,103	188
1895	20,457,604	38,071,540	1.89	.08	118,957	189

METALLURGICAL PRODUCTION OF BELGIUM. (*a*) (IN METRIC TONS AND VALUE PER TON.)

Year.	Forge Pig Iron.		Foundry Pig Iron.		Bessemer Pig Iron.		Thomas Pig Iron.		Total Pig Iron.	
1891	445,436	$10.18	56,241	$11.78	147,913	$14.01	34,536	$11.40	684,126	$11.20
1892	442,009	9.15	67,236	9.62	190,599	13.09	53,424	10.39	753,268	10.28
1893	428,480	9.00	74,630	9.54	165,077	11.59	77,077	9.47	745,264	9.68
1894	378,045	9.32	80,110	9.45	170,420	11.43	190.022	10.18	818,597	9.97
1895	329,750	8.94	85,950	8.75	161,606	11.07	252,428	10.05	829,234	9.65

Year.	Wrought Iron.		Cast Steel.		Rolled Steel.		Lead.		Silver—Kgms.		Zinc.	
1891	497,380	$29.20	243,913	$19.63	206,305	$28.22	12,698	$61.35	33,950	$32.76	85,999	$112.26
1892	479,008	27.09	260,037	17.90	208,281	26.50	10,146	53.03	30,267	28.94	91,546	101.74
1893	485,021	25.51	273,113	16.79	224,922	25.67	12,006	51.23	26,717	25.79	95,665	82.79
1894	453,290	25.41	405,661	16.27	341,318	23.55	14,120	49.90	28,961	20.91	97,041	74.54
1895	445,899	24.99	454,619	15.14	367,947	23.05	15,573	53.97	31,543	21.75	107,664	71.51

QUARRY PRODUCTION IN BELGIUM. (*a*) (VALUES IN DOLLARS.)

Year.	Building Stones. (*b*)		Chalk, Marl.		Dolomite.		Flagstones.		Limestone for Flux.	
	Cubic Meters.	Value.	Cubic Meters.	Value.	Cubic Meters.	Value.	Square Meters.	Value.	Cubic Meters.	Value.
1891.....	1,737,951	$3,502,800	38,075	$28,600	1,500	$600	123,875	$76,200	197,365	$61,600
1892.....	2,502,692	3,980,800	32,610	26,600	3,100	1,800	105,793	67,400	140,800	55,800
1893.....	2,645,622	4,285,000	46,295	28,400	3,500	1,400	90,780	52,800	166,425	70,200
1894.....	2,721,399	4,345,250	33,700	24,248	(?)	.....	96,109	52,800	77,900	43,383
1895.....	2,626,438	4,528,280	100,160	31,200	(?)	.....	95,137	29,640	163,800	87,400

Year.	Marble.		Paving Stones.		Phosphate of Lime.		Plastic Clay.		Quartz Gravel.	
	Cubic Meters.	Value.	Pieces.	Value.	Cubic Meters.	Value.	Metric Tons.	Value.	Cubic Meters.	Value.
1891.....	11,350	$441,800	86,158,700	$1,517,200	291,080	$1,006,000	222,760	$279,200	465,380	$228,000
1892.....	11,750	455,400	88,048,700	1,497,800	268,210	756,000	276,855	341,800	18,100	10,40
1893.....	13,147	506,800	96,041,650	1,714,000	331,230	881,000	192,262	287,200	7,280	3,800
1894.....	11,849	383,760	84,309,000	1,390,481	371,776	848,085	172,010	241,160	6,640	3,067
1895.....	12,790	424,240	92,378,800	1,575,160	506,730	683,140	195,485	272,940	202,590	103,200

Year.	Sand.		Silica for Pottery		Slate.			Sulphate of Baryta.		Whetstones.	
	Cubic Meters.	Value.	Cubic Meters.	Value.	Pieces.	Cubic Meters	Value.	Metric Tons.	Value.	Pieces.	Value.
1891.....	328,731	$126,000	34,300	$65,400	31,569,000	220	$189,400	20,000	$16,000	*d*130,000	$24,400
1892.....	383,545	163,600	*c*38,370	48,800	33,903,000	415	224,400	42,000	56,800	*d*153,000	38,000
1893.....	389,970	130,800	32,850	40,400	32,508,500	615	212,800	41,500	58,200	100,000	9,200
1894.....	374,700	136,611	28,625	34,400	32,011,000	475	221,540	40,000	60,000	87,000	7,099
1895.....	381,170	158,120	24,870	20,000	33,652,000	690	251,720	32,750	45,860	70,000	11,120

MINERAL IMPORTS OF BELGIUM. (*e*) (IN METRIC TONS.)

Year.	Coal.		Briquettes.		Coke.		Lime.		Iron Ores, Iron and Steel Filings.		Sulphur.		Various.
1889..	1,004,624	$2,893,370	1,345	$4,008	18,545	$74,922	35,313	$77,600	1,805,250	$4,332,800	51,771	$2,070,800	$7,649,200
1890..	1,719,534	6,018,369	1,704	6,986	65,339	341,854	36,311	79,800	1,164,548	3,947,600	42,309	1,692,400	9,563,200
1891..	1,621,065	5,608,885	3,686	15,112	140,576	598,854	25,378	55,800	1,534,609	3,071,400	33,020	1,320,800	10,282,400
1892..	1,486,212	4,607,257	5,542	17,734	191,054	668,689	25,884	57,000	1,679,485	3,191,200	24,973	899,000	9,979,800
1893..	1,288,640	3,865,920	5,545	17,467	287,560	983,455	29,460	58,920	1,684,679	3,200,997	17,838	642,156	8,480,880
1894..	1,377,099	*f* 4,241.200	4,317	(*g*)	326,188	1,154,800	19,209	37,400	1,942,907	3.691,600	24,982	899,400	9,802.000
1895..	1,530,364	.....	3,452	.....	362,834	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Various Other Metals.	Stone.		Chemical Products.	Glass and Glassware. (*h*)	Salt, Coarse.		Salt, Refined.	
1889.....	$1,800,400	.....	.....	$15,093,200	$332,600	78,117	$468,400	32,320	$323,200
1890.....	1.498,000	54,433	$762,000	11,928,200	409.800	64,031	384,200	39,100	391,000
1891.....	2,159,800	59,042	826,600	13,780,400	409,800	79,422	476,600	39,586	395,800
1892.....	2,002,600	76,491	1,242,600	14,845,800	416,000	86,330	518,000	34,162	341,600
1893.....	1,622,947	49,648	747,918	16,520,549	487,560	84,524	507,145	39,561	395,612
1894.....	2.544,000	107,329	1,428,400	17,791,400	556,400	92,215	533,200	44,414	444,200

IMPORTS OF METALLURGICAL PRODUCTS INTO BELGIUM. (IN METRIC TONS.)

Year.	Steel.		Copper and Nickel.		Iron, Pig and Scrap.		Iron, Wrought.		Gold, Silver, and Platinum Jewelry	Lead.		Zinc.	
1889..	12,376	$738,800	7,550	$2,635,600	263,357	$3,027,000	24,070	$967,200	$662,800	30,312	$2,425,000	5,113	$429,400
1890..	13,517	665,600	7,386	2,580,800	264,275	3,435,600	23,136	985,200	755,200	35,985	2,878,800	5,242	440,400
1891..	13,174	599,000	8,269	2,199,800	205,482	2,401,000	25,286	987,200	678,200	34,093	2,044,600	6,972	794,800
1892..	17,172	838,000	8,222	2,056,400	174,177	1,970,600	24,321	885,200	764,200	31,054	1,614,800	5,936	617,400
1893..	29,819	1,190,643	8,025	1,968,962	186,198	2,107,443	30,817	1,169,399	874,566	38,028	1,901,420	5,833	501.632
1894..	31.587	905,600	10,526	2,380,400	252,131	2,661,600	26,714	942.000	846,600	43,491	2.087.600	9,130	675,600
1895..	35,987	.....	.....	.....	.....	.....	.....	.....	.....	45,594	.....	8,550	.....

MINERAL EXPORTS OF BELGIUM. (*e*) (IN METRIC TONS.)

Year.	Coal.		Briquettes.		Coke.		Lime.		Iron Ores. Iron and Steel Filings.		Sulphur.		Various Other Minerals
1889	4,219,700	$12,325,536	317,164	$945,129	1,219,958	$4,928,634	328,024	$721,600	157,908	$270,600	4,528	$181,200	3,563,200
1890	4,533,785	15,868,247	317,628	1,302,275	1,064,759	5,570,819	357,474	786,400	174,231	295,200	4,232	169,200	5,288,800
1891	4,750,232	16,435,803	358,691	1,470,633	933,668	3,977,426	373,671	822,000	194,933	370,200	4,071	164,800	5,958,600
1892	4,539,485	14,072,403	351,570	1,125,024	991,028	3,468,598	385,399	847,800	228,804	454,000	3,868	139,200	5,446,600
1893	4,849,887	14,549,661	489,225	1,541,059	941,663	3,220,487	404,039	808,078	172,967	339,902	4,609	165,918	6,125,735
1894	4,539,525	13,936,400	573,463	.... ..	879,278	3,112,600	411,803	803,000	260,746	509,000	3,550	127,800	3,717,200
1895	4,661,477	.....	459,702	.....	870,983	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Steel.		Copper and Nickel.		Iron, Pig and Scrap.		Iron, Wrought.		Gold, Silver, and Platinum Jewelry.	Zinc.	
1889	118,217	$6,288,400	3,751	$1,390,800	22,573	$288,800	412,453	$14,707,400	$30,200	67,489	$6,344,000
1890	109,604	5,811,200	4,671	1,692,800	23,165	301,200	356,109	13,585,000	21,600	67,026	6,300,400
1891	87,431	4,524,800	4,674	1,318,200	23,963	317,600	359,740	12,182,200	17,000	68,026	7,755,000
1892	98,867	5,189,000	4,659	1,186,800	27,748	355,000	337,999	10,752,000	29,600	76,823	7,989,600
1893	114,530	7,613,600	5,480	1,447,589	28,017	464,702	318,197	9,276,604	81,200	75,828	6,521,200
1894	156,266	11,324,800	6,792	1,624,200	20,548	248,800	335,595	10,152,200	81,200	81,248	6,012,400
1895	171,643	.....	.....	.....	.....	.....	.....	.....	.....	88,316	.....

Year.	Lead.		Various Metals.	Stone.		Chemical Products.	Glass and Glassware. (*h*)	Salt, Coarse.		Salt, Refined.	
1889....	20,070	$1,605,600	$584,200	829,615	$11,614,600	$7,039,600	$9,268,000	248	$5,400	216	$2,200
1890....	27,370	2,189,600	493,200	911,258	4,556,200	7,035,800	9,026,800	341	2,000	106	1,000
1891....	24,073	1,414,400	521,800	872,937	4,364,600	7,011,200	9,054,800	1,216	7,200	152	1,600
1892....	24,709	1,284,800	622,200	927,767	5,504,000	8,019,800	8,887,600	1,118	6,800	228	2,200
1893....	29,947	1,497,358	544,178	1,112,273	5,419,974	8,683,269	9,902,816	556	3,339	148	1,484
1894....	34,690	1,665,200	780,000	1,020,086	4,752,000	9,706,800	10,083,800	871	5,200	158	1,600
1895....	39,996	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) From *Statistiques des Mines, Minières, Carrières, et Usines Métallurgiques.* Belgium also produced, in 1891, grindstones, 90 cu. m., value $400; in 1892, grindstones, 17,100 cu. m., $1400; kaolin, 750 cu. m., $1600; in 1893, ochery earth, 880 cu. m., $1400; feldspar, 1800 cu. m., $3200; in 1894, ochery earth, 250 cu. m., $1000; feldspar, 2150 cu. m., $3560; in 1895, ochery earth, 500 cu. m., $2000; feldspar, 1900 cu. m., $1560; conglomerate rock, 245 cu. m., $6680. (*b*) Includes lime and rubble; also in 1895 includes conglomerate rock produced.

(*c*) Not including kaolin. (*d*) Not including grindstones.

(*e*) From *Annuaire Statistique de la Belgique*, except the import figures for 1893, which were furnished by Mr. E. Harzé, *Ingénieur Principal au Ministère de l'Intérieur.*

(*f*) Including the value of briquettes imported. (*g*) Value included under that of briquettes.

(*h*) Including crushed glass, of which the imports were in 1890, 820 tons, $14,800; in 1891, 841 tons, $8400; in 1892, 1178 tons, $11,800; in 1893, 2569 tons, $15,400; in 1894, 942 tons, $5600. Exports in 1890, 3217 tons, $19,000; in 1891, 3411 tons, $34,200; in 1892, 3274 tons, $32,800; in 1893, 2569 tons, $15,400.

CANADA.

THE increase in the output of coal in 1896 was altogether due to Nova Scotia, which produced 296,153 net tons more than in 1895, but this increased output was offset in a large measure by a decrease of 104,629 tons in the production of British Columbia. Copper shows a large decrease in Ontario and a small increase in Quebec; British Columbia put out 3,818,556 lbs., *i.e.*, 1,848,193 lbs. more than in 1895. In gold, large increases are to be credited to Nova Scotia and Ontario, while British Columbia produced $497,675 more than in 1895. Owing to the dull state of the iron market during the year, the production of iron ore fell off considerably in all the provinces, with the exception of Ontario, where a blast furnace was erected at Hamilton, the company obtaining all the ores mined in that province. The production of lead is entirely from British Columbia; it increased in the year by over 1,000,000 lbs., but a decrease in the market price of some 8% lessened the total value by $28,582. Natural gas shows a decrease in value of $146,761, or about 34%, which was due to a heavy falling off in the production of the Welland field. It must be noted that the values in the following table do not in several instances represent actual values at point of production, the substances in question being exported in unfinished state, but reckoned at final market value. The data used are given in the footnote.

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (*a*) (IN METRIC TONS.)

Year.	Arsenic.		Asbestos.		Baryta.		Bricks. Thousands. (*b*)		Building Material.			
									Stone—Cu. Yds. (*b*)		Flagstones. Sq. Ft.	
1892....	*Nil.*	.....	5,518	$390,462	286	$1,260	202,147	$1,251,934	219,747	$609,827	13,700	$1,869
1893....	*Nil.*	.....	5,743	310,156	*Nil.*	.....	290,000	*c*1,800,000	.....	*c*1,100,000	40,500	3,487
1894....	6.4	$420	6,935	420,825	980	2,830	.....	1,800,000	.....	1,200,000	152,700	5,298
1895....	*Nil.*	.....	7,943	368,175	8	168	*c*308,836	1,670,000	.....	1,095,000	80,005	6,867
1896....	*Nil.*	.....	11,113	429,856	131	715	.....	1,600,000	.....	*c*1,000,000	.....	6,710

Year.	Building Material—*Continued.*				Cement. Barrels.		Fireclay.		Pottery	Tiles and Sewer Pipes. (*d*)	Coal.		Coke. (*e*)	
	Granite.		Marble.											
1892....	22,047	$89,326	308	$3,600	117,408	$147,663	1,806	$4,467	$265,811	$558,517	2,986,999	$7,184,510	50,926	$160,249
1893....	20,431	94,393	535	5,100	158,597	194,015	490	700	213,186	550,000	3,481,383	8,423,759	55,409	161,790
1894....	14,901	109,936	*Nil.*	.....	108,142	144,637	489	2,167	162,144	450,325	3,508,777	8,499,141	52,646	148,551
1895....	17,453	84,838	181	2,000	128,294	173,675	1,206	3,492	151,588	467,045	3,188,408	7,727,446	48,404	143,047
1896....	16,980	106,709	67	905	147,790	201,505	605	1,427	163,905	378,875	3,395,824	8,006,305	45,196	111,560

Year.	Copper, Fine. (f)		Feldspar.		Gold—Kgms. (h)		Graphite.		Grindstones.		Gypsum.	
1892....	3,215	$826,849	159	$525	1,566	$907,601	151	$3,763	4,793	$51,187	218,679	$241,127
1893....	3,679	875,865	*g*612	5,025	1,692	976,603	*Nil.*	.....	4,173	38,379	174,698	196,150
1894....	1,242	735,017	*Nil.*	.....	1,806	1,042,055	*i*3	223	3,408	32,717	203,300	202,031
1895....	3,987	949,229	*Nil.*	.....	2,875	1,910,900	199	6,150	3,153	31,932	205,188	202,608
1896....	4,258	1,021,148	435	1,368	4,229	2,810,206	(*s*)	9,455	3,323	32,810	186,160	174,403

Year.	Iron Ore.		Chrome Iron Ore.		Pyrites.		Iron, Pig. (j)		Lead (in Ore). (k)	
1892....	93,667	$263,866	*Nil.*	.....	54,223	$179,310	38,504	$637,421	802	$72,505
1893....	113,946	299,368	*Nil.*	.....	53,108	175,626	50,754	790,283	968	78,996
1894....	99,784	226,611	907	$20,000	36,765	121,581	45,327	646,447	2,587	185,355
1895....	93,257	238,070	2,882	41,301	31,024	102,594	47,586	696,440	10,467	749,966
1896....	80,020	184,313	2,143	25,982	30,586	101,155	(*s*)	(*s*)	10,977	721,384

Year.	Lime—Bushels. (b)		Limestone for Flux.		Manganese Ore.		Mica.	Nickel—Kgms. (m)		Ochers.	
1892....	2,260,640	$411,270	20,836	$21,492	104	$10,250	*l*$104,745	1,094,837	$1,399,956	354	$5,800
1893....	6,750,000	*c*900,000	25,217	27,519	193	14,578	*l*75,719	1,806,669	2,071,151	971	17,710
1894....	(*s*)	900,000	31,843	34,347	67	4,80	*l*45,581	2,225,995	1,870,958	554	8,690
1895....	5,225,000	*c*700,000	31,370	32,916	113	8,464	65,000	1,763,836	1,360,984	1,215	14,600
1896....	(*s*)	650,000	(*s*)	(*s*)	11	900	60,000	1,587,600	1,155,000	2,041	10,925

Year.	Crude Petroleum. Barrels. (n)		Phosphate.		Platinum.	Precious Stones.	Roofing Cement.		Sand and Gravel (Exports).		Molding Sand.	
1892....	779,753	$982,489	10,825	$157,424	$3,500	$1,000	726	$12,000	270,235	$85,329	313	$1,380
1893....	798,406	834,334	7,437	70,942	1,800	1,500	863	5,441	298,569	121,795	4,291	9,086
1894 ...	829,104	835,322	6,613	43,740	950	1,500	513	3,978	294,523	86,940	5,637	12,428
1895....	728,665	1,090,520	1,653	9,565	3,800	1,650	(*s*)	3,153	251,441	118,359	6,137	13,530
1896....	726,822	1,155,646	517	3,990	(*s*)	(*s*)	(*s*)	(*s*)	(*s*)	120,000	(*s*)	(*s*)

Year.	Salt.		Silver—Kgms. (o)		Slate.		Soapstone.		Terra Cotta. (p)	Various Products. (q)	Whiting.	
1892....	41,265	$162,041	9,661	$269,489	4,699	$69,070	1,246	$6,240	$97,239	$752,610	.....	.....
1893....	56,539	195,926	13,000	330,128	6,452	90,825	640	1,920	55,704	662,977	.....	.....
1894....	51,890	170,687	26,366	534,049	.....	75,550	831	1,640	65,600	641,421	500	$750
1895....	47,515	160,455	55,230	1,158,633	.....	58,900	431	2,138	195,123	884,267	.....	.....
1896....	39,877	169,677	99,709	2,147,589	(*s*)	53,370	372	1,230	110,855	897,997	(*s*)	(*s*)

(*a*) From reports compiled by the Geological Survey of Canada. (*b*) Incomplete. (*c*) Estimated.
(*d*) Contains in 1892, 15,689 M. tiles, value $190,857; sewer pipe, value $367,660; in 1893, 100,000 M. tiles, estimated value $200,000; sewer pipe, $350,000; in 1894, tiles, $200,000; sewer pipe, $250,325; in 1895, 19,200 M. tiles (estimated), value $210,000; sewer pipe, $257,045; in 1896, tiles, $225,000; sewer pipe, $153,875.
(*e*) Oven coke, nearly all the production of Nova Scotia.
(*f*) Copper contents of ore, matte, etc., values per lb.; 1892, 11⅜c.; 1893, 10⅛c.; 1894, 9½c.; 1895, 10⅛c.; 1896, 10$^{9}_{10}$c. (*g*) Includes quartz, 91 metric tons, value $500.
(*h*) Nova Scotia and Ontario gold is computed at $19.50 per oz.; British Columbia gold and that of the Northwestern Territories, including the Yukon district, at $17; Quebec gold at $18 per oz. For 1894 and 1895 the quantity given is in fine metal. (*i*) Export returns.
(*j*) Native iron ore converted into pig iron in 1892, 87,950 metric tons, value $250,966; in 1893, 112,539 metric tons, value $296,979; in 1894, 98,766 metric tons, value $223,861; in 1895, 84,558 metric tons, value $218,336.
(*k*) Lead contents of ores in 1892 at 4.1c. per lb., in 1893 at 3.7c. per lb., in 1894 at 3¼c. per lb., in 1895 at 3¼c. per lb., and in 1896 at 2.98c. per lb.
(*l*) Exports, plus quantity sold to Canadian electrical and stove manufacturers.
(*m*) In 1892, exports of fine nickel in matte, ore, etc., valued at 58c. per lb. (In matte at Sudbury the values of the metals would be nearer 6c. for copper and 18⅜c. for nickel.) Total value at Sudbury of the matte shipped during 1892, $450,560. In 1893 at 52c., in 1894 at 38½c., in 1895 at 35c., and in 1896 at 33c. per lb., which represent the final market values of the nickel. In matte its value at the mine would be about 13c.
(*n*) Calculated from the inspection returns at 100 gals. crude to 38 gals. refined oil. The value of the crude was in 1891, $1.33 per bbl. of 35 imperial gals.; in 1892, $1.26; in 1893, $1.04½; in 1894, $1.00¾; in 1895, $1.49⅜; in 1896, $1.59 per bbl. 1 barrel refined = 42 imperial gals.
(*o*) Silver contents of ore; values for production and exports: 1892, $0.86⅜; 1893, 0.76^{9}_{10}$; 1894, $0.63; 1895, $0.65; 1896, $0.67 per oz.
(*p*) In 1893 includes porous fireproof terra cotta; in 1894 includes porous fireproof terra cotta and other structural and ornamental forms.
(*q*) Estimated value of products not returned in 1892, not including natural gas, value $150,000, and mineral water, 640,380 gals., value $75,348; 1893 includes 5.3 metric tons of zinc, value $470 (zinc contents of ore at 4c. per lb.); natural gas, $366,233; 1894 includes 163 metric tons lithographic stone, value $30,000; also natural gas, value $313,754, being the gross amount received through sale; 1895 includes mercury, $2343; lithographic stone, $2000; natural gas, $423,032, and 739,382 gals. mineral water, value $126,048; in 1896, 602 metric tons tripolite, value $9960; 706,372 gals. mineral water, value $111,736; natural gas, $276,301. (*s*) Not reported.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (a) (IN METRIC TONS.)

Year. (h)	Brass. (f)	Cement	Clays.	Bricks and Tiles.	Earthenware and China.	Coal, Anthracite. (b)		Coal and Coke.		Copper (f)	Glass.	Gold and Silver. (f)
1892..	$537,105	$287,729	$82,619	$81,495	$748,810	1,341,824	$5,640,346	1,579,237	$4,333,490	$431,427	$1,257,858	$261,471
1893..	514,171	327,148	70,565	89,750	709,737	1,361,277	6,355,285	1,591,711	4,168,515	475,714	1,219,543	298,439
1894..	453,671	284,471	70,731	167,137	695,514	1,388,468	6,354,040	1,405,656	3,515,845	281,801	1,209,203	283,645
1895..	527,454	251,926	62,730	123,461	547,935	1,274,019	5,350,627	1,626,681	3,376,517	258,214	1,120,986	341,476
1896.	477,279	255,029	62,984	193,594	575,493	1,428,255	5,667,096	1,748,875	3,352,767	294,410	1,104,481	340,241

Year. (h)	Coin and Bullion.	Gunpowder	Iron and Steel. (f)	Lead. (f)	Metal Manufactures.	Mineral Oils. Gallons.		Precious Stones and Jewelry.	Salt.		Stone and Marble (f)	Tin. (f)
1892..	$1,818,530	$136,171	$12,625,422	$317,142	$373,819	5,793,636	$494,004	$344,827	100,008	$380,958	$276,005	$1,594,205
1893..	6,534,200	143,028	13,199,523	288,636	353,225	6,249,946	473,279	366,623	96,595	361,300	231,497	1,274,866
1894..	4,023,072	112,781	11,310,771	203,644	317,145	6,666,323	446,492	427,796	96,406	381,636	227,356	1,310,389
1895..	4,576,620	142,305	9,249,749	192,046	302,874	6,852,425	442,406	490,076	95,342	362,592	200,786	972,078
1896..	5,226,319	136,818	10,952,607	239,109	307,991	7,130,086	628,383	612,741	96,467	363,438	219,298	1,236,640

IMPORTS—*Continued.*

MINERAL EXPORTS OF DOMESTIC PRODUCE FROM THE DOMINION OF CANADA. (a) (IN METRIC TONS.)

Year. (h)	Whiting.	Zinc.	Various Metals.	Various Minerals.	Asbestos.	Agric. Implements.	Barytes	Brick, Thousands	Cement	Cinders.	Clay, M'f's of.	Coal.
1892..	$26,867	$127,302	$199,777	$107,103	6,637	$514,412	$402,778	....	504	$2,750	$1,495	$5,451	120	857,403	$3,195,467
1893..	25,563	124,360	198,783	100,332	5,351	396,718	462,253	....	3,625	24,560	894	4,118	113	823,934	3,114,558
1894..	26,649	96,882	244,486	125,756	5,651	339,706	465,682	...	4,502	33,013	1,268	2,941	67	903,557	3,321,565
1895..	24,441	68,954	179,776	96,385	7,796	493,075	663,718	$900	1,417	6,774	443	3,163	671	1,007,506	3,578,195
1896..	27,322	87,074	190,551	130,463	8,698	482,679	593,464	168	1,216	7,788	1,035	2,125	496	1,929,934	3,249,069

Year. (h)	Coke.		Charcoal.	Coin and Bullion	Copper Ore.		Copper Matte, Regulus, etc.		Copper, Fine.		Explosives.		Glass and Glassware.	Grindstones	Gypsum, Crude.		Iron and Steel, M'f's of
1892..	..	....	$46,817	$306,447	308	$30,755	...	.....	1,723	$185,848	75	$61,763	$1,964	$23,465	161,949	$194,304	$243,857
1893..	..	....	48,700	309,459	64	3,850	326	$391,969	.....	.....	58	56,764	4,648	25,702	160,108	178,979	316,454
1894..	2	$10	33,191	310,006	(c)	.....	(c)	.....	(c)	.....	35	29,050	4,433	19,330	147,337	160,082	295,924
1895..	24	108	21,823	256,571	(c)	.....	(c)	.....	(c)	.....	173	98,083	4,738	12,042	145,967	156,897	308,711
1896..	17	74	36,086	207,532	(c)	.....	(c)	.....	(c)	.. ..	120	92,763	6,881	18,853	182,217	205,641	506,946

Year. (h)	Lime.	Metals Not Iron and Steel, M'f'es of	Mica, Crude and Cut.		Mica, Ground	Mineral Oils. Gallons.		Nickel, in Ore.		Antimony.		Ores.				
												Gold Quartz, etc.	Iron.		Lead.	
1892..	$124,152	$17,495	618	$63,708	$4.758	440,906	$18,217	6,911	$617,639	4	$60	$316,177	6,992	$36,935	27	$5,200
1893..	106,105	24,113	292	92,529	4,371	178,101	6,814	4,048	427,557	..	....	247,868	7,086	26,114	68	2,329
1894..	71,171	24,169	155	26,553	.....	68,740	2,722	3,815	808,790	..	....	318,258	1,678	9,026	1,503	65,337
1895..	83,595	18,932	335	46,894	575	63,543	3,572	3,647	599,568	..	....	612,729	4,290	43,088	8,191	333,763
1896..	76,451	31,308	251	55,514	113	18,241	2,971	3,473	486,651	..	....	1,099,053	2,674	39,999	9,209	408,625

Year. (h)	Ores.				Phosphates.	Fertilizers.	Plumbago.		Salt.		Sand and Gravel.		Slate.		Stone.	Tin, M'f'es
	Manganese.		Silver,													
1892..	150	$6,106	295	$193,441	15,643	$380,462	$8,763	58	$438	83	$763	208,445	$60,285	57	$1.507	$60,209	$5,115
1893..	132	13,596	379	65,406	10,786	132,475	7,706	227	e3,586	102	955	303,031	117,760	131	2,654	44,733	6,160
1894..	45	4,353	d20	423,707	4,488	40,400	31.413	...	10	127	1,280	274,664	96,453	67	1.390	46,833	6,151
1895..	116	7,693	32	651,737	3,800	33,810	32,174	4	291	137	1,136	273,924	90,093	201	3,964	60 .05	9,531
1896..	79	1,988	71	1,595,548	479	5,280	36,137	94	10,771	76	767	259,585	119,847	275	8,983	49,669	5,777

(a) From the tables of the *Trade and Navigation of the Dominion of Canada.* The imports figures are for home consumption only. The exports are those of domestic produce. (b) Including anthracite coal dust. (c) In 1894 was exported copper contents of ores, matte and regulus, and fine copper, 541 tons, value $88,352; in 1895, 1722 tons, value $222,657; in 1896, 1788 tons, value $194,771. (d) Silver contents of ores. (e) Also exported in 1893, plumbago, manufactures, value $198. (f) And manufactures thereof. (h) Fiscal years ending June 30.

THE MINERAL PRODUCTION OF CANADA BY PROVINCES.

Besides the statistics which are collected for the whole Dominion by the Geological Survey of Canada, the Bureaus of Mines of several of the provinces issue excellent reports on their mineral production. For British Columbia, Ontario, and Nova Scotia, in which provinces the principal mines of the Dominion are situated, these reports are especially good. The Ontario report covers the entire production of nickel in Canada, the only mines producing this metal being situated in the Sudbury district. The British Columbia report includes practically all the production of lead, the mines producing this metal in other provinces being insignificant. The entire Canadian production of petroleum is derived from the Petrolia field in the Province of Ontario.

Canada has not yet a metallurgical industry of much consequence, and its statistics of metal production are based upon the metal contents of ores and mattes exported. The copper-nickel ore of Sudbury is smelted to matte in works near the mines, which is exported to England and the United States, chiefly to the latter, and its metallic contents consequently enter the markets in those countries. Similarly the copper ore of British Columbia is smelted to matte in local works, and the matte is shipped to the United States, largely to Butte, Mont. The British Columbia lead ore is mostly exported to the United States in crude form. A good deal of it is smelted in bond at Denver, Colo., and its lead. after desilverization, is then shipped to Liverpool by way of Galveston, Tex. Practically all of the chrome ore and mica reported in the Dominion statistics is produced in the Province of Quebec.

MINERAL PRODUCTION OF BRITISH COLUMBIA. (*a*) (IN METRIC TONS.)

Year.	Coal.		Coke.		Copper.		Gold—Kgms.				Lead. (*b*)		Silver—Kgms.	
							Lode.		Placer.					
1887....	419,974	$1,240,080	....	.....	.....	.....	.....	.....	.....	$693,709	.....	.....	550	$17,331
1888....	497,130	1,467,903	....	.....	.....	.....	.....	.....	.....	616,731	.....	.....	2,481	75,000
1889....	589,107	1,739,490	....	.....	.....	.....	.....	.....	.....	588,523	.....	.....	1,654	47,873
1890....	688,990	2,034,420	....	.....	.....	.....	.....	.....	.....	490,435	51	$5,805	2,190	73,948
1891....	1,045,563	3,087,291	....	.....	.....	.....	.....	.....	.....	429,811	267	25,607	140	4,000
1892....	839,556	2,479,005	....	.....	.....	.....	.....	.....	.....	399,526	802	72,505	2,400	66,935
1893....	993,947	2,934,882	....	.....	.....	.....	36	$23,404	.....	356,131	968	78,996	7,060	195,000
1894....	1,029,160	3,038,859	.....	.....	147	$16,234	194	125,014	.....	405,516	2,568	169,875	23,214	470,219
1895....	954,688	2,818,962	459	$2,260	432	47,642	1,221	785,271	749	481,683	7,473	532,255	46,546	977,229
1896....	859,775	2,327,145	625	3,075	1,732	190,926	1,936	1,244,180	846	544,026	10,977	721,384	97,519	2,100,689

(*a*) From the *Annual Report of the Minister of Mines.* (*b*) Attention should be called to the discrepancy between certain of these figures and those of the Canadian Geological Survey; practically the total production of lead in the Dominion is from British Columbian ore.

MINERAL PRODUCTION OF NOVA SCOTIA. (*a*) (IN METRIC TONS.)

Year.	Coal.	Coke.	Copper Ore.	Gold. Kgms.	Grindstones, etc.	Gypsum. (*c*)	Iron Ore. (*b*)	Lead Ore.	Limestone.	Manganese Ore.	Moulding Sand. (*c*)
1892....	1,973,864	55,880	24	622.0	$11,792	146,125	68,040	0.907	18,144	101	159
1893....	1,709,636	52,438	1,134	436.7	6,971	89,130	60,635	*Nil.*	18,765	103	.....
1894....	2,235,438	60,592	.....	621.5	6,581	96,318	75,762	*Nil.*	27,216	22	.....
1895....	2,122,673	44,361	.....	687.8	17,189	135,433	80,910	*Nil.*	30,659	112	.....
1896....	2,271,239	59,681	10	796.0	30,317	132,577	57,235	*Nil.*	31,669	131	.....

(*a*) From the *Annual Reports of the Department of Mines*, Nova Scotia. The official figures for 1893 are for nine months only, according to the provisions of the Acts of 1893, which require that all reports of the several departments of the government shall in 1893 be made up for a period of nine months ending September 30. From 1893 the fiscal year of the Province begins on the first day of October and ends on the 30th of September, and all reports must be made up for the year so beginning and ending.
(*b*) Not including imported ore. (*c*) Amount exported.

MINERAL PRODUCTION OF ONTARIO. (a)

Year.	Brick—Number.						Building Stone.					
	Common.		Pressed.				Dimension Stone. Cubic Feet.		Heads and Sills—Cu. Ft.		Coursing Stone—Sq. Yds	
			Plain.		Fancy.							
1891....	160,000,000	$950,000	12,647,909	b$156,699	570,000	(c)	(d)	.....	(d)	.....	(d)	.....
1892 ..	175,000,000	980,000	20,342,000	198,350	1,323,000	$32,253	2,600,000	$680,000	50,000	$26,000	64,000	$42,000
1893....	162,350,000	932,500	20,208,000	b217,373	1,373,700	(c)	1,400,000	260,000	44,700	21,000	170,000	180,000
1894....	131,500,000	690,000	22,460,000	198,510	2,896,000	34,160	1,340,000	360,470	47,070	15,900	22,000	36,000
1895....	126,245,000	705,000	15,253,370	115,695	2,312,497	24,075	(d)	.....	(d)	.....	(d)	.....

Year.	Building Stone—*Continued.*			Calcined Plaster, etc. Metric Tons.		Cement—Barrels.				Cobalt. Metric Tons		Copper. Metric Tons.	
	Rubble, etc. Cubic Yards.		Total Value.			Natural Rock.		Portland.					
1891....	(d)	.....	$1,000,000	e4,445	$50,800	46,178	$39,419	2,033	$5,082	(h)	.....	(h)	.....
1892....	730,000	$132,000	880,000	f2,258	35,690	54,155	38,580	20,247	47,417	7.7	$3,713	1,756	$232,135
1893....	410,000	260,000	721,000	g 145	15,000	74,353	63,567	31,924	63,848	17.3	9,400	1,298	115,200
1894....	223,000	142,000	554,370	1,036	22,697	55,323	48,774	30,580	61,060	2.9	1,500	2,493	195,750
1895....	(d)	.....	438,000	402	13,095	55,219	45,145	58,699	114,332	.....	.....	2,146	160,913

Year.	Gold.			Gypsum. Metric Tons.		Lime—Bushels.		Mica. Metric Tons.		Natural Gas. 1,000 Cubic Feet.		Nickel. Metric Tons.	
	Ore. M. Ton	Metal Kgms.	Value.										
1891....	i1,814	(h)	.....	4,853	$12,200	2,350,000	$300,000	218.0	$31,200	(h)	.....	i77,828	$324,240
1892....	3,320	(h)	$36,900	3,510	14,100	2,600,000	350,000	6.3	1,500	(h)	$160,000	1,888	590,902
1893....	5,044	52.7	32,960	2,556	7,363	2,700,000	364,000	63.5	8,600	2,342,000	238,200	1,499	454,702
1894....	2,202	62.9	32,776	2,951	9,760	2,150,000	280,000	(h)	.....	1,653,500	204,179	2,332	612,724
1895....	5,896	94.2	50,281	3,059	7,471	2,090,000	280,000	22.6	2,900	3,320,000	282,986	2,100	404,861

Year.	Petroleum.										
	Crude. Imperial Gallons. (j)		Refined Products. (j)								
			Illuminating Oil. Imp. Gals.		Lubricating Oil. Imp. Gals.		All Other Oils. Imp. Gals.		Paraffine Wax. Metric Tons.		Fuel Products
1891....	31,312,645	$1,209,558	(h)	.....	(h)	.....	(h)	.....	(h)	.....	(h)
1892....	28,000,000	1,000,000	10,862,894	$919,315	3,457,570	$138,304	7,654,723	$272,577	29.4	$70,239	(h)
1893....	34,055,000	1,099,868	13,322,320	1,372,209	4,239,847	277,500	11,220,705	323,156	102.0	143,325	$72,500
1894....	34,912,360	1,094,852	14,349,472	1,337,040	3,817,181	242,688	10,632,141	343,416	1,249.0	152,467	71,326
1895....	33,351,997	1,403,960	10,924,826	1,237,328	2,400,404	205,591	7,081,717	285,308	891.0	86,608	79,589

Year.	Pottery.	Salt. Metric Tons.		Sewer Pipe.	Silver Ore. Metric Tons.		Terra Cotta.	Tiles—Thousands.				Total Value.
								Drain.		Roofing.		
1891....	$45,000	40,068	$157,000	k$270,000	13,539.0	$64,475	(l)	7,500	$90,000	400	(c)	$4,705,673
1892....	80,000	39,360	162,700	(h)	9.07	732	$20,119	10,000	100,000	383	$8,613	5,374,139
1893....	115,000	43,863	149,850	230,000	453.6	2,500	(c)	17,300	190,000	53	(c)	6,120,753
1894....	134,000	31,947	115,551	207,000	*Nil.*	.....	52,360	25,000	280,000	100	1,200	6,086,758
1895....	108,000	46,275	188,101	133,159	*Nil.*	.....	38,500	14,330	157,000	375	6,200	5,170,138

(a) From the *Annual Reports of the Bureau of Mines of Ontario*, Toronto. In 1894 there was also produced sand and gravel 733,500 cubic yards, value $203,450.

(b) Comprising the values of plain and fancy pressed brick, roofing tile, and a quantity of ornamental and porous terra cotta.

(c) Value included with that of plain pressed brick.

(d) Quantity not stated, and value included in the total of building stone.

(e) Including phosphate of lime only.

(f) Comprising 2160 metric tons of phosphate of lime, value $23,810, and 98 metric tons of alabastine and plastico, value $11,880.

(g) Consisting of 127 metric tons of alabastine and plastico, value $14,800, and 18 metric tons of phosphate of lime, value $200.

(h) Not reported.

(i) Ore raised.

(j) One barrel of crude oil is reckoned at 25 imperial gallons, and 1 barrel refined at 42 imperial gallons.

(k) Number produced, 1,375,000.

(l) In 1891 about 782,091 pieces of terra cotta were manufactured; value included with that of plain pressed brick.

CHILE.

DEVELOPMENT of the mining industry in Chile is retarded by various conditions, important among which is the lack of railway communication. This, however, is gradually being improved. Recently the government purchased the railway from Coquimbo to Ovalle, with the branch to Panulcillo, and by the prolongation of the road from Ovalle to Huatulame and the lowering of freight rates which was immediately ordered, the mineral industry of the tributary district has already benefited much. The reconstruction of the railway from Serena to Elqui and Rivadavia, which is now the railway of the greatest perspective in Chile, is in progress. This will open the great deposits of manganese in the quebrada of Marqueza, and the recently discovered gold and silver mines of Rio Seco, which promise to be extensive and lasting. There are also numerous mines of copper in the Elqui district, two of which, recently discovered, seem to be good.

A second difficulty in mining in Chile is the scarcity of laborers, which is due partly to the customs of the country, whereby there are long and frequent holidays and fiestas. Another cause of the insufficiency of labor is drunkenness, which the government takes no measures to correct. A third cause is in the lack of powerful explosives, wherefore a greater number of men is required than if they were available. At the present time no dynamite is made in Chile, and the importations are subject to a revenue impost, which together with the high cost of transporation makes it very expensive in the mining districts.

Another great drawback in the mining industry of Chile is the high price of coal. In 1895 this ranged between 12 and 14 Chilean dollars* per metric ton. The Chilean coal is lignite, and although of good quality has less calorific power than English or Australian coal, the former being about 25% and the latter about 20% better.

The Chilean lignite occurs in the Tertiary formation of the south of the country, the coal-bearing measures resting on crystalline stratified rocks, mica schist and slate. Immediately over these rocks occurs a sheet of conglomerate and then in order are sandstones of distinct colors, combined with sandy and clayey beds, and finally pure clay. The number of coal seams is few and their strength is little. The lowest seam which is about 30 cms. thick is inexploitable; above this there is a seam which attains 120 cms. thickness; and a short distance higher is a seam varying from 90 cms. to a meter in thickness.

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* At present $1 Chilean = 18d. sterling; or $1 American = 4s. 1½d. = $2.74 Chilean.

The deposits of the Bay of Arauco, and especially the well known one of the Compañia de Lota are types of this formation. The principal mines are situated around the Bay of Coronel, which is a part of the Bay of Arauco. The seams on which they are opened dip under the sea, whither they have now been followed.

Although the number of seams in the Chilean coal formation is few, its extension is very considerable. On the other side of the Cordillera de Nahuelbuta there are seams known in Angol, Mulchen, Nacimiento, and near Imperial. In the Province of Valdivia seams are known in the interior near the Calle-Calle and Futa rivers; also in the valley of the great river Maullin, in the Province of Llanquihue, in Pargas, in Chiloé and in Magallanes. In none of these places has there been any exploitation yet, though it is from them without doubt that Chile must look for a cheaper coal supply. In the center and in the north of Chile there are insignificant seams, probably of the Jurassic formation. In the Province of Atacama there is a vertical seam of anthracite, which is, however, valueless on account of its high percentage of ash. Since the Carboniferous formation does not exist in Chile there is no prospect that true coal will be found within its borders.

The latest and only statistics of the production of coal in Chile are those collected by the Sociedad Nacional de Mineria in 1894. Correcting an error in these, the production in that year was 771,241 metric tons, of which 531,019 were mined under the sea.

The importation of coal into Chile in 1894 was 775,639 metric tons; in 1895 it was 574,063 metric tons. The statistics of imports of coal include coke. The annual consumption of coal in Chile at the present time is about 1,245,934 tons, which is divided approximately as follows: By the State railways, 200,000; by the navy, 50,000; by gas works, 70,000; coke by smelteries, 35,000; by the nitrate industry, 150,000; by private railways, 150,000; by mines and smelting works, 200,000; by coastwise navigation and other industries, 390,934. These figures are probably not exact, however, since a good deal of the coal entered as consumed in Chile is in reality probably used in Bolivia.

The more part of the silver produced in Chile is from deposits in the sedimentary Jurassic and Liassac formations, the principal districts being Rodeitos, Argueros, Agua Amarga, and Tunes, Chañarcillo, Tres Puntas, La Florida, Esmeralda, Caracoles, Huantajaya, and Santa Rosa. These districts have been thoroughly explored, except that part of Huantajaya from La Tuana to the plain of the interior. Besides these mines in the Jurassic formation there are others in the andesites and trachytes of the Andes. Among these the Condoriaco and Lomas Bayas districts are auriferous. The Cordilleras contain also many silver-copper veins. In the Province of Coquimbo a gold-silver mine was discovered about a year ago in the Cordillera de Rio Seco, which promises to be one of the largest producers in the country. The reconstruction of the railway from La Serena to Rivadavia will soon give it much cheaper freight than it has at present.

Gold is produced in Chile from both placers and lodes. Of the latter the most important are in the Province of Antofagasta; in the Huanaco district in Taltal; on the coast of Paposo and Taltal; in the Inca, Jesus Maria, Cachiyuyo and other districts in Atacama; in the Torno, Andacollo, and Talca districts in Coquimbo; in Illapel, Hierro Viejo, Petorca, Peldehül, Tiltil, and Casablanca.

The low price of silver has diverted attention to gold-mining, and new plants for beneficiating the ores have been erected, among others cyanide works in Alhué and Andacollo. The gold mining industry of Chile suffers, however, from lack of capital, most of the undertakings being by individuals or by very small companies. The only placers where hydraulicking was in use have been abandoned. There are many opportunities in Chile for the application of pyritic smelting, many veins of gold and silver-bearing sulphide ores, containing a little copper, being known. Encouraging experiments have already been made with this method at the Playa Blanca works in Antofagasta, Chile, and also by a French chemist, Ferdinand Gautier, at Potosi, Bolivia. The latter is now erecting a furnace of American model at Poopó, Bolivia.*

MINERAL EXPORTS OF CHILE. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Borax.		Calcium Borate.		Coal.		Copper Bars.		Copper Matte. (*b*)		Copper Matte, Argentiferous. (*b*)	
1891....	815	$106,969	6,463	$318,080	.....	$2,053,988	17,464	$4,872,964	2,838	$338,558	1,666.0	$681,776
1892....	.....	.....	.....	.....	215,140	1,535,029	19,144	5,076,922	.....	.....	1,531.3	301,239
1893....	.....	.....	.....	.....	209,711	1,677,904	19,328	4,844,353	.....	.....	.....	.....
1894....	.....	.....	6,595	.....	211,532	1,299,600	19,640	.....	.....	.....	9.3	.....
1895....	.....	.....	4,355	.....	198,237	1,235,725	20,042	.....	417	.....	664.0	.....

Year.	Copper Ores. (*c*)		Copper-Silver Ores. (*d*)		Gold—Kgms.		Gold Ores.		Gold and Silver Ores.		Iodine.	
1891....	5,860	$203,403	47.0	$4,694	599.00	$479,789	288	$69,142	87.00	$65,343	424.02	$4,240,190
1892....	.....	.....	92.5	11,197	854.55	683,638	97	19,010	2.73	961	513.85	5,138,460
1893....	.....	.....	.....	.....	954.54	602,761	.....	.....	.....	.....	595.34	5,953,380
1894....	2,029	.....	13.5	.....	.....	928,115	.....	.....	.....	.....	323.25	.....
1895....	6,963	.....	86.4	.....	.....	750,310	.....	.....	.....	.....	144.25	.....

Year.	Manganese Ores.		Nitrate of Soda.		Silver Bars.		Silver, Old Plate		Silver-Lead Ores.		Silver Ores.	
1891....	35,610.4	$356,106	891,727.1	$32,418,491	115,630.38	$4,506,771	.....	$250,512	126	$74,338	743	$340,312
1892....	51,685.1	516,851	797,827.0	31,785,060	138,845.38	5,017,390	1,147.88	311,535	....	.....	915	418,077
1893....	36,740.9	367,410	943,570.0	39,211,913	140,368.60	4,418,072	.....	.....	....	.....	330	206,996
1894....	47,993.8	380,000	1,081,337.0	35,410,950	.....	.....	.....	.....	....	.....	370	317,800
1895....	24,075.1	191,085	1,220,427.0	36,043,400	.....	.....	.....	.....	....	.....	2,137	832.050

MINERAL IMPORTS OF CHILE. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Coal.	Iron.							Mineral and Vegetable Oils.
		Bar and Pig.	Galvanized.	Pipings.	Rails.	Wire.	Steel, Bar and Sheet	Total Value.	
1891.....	$4,113,770	.....	.....	.....	.....	.....	.....	$512,668	.....
1892.....	3,078,044	.....	.....	.....	.....	.....	.....	658,586	.....
1893.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1894.....	3,766,715	$537,060	$998,115	$588,485	$558,310	$341.000	$306,455	.....	$696,950
1895.....	3,624,300	611,535	569,510	1,092,280	819,575	417,255	683,180	.....	1,088,065

(*a*) From the *Boletin de la Sociedad Nacional de Mineria*, 1892, p. 46; from *Estadistica y Geografia de la Republica de Chile*, Santiago, 1894, and from *Deutsches Handels Archiv*, 1894. Exports and imports for 1894 and 1895 from *British Consular Report No. 1795*. (*b*) 50% copper. (*c*) 20% copper. (*d*) 15% copper; 0.63% silver. (*f*) Copper, silver, and gold matte, 188.1 metric tons, $40,037, and copper, silver, and gold ore, 219.1 tons, $26,356, were also exported during 1892.

*This review of the mineral industry in Chile is based on a paper by Señor Don Alberto Herrmann in the *Boletin de la Sociedad Nacional de Mineria*, Dec. 31, 1896. Reference should also be made to the articles on "Copper" and "Nitrate of Soda" elsewhere in this volume.

CHINA.

THERE is prospect that the development of the mineral resources of China, which are now lying fallow, will soon be undertaken, though progress is bound to be slow. Probably the exploitation of the iron ore deposits will be the first to be commenced on a large scale. Recently an arrangement has been effected for opening those in the Province of Honan, and the erection of blast furnaces and steel-works at Tsinghauchen, prefecture of Weihuifu.

There are no statistics of the mineral production of China, official or unofficial. Its importance and extent can only be judged from the statements of travelers, and from the reports of imports and exports, of which the latest official returns are embodied in the following tables:

MINERAL IMPORTS OF CHINA. (*a*) (IN METRIC TONS.)

Year.	Coal.	Copper, Sheet.	Copper, Old.	Gypsum.	Iron, Bar, Hoop.	Iron, Nail, Rod.	Iron, Manufactured.	Iron, Pig.	Iron, Old.	Iron, Wire.
1890.....	100,736	574	970	11,000	4,147	16,080	4,180	588	17,979	1,022
1891.....	102,403	750	926	7,658	5,041	18,120	5,290	408	22,656	1,110
1892.....	251,195	822	834	9,326	4,431	17,316	4,400	316	19,660	1,308
1893.....	186,796	610	540	9,858	3,855	15,270	3,867	396	16,316	1,325
1894.....	107,872	161	147	2,488	6,076	14,930	972	1,190	24,516	2,563

Year.	Kerosene. Gallons.	Lead, Pig.	Lead, White.	Lead, Yellow.	Quicksilver.	Salt.	Steel.	Tin.	Tin, Slabs.	Tin, Plates.	Yellow Metal.
1890.....	15,384,000	7,719	8	7	58	3,729	1,189	.....	2,366	154	537
1891.....	23,644,000	8,334	9	11	98	9,570	1,660	29	2,209	113	606
1892.....	23,286,000	8,604	13	12	154	8,976	1,557	73	2,100	187	602
1893.....	34,764,000	8,520	8	9	107	12,798	2,015	17	2,616	258	940
1894.....	45,141,000	9,644	181	23	45	15,379	3,171	.....	3,326	652	1,398

MINERAL EXPORTS OF CHINA. (*a*) (IN METRIC TONS.)

Year.	Alum, White.	Coal.	Copper, Cash. Strings.	Copper, Tribute.	Gypsum.	Ironware.	Steel.	Tin.
1890.....	3,199	111,016	2,440	.....	8,046	2,836	305	65
1891.....	4,005	105,440	26,800	927	6,894	2,841	240	126
1892.....	4,760	140,967	6,091	618	8,670	2,469	207	73
1893.....	5,511	131,773	464	618	8,956	2,812	164	16
1894.....	4,056	137,978	617	660	4,213	1,849	329	20

(*a*) From special tables, statistical series of the *Imperial Maritime Customs.*

IMPORTS AND EXPORTS OF GOLD AND SILVER OF CHINA. (*a*)
(1 Haikwan-tael = $1.)

IMPORTS.			EXPORTS.		
Year.	Gold.	Silver.	Year.	Gold.	Silver.
1893.....	$5,142,814	$49,515,467	1893.....	$12,643,783	$37,612,051
1894.....	4,165,729	56,152,750	1894.....	12,859,040	34,263,962

EXPORTS OF GOLD FROM KOREA. (*a*)
(1 Haikwan-tael = $1.)

EXPORTS.			
Year.	Gold.	Year.	Gold.
1893.....	$918,659	1894.....	$950,973

MINERAL IMPORTS, EXPORTS, AND RE-EXPORTS OF AMOY. (*a*) (IN METRIC TONS.)

	IMPORTS.						EXPORTS.			RE-EXPORTS.	
Year.	Iron, Bar, Nail-Rod.	Iron, Old.	Kerosene. Gallons.	Lead, Pig.	Quicksilver.	Tin, Slabs.	Bricks and Tiles, Pieces.	Chinaware.	Iron, Pans, Ironware.	Lead, Pig.	Tin, Slabs.
1890....	55	395	906,900	174	22	393	5,434,172	906	448	168	270
1891....	78	546	1,181,920	174	21	395	3,131,719	1,120	499	118	240
1892....	55	512	1,895,080	266	18	462	2,855,712	1,243	529	180	174
1893....	54	778	1,533,290	200	13	435	4,357,208	1,070	639	150	156
1894....	50	496	4,332,465	329	11	724	3,129,623	1,272	509	83	166

MINERAL IMPORTS OF SHANGHAI. (*d*) (IN METRIC TONS.)

Year.	Coal. (*b*)	Copper	Iron.	Lead.		Kerosene. (*c*)	Quicksilver.	Steel.	Tin.		Yellow Metal and Nails.	Zinc in Sheets.
				Pig.	Sheets.				Slabs.	Plates.		
	Tons.	Tons.	Tons.	Tons.	Tons.	Cases.	Kgms.	Tons.	Tons.	Tons.	Tons.	Tons.
1892....	.....	1,571	59.468	5,366	52	2,092,567	90,024	1,265	2,440	725	53	283
1893....	396,947	1,577	48,386	4,799	49	2,559,500	31,460	929	2,170	479	263	267
1894....	420,556	2,032	51,591	5,854	.....	3,560,989	21,910	996	2,683	2,097	278	218
1895....	.....	.....	41,646	5,870	.....	1,504,541	35,013	1,523	2,469	3,004	205	408
1896....	.....	1,837	58,421	4,537	.....	3,830,400	27,442	3,312	3,303	1,316	287	1,132

MINERAL IMPORTS OF KOREA. (*a*) (IN METRIC TONS.)

Year.	Brass Metal, etc.	Coal.	Copper, Pigs and Slabs.	Copper, Nails, Wire.	Copper Ore.	Iron, Nail, Rod.	Iron, Bar, Hoop.	Iron, Old.	Iron, Nails.	Kerosene. Gallons.
1890.....	54	401	346	18	.....	72	176	387	65	415,325
1891.....	55	1,291	1,148	37	.....	9	221	296	42	540,390
1892.....	115	1,373	653	18	198	31	192	260	92	724,563
1893.....	109	791	1,064	13	215	62	284	451	90	936,000
1894.....	59	390	248	5	245	.....	.....	372	73	903,500

Year.	Lead.	Metals, Manufactured.	Nickel.	Quicksilver.	Salt.	Spelter, Zinc.	Steel.	Tin, Slabs.	Tin Compounds.	Tin, Plates.	White Metal.
1890.....	.....	180	.....	4.32	146	27	89	63	32	38	28
1891.....	240	360	.....	3.62	1,305	828	86	49	28	31	28
1892.....	46	210	0.96	2.46	1,625	475	45	67	31	17	16
1893.....	142	150	1.14	3.60	2,652	1,116	71	75	17	48	24
1894.....	24	75	.....	1.02	5,637	41	.....	49	11	.....	8

(*a*) From special tables, statistical series of the *Imperial Maritime Customs.*
(*b*) From Wheelock & Co.'s report, Shanghai. (*c*) 1 case holds 10 gallons.
(*d*) From Alex. Bielfeld's reports, Shanghai. Tons converted from piculs (1 picul = 0.06048 metric ton).

FRANCE.

THE French mineral statistics are invariably tardy in appearance, and those for 1894 are the latest complete figures that have been issued. There are official figures for a few of the more important substances in 1895. For the production of iron, steel, coal, and zinc in France in 1896 reference should be made to the respective captions elsewhere in this volume.

MINERAL PRODUCTION OF FRANCE. (*a*) (IN METRIC TONS; 5f. = $1.)

Year.	Alunite.		Antimony Ore.		Bauxite.		Bituminous Substances. (*b*)		Copper Ore.		Manganese Ore	
1891	.....	.....	5,316	$133,518	22,260	$190,750	260,626	$338,889	16	$1,625	15,343	$90,316
1892	.....	.....	5,100	135,200	23,298	196,524	224,311	335,600	251	6,078	32,406	205,074
1893	253	$2,530	7,200	130,000	33,923	261,248	222,000	322,400	.....	.....	38,080	290,073
1894	354	3,540	6,144	81,231	26,032	32,588	230,603	351,561	.....	.....	32,751	200,277
1895	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Iron Ore.		Iron Pyrites.		Mineral Fuel.					
					Coal.		Lignite.		Peat.	
1891	3,579,286	$2,557,084	246,827	$682,412	25,676,463	$63,560,000	523,282	$950,000	168,365	$387,454
1892	3,706,748	2,516,168	230,480	572,256	25,697,223	64,038,800	481,468	896,600	165,445	394,251
1893	3,517,433	2,339,169	231,025	561,284	25,172,792	58,105,400	478,189	852,800	174,290	460,174
1894	3,672,101	2,455,449	283,439	684,908	26,964,000	58,537,920	453,000	987,540	131,717	356,606
1895	.....	.....	.....	.....	27,801,276	.....	434,763	.....	.....	.....

Year.	Lead-Silver Ore.		Salt.		Sulphur Ore. (*c*)		Zinc Ore.	
1891	25,897	$946,324	810,675	$2,827,777	6,749	$20,072	56,338	$1,285,504
1892	21,656	673,164	973,752	3,353,390	7,231	20,972	69,236	1,134,400
1893	24,599	530,742	1,103,287	3,317,471	3,733	13,022	77,466	1,061,166
1894	29,055	473,920	890,607	2,787,624	851	2,747	80,065	825,867
1895	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) The French tables are from *Statistique de l'Industrie Minérale*, except the imports and exports for 1896, which are from *l'Economiste*, and the imports and exports of Tunis, which are from *British Consular Reports*.
(*b*) Includes pure bitumen, bituminous schist, bituminous sand, and asphalt limestone; see details in special tables below. (*c*) Sulphur and limestone impregnated with sulphur.

PRODUCTION OF MINERAL OIL AND ASPHALT IN FRANCE. (IN METRIC TONS; 5f. = $1.)

Year.	Mineral Oil.		Asphalt.		Year.	Mineral Oil.		Asphalt.	
1890	7,906	$182,513	11,199	$97,527	1893	8,710	$189,580	13,608	$103,165
1891	9,926	239,460	15,070	132,407	1894	8,762	192,600	14,000	105,700
1892	8,008	191,160	14,690	138,702	1895	.....	.....	.....	.....

METALLURGICAL PRODUCTION OF FRANCE. (IN METRIC TONS; 5f. = $1.)

Year.	Aluminum.		Antimony.		Copper.		Iron, Pig.		Iron, Wrought.		Steel.	
1891....	36.0	$85,520	880	$177,153	2,125	$635,471	1,897,387	$24,827,169	833,409	$29,546,616	638,530	$33,497,528
1892....	75.0	125,600	754	148,362	2,163	567,000	2,057,300	24,973,800	828,519	29,533,200	682,500	36,823,600
1893....	137.0	194,900	868	157,300	6,587	1,507,700	2,003,100	23,347,000	808,171	23,347,400	789,852	33,367,000
1894....	270.0	274,400	1,012	136,000	6,415	1,353,600	2,069,700	23,600,000	786,000	25,800,000	790,773	33,895,800
1895....	.....	.....	.....	.....	8,200	.....	2,005,889	.....	743,671	.....	716,931	.....

Year.	Lead. (b)		Litharge.		Nickel.		Gold—Kgms.		Silver—Kgms.		Zinc.	
1891.....	6,680	$417,116	63	$3,906	330	$336,400	220	$140,800	71,303	$2,281,696	20,596	$2,070,333
1892.....	8,776	510,478	49	2,619	1,244	1,236,400	210	144,075	103,207	2,994,163	20,609	1,993,956
1893.....	8,119	404,187	33	1,617	2,045	1,237,560	300	192,000	98,077	2,550,020	22,419	1,843,649
1894..... ..	8,758	365,800	.....	.....	1,545	1,237,600	376	240,600	96,955	2,133,000	23,387	1,815,400
1895.....	7,600	.....	.....	.....	.....	.....	.....	.....	.....	.....	24,200	.....

(b) Lead produced from native ore only, and does not include the metal produced from foreign mineral imported into France to be desilverized.

MINERAL AND METALLURGICAL IMPORTS OF FRANCE. (IN METRIC TONS; 5f. = $1.)

Year	Bituminous Substances. (b)		Coal. (c)		Copper.		Gold.		Silver.	Iron & Steel.		Castings.	
1891	273,000	$8,743,600	10,542,000	$37,954,200	30,074	$8,694,000	$72,500,000	$35,280,000	45,856	$1,258,600	56,756	$681,000
1892	312,000	10,509,600	10,357,000	36,921,000	24,384	6,796,600	77,520,000	24,000,000	57,790	1,902,200	84,520	1,835,600
1893	860,000	11,689,600	11,401,000	36,464,800	29,880	7,276,200	61,094,356	31,743,279	82,529	2,000,600	93,616	1,039,400
1894	59,980	1,439,600	11,644,000	34,335,400	30,300	6,863,800	92,308,712	17,719,346	50,034	1,867,800	47,118	714,600
1895	43,975	879,600	10,261,069	42,953,250	38,196	8,859,400	50,775,039	28,230,595	66,240	1,619,400	36,247	581,200
1896	30,954	619,080	10,179,320	34,587,600	47,125	10,888,800	59,767,860	31,150,154	48,423	1,661,400	18,323	369,000

Year.	Lead.		Nickel.		Nitrate of Soda.	Stone.	Sulphur.		Tin.		Zinc.	
1891.....	74,502	$4,506,600	801	$1,031,400	$7,519,400	$3,972,800	88,678	$2,416,000	6,204	$2,847,000	30,449	$3,447,000
1892.....	72,721	5,166,200	2,703	1,456,400	9,013,600	4,456,000	98,721	2,329,200	7,085	3,382,600	29,884	3,305,400
1893.....	77,679	5,262,800	1,866	1,046,200	6,034,400	4,925,800	98,750	2,329,200	8,890	3,683,000	35,200	3,781,200
1894.....	84,167	5,236,800	267	269,400	7,609,400	4,989,000	112,536	1,689,200	8,137	2,969,400	33,652	2,787,400
1895.....	66,241	4,626,800	252	300,600	8,624,200	3,144,200	110,989	1,598,400	7,691	2,645,800	25,652	1,988,000
1896.....	79,749	5,432,000	421	374,600	9,455,200	2,983,000	111,515	1,607,800	8,434	2,921,800	33,459	2,576,800

(b) Includes bitumen, bituminous schist and sand, and asphaltic limestone. (c) Including coke.

MINERAL IMPORTS OF FRANCE. (IN METRIC TONS.)

Year.	Lead Ore.	Copper Ore	Antimony Ore.	Gold Ore. Kgms.	Iron Ore	Iron Pyrites	Manganese Ore.	Nickel Ore.	Silver Ore Kgms.	Tin Ore.	Zinc Ore.
1891..	3,893	10,024	208	7,176	1,438,000	45,457	44,919	6,165	1,245,000	628	36,437
1892..	7,281	16,850	123	13,277	1,684,000	47,502	43,888	11,927	3,539	523	41,931
1893..	5,261	20,527	175	2,606	1,630,000	56,505	35,531	18,236	4,092,000	78	34,221
1894..	5,825	10,191	153	b24,824	1,638,458	56,672	43,345	14,941	3,159,514	123	34,955
1895..	5,032	10,450	.....	b8,811	1,651,369	67,930	41,403	10,303	c524,394	104	41,622
1896..	5,570	8,584	.....	b101,499	1,862,065	45,788	61,585	15,756	c1,818,650	7	50,899

(b) Gold and platinum in ore, sheets, leaves, or threads. (c) Silver in ore, sheets, leaves, or threads.

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (IN METRIC TONS.)

Year.	Aluminum.	Antimony.		Copper.		Gold.		Iron.				Iron Pyrites	Lead.	
		Ore.	Metal	Ore.	Metal	Ore.	Metal	Ore.	Pig.	Casting	Steel.		Ore.	Metal
1891....	.....	1,153	78.0	11,802	6,528	.....	6,045	299,000	196,271	145,412	54,019	12,120	13.280	10,657
1892....	42	1,333	90.0	12,387	8,526	.....	4,222	305,000	161,173	159,453	30,323	22,455	10,823	10,057
1893 ...	43	492	165.0	10,198	7,601	5,211	4,222	302,000	130,028	143,652	28,426	38,060	14,032	9,243
1894 .	31	1,173	20.0	4,536	6,598	966	4,448	247,519	140,718	156,638	37,361	48,627	11,868	8,133
1895....	110	832	68.2	1,772	8,829	.....	1,353	236,923	15,054	10,755	29,074	37,968	8,670	8,037
1896....	793	736	73.8	1,261	10,494	.....	2,201	233,075	25,900	13,944	63,367	14,232	8,607	10,129

Year.	Manganese Ore.	Nickel.		Quicksilver.	Silver—Kgms.		Tin.		Zinc.	
		Ore.	Metal.		Ore.	Metal.	Ore.	Metal.	Ore.	Metal.
1891.....	1,494	26	210	6	103,222	17,800	88	1,014	31,662	5,177
1892.....	8,541	455	319	5	195	18,950	222	918	32,023	7,034
1893.....	12,123	701	191	5	61,219	21,642	.....	973	47,426	8,819
1894.....	8,039	241	354	25	51,272	11,472	4	816	58,281	6,595
1895.....	16,193	.....	408	.....	.....	13,567	.....	650	61,291	5,849
1896.....	10,912	.....	490	.....	.....	9,849	.....	745	62,415	10,485

MINERAL PRODUCTION OF ALGERIA. (IN METRIC TONS; 5f. = $1.)

Year.	Antimony Ore.		Copper Ore.		Iron Ore.		Iron, Cast.		Lead-Silver Ore.		Quicksilver Ore.		Zinc Ore.		Copper Matte.		Salt.	
1892	48	$2,470	21,907	$281,022	452,603	$807,140	445	26,074	349	$7,463	178	$1,323	21,907	$281,022	448	53,505	24,784	$104,086
1893	79	4,740	24,390	197,866	393,921	702,388	517	34,916	212	4,181	757	9,806	24,390	197,866	620	64,200	19,008	93,416
1894	175	5,200	.....	.....	343,830	527,783	538	32,499	276	4,045	866	11,602	29,703	163,392	700	64,600	17,830	74,763

Algeria exported in 1891, 13,928 tons of lead and copper ore, and 9748 tons of zinc ore; in 1892, 12,959 of lead and copper ore, and 21,436 of zinc ore; in 1893, 6762 of lead and copper ore, and 27,844 of zinc ore; in 1894, 2673 of lead and copper ore, 28,788 of zinc ore, and 71 tons of antimony ore.

VALUE OF IMPORTS AND EXPORTS OF TUNIS. (£1 = $5.)

Year.	IMPORTS.							EXPORTS.
	Building Materials.	Coal.	Iron, Steel, Including Rails.	Marble.	Machinery.	Metal Goods.	Mineral Oils.	Zinc.
1891.....	$65,380	$121,550	.....	.....	$109,660	.....	$43,110	$935
1892.....	159,850	121,180	$88,075	$23,385	88,675	$704,500	67,755	49,030
1893.....	.....	92,810	219,485	.....	116,350	485,225	61,935	121,390
1894.....	.....	122,510	176,570	.....	157,600	399,945	75,731	226,000

MINERAL PRODUCTION AND EXPORTS OF THE FRENCH COLONIES. (IN METRIC TONS.)

Tunis produced during 1894, 31,000 tons of calamine and 7000 tons of salt.

New Caledonia.—Production of nickel ore amounted in 1892 to 83,000 tons; 1893, 59,130; 1894, 61,243. The exports in 1893, 45,613; 1894, 40,089 tons. Average value, $14 per ton. Cobalt production 1893, 2220 tons, exports 520 tons; 1894, 4112 tons, exports 4156 tons. Also exported in 1894 6 tons matte valued at $700 per ton. Value of cobalt averaged $14 per ton. No chrome ore was produced in 1893, but exports amounted to 242 tons. Production in 1894 reached 2927 tons, with exports during that year aggregating 1042 tons, average value $12 per ton.

Indo-China and Tonkin.—The production of coal in 1893 was 250,000 tons; for 1894, the exports and consumption only are known; exports in 1894 amounted to 100,000 tons, valued at $198,400, and the consumption 12,753 tons.

Annam.—Exported during 1894, 1651 tons of coal, valued at $4800.

Cochin-China.—6200 kgms. jet valued at $2400, were produced in 1894.

Guiana.—Besides producing gold, phosphate has been mined since 1888, and the following amounts produced: 1888, 2207 tons; 1889, 2559; 1890, 4187; 1891, none; 1892, 3938; 1893, 2561; 1894, 6605. All this phosphate is shipped to Nantes, France, at a freight rate of $5.40 per ton.

GERMANY.

THE mining industry of the German Empire was on the whole very prosperous in 1896, and an increase in the production of most of the important metals is to be noted. The details will be found under the respective captions "Lead," "Copper," "Zinc," etc.

The iron industry was especially active in 1896, the production increasing more than 500,000 metric tons, the more part of the increase being in pig for Thomas-Gilchrist steel. The imports of iron ore also showed a large increase. The exports were about the same as usual, consisting chiefly of Alsace-Lorraine product, which is shipped to France and Belgium.

The iron industry in Upper Silesia has been undergoing a great development. At the Königshütte a new rolling-mill has been completed, and a battery of coke-ovens and a waste-product recovery plant are in course of erection. The Laurahütte is extending its steel plant, and the erection of another blast furnace is contemplated. At the Fälvahütte a large amount of new plant is being added to the rolling-mill department, while the Huldschinsky Hüttenwerke of Gleiwitz is erecting a new Bessemer-steel plant and rolling-mill. The Friedenshütte is at present putting down a battery of coke ovens with waste-product recovery plant, while at the Julienhütte of the Upper Silesian Iron Industry Company a similar plant is being erected, and also a new blast-furnace.

The production of coal increased importantly, 265 collieries being in operation; of these 68 were in the Breslau district (Silesia) and 162 in the Dortmund (Westphalia). The Silesian collieries, though the fewer in number, are generally larger producing a little more than half as much as the Westphalian.

MINERAL PRODUCTION OF GERMANY AND LUXEMBURG. (*a*) (IN METRIC TONS AND DOLLARS.)
($1 = 4 MARKS.)

Year.	Antimony Ore.		Arsenic Ore.		Asphaltum.		Boracite.		Coal.		Cobalt, Nickel, & Bismuth Ore.		Copper Ore.	
1891...	1	$41	3,124	$32,563	49,150	$93,928	177	$12,660	73,715,653	$147,379,551	1,074	$159,328	587,626	$5,216,192
1892...	....	.....	2,146	19,491	53,279	104,713	179	13,614	71,372,193	131,744,794	3,185	198,208	567,738	5,128,386
1893...	15	75	2,758	25,391	47,238	89,245	162	11,662	73,852,330	124,598,755	4,490	188,713	584,950	4,530,794
1894...	....	.....	2,906	30,006	55,981	112,762	176	11,065	76,741,127	127,275,053	4,524	194,645	588,195	4,060,039
1895...	24	218	3,546	35,311	59,563	113,606	150	8,781	79,169,276	134,723,786	5,180	163,450	633,355	3,844,961
1896...	....	.....	.....	.....	.....	.....	...	.....	85,639,861	148,263,717	4,087	129,688	717,306	4,239,362

Year.	Epsomite.		Graphite.		Iron Ore.		Iron Pyrites.		Other Vitriol and Alum Ores.		Kainite.	
1891...	7,454	$16,314	3,824	$73,540	10,657,521	$9,852,076	128,288	$239,468	2,406	$1,517	472,256	$1,701,640
1892...	10,207	22,356	4,036	63,240	11,539,133	10,319,938	115,243	215,887	2,973	2,016	548,445	1,955,717
1893...	8,818	17,062	3,140	52,010	11,457,533	9,950,266	121,329	218,687	791	1,250	646,755	2,352,284
1894...	8,252	17,999	3,133	45,732	12,392,065	10,544,385	134,787	244,864	465	977	726,524	2,578,208
1895...	7,328	14,856	3,751	50,612	12,349,600	10,268,935	127,036	243,878	351	525	680,174	2,402,339
1896...	.....	.....	.....	.....	14,162,315	12,849,650	124,950	231,579	.....	.....	856,290	2,989,736

Year.	Other Potash Salts.		Lead Ore.		Lignite.		Manganese Ore.		Petroleum.	
1891.....	898,993	$2,771,494	159,215	$4,163,988	20,536,625	$13,541,457	40,335	$202,445	15,315	$298,659
1892.....	802,630	2,532,169	163,372	3,671,881	21,171,857	14,626,474	32,861	126,924	14,527	219,941
1893.....	879,477	2,815,828	168,413	3,536,042	21,573,823	13,755,744	40,798	122,988	13,974	195,733
1894.....	917,049	2,992,149	162,676	3,025,884	22,064,575	13,287,909	43,702	116,412	17,232	243,112
1895.....	841,748	2,776,504	161,614	3,234,902	24,788,363	14,502,821	41,327	124,534	17,051	240,614
1896.....	924,140	2,997,667	154,660	3,093,214	26,797,880	15,233,273	.....	.....	20,395	297,128

Year.	Rock Salt.		Silver and Gold Ore.		Tin Ore.		Uranium and Wolfram Ore.		Zinc Ore.	
1891.....	666,793	$744,754	22,569	$1,151,723	75	$22,839	47	$10,564	793,544	$6,238,384
1892.....	662,577	708,046	17,536	910,501	63	23,001	48	11,057	800,237	5,305,324
1893.....	669,043	736,029	18,778	774,617	53	21,913	44	10,778	787,911	3,572,711
1894.....	734,937	784,923	19,080	629,672	211	16,309	40	6,194	728,616	2,569,514
1895.....	686,940	777,095	10,845	427,047	154	15,197	29	4,449	706,423	2,644,314
1896.....	755,833	755,888	18,487	596,604	.....	.....	.....	.....	729,872	4,257,473

METALLURGICAL PRODUCTION OF GERMANY. (*a*)

Year.	Antimony and Alloys.		Arsenical Products.		Cadmium—Kilos.		Copper, Matte and Black.		Copper, Pig.		Gold—Kilos.	
1891.....	198	$43,175	1,988	$138,626	2,797	$2,468	596	$46,139	24,092	$6,952,884	2,432	$1,685,500
1892.....	249	44,931	1,667	109,127	3,200	2,850	625	24,630	24,781	6,189,532	2,553	1,776,250
1893.....	407	64,485	1,794	119,588	5,000	5,461	842	63,735	24,011	5,860,517	2,551	1,774,000
1894.....	424	65,244	2,389	166,798	6,520	6,886	676	26,928	25,722	5,467,554	3,204	2,232,750
1895.....	989	107,960	3,014	207,416	7,047	9,809	789	49,890	25,777	5,818,903	3,547	2,494,379
1896.....	1,326	156,111	2,637	221,397	.....	.....	.....	.....	29,319	7,293,387	2,487	1,728,938

Year.	Iron.		Lead, Pig.		Litharge.		Mineral Paints.		Nickel Products. (*b*)		Silver—Kilos.		Sulphur.	
1891	4,631,218	$58,107,003	95,615	$5,816,587	3,124	$197,317	2,332	$61,820	1,062	$1,606,200	444,852	$14,749,420	2,020	$51,050
1892	4,937,461	57,324,072	97,742	5,136,861	3,468	201,765	2,629	68,313	1,238	1,713,556	489,350	14,307,172	5,155	54,460
1893	4,986,030	54,986,500	94,659	4,519,250	3,551	183,181	2,993	84,008	1,402	1,703,286	449,383	11,766,155	2,151	51,308
1894	5,310,885	57,094,968	100,753	4,762,451	3,646	192,043	2,834	72,800	997	1,154,757	444,213	9,653,769	2,168	50,209
1895	5,464,501	59,237,997	111,058	5,569,475	3,433	190,806	2,655	72,443	1,143	1,111,074	391,978	8,600,828	2,061	40,024
1896	6,295,272	73,930,749	113,792	6,258,097	3,930	235,698	.....	.....	1,391	1,365,330	428,429	9,718,000	....	.....

Year.	Sulphuric Acid.		Sulphates.								Tin.		Zinc.	
			Copper.		Iron.		Zinc & Nickel.		Mixed.					
1891	467,633	$4,018,746	3,502	$288,938	.....	.....	4,117	$103,453	298	$10,038	287	$131,131	139,353	$15,639,337
1892	488,047	3,716,014	4,024	281,234	9,393	$53,854	4,479	92,372	217	7,684	684	309,973	139,938	13,765,510
1893	517,790	3,887,500	4,773	353,351	8,457	41,763	4,648	84,305	233	8,535	951	348,135	142,956	11,821,500
1894	503,621	3,592,416	4,809	363,034	8,899	39,488	4,357	88,214	202	7,436	896	271,804	143,577	10,453,272
1895	534,740	3,623,231	4,638	341,600	9,820	40,579	4,017	55,950	170	6,492	884	266,292	150,286	10,409,160
1896	577,942	3,536,455	6,046	466,478	.....	.....	.....	.....	....	.....	459	135,734	153,100	11,777,005

(*a*) From *Vierteljahrs- und Monatshefte zur Statistik des Deutschen Reichs.*
(*b*) The nickel products include bismuth, cobalt, and uranium salts.

EXTRACTION OF SALTS FROM AQUEOUS SOLUTION. (*a*)

Year.	Alum.		Aluminum Sulphate.		Common Salt.		Glauber Salt.		Magnesium Chloride.	
1891....	5,619	$137,590	28,710	$541,805	503,386	$3,355,586	79,983	$514,301	15,619	$39,528
1892....	4,270	117,465	29,588	570,735	504,687	3,460,869	74,184	504,277	14,386	50,828
1893....	4,102	115,075	27,082	505,165	504,522	3,494,221	77,145	498,539	12,764	44,444
1894....	3,914	107,092	26,804	480,174	522,590	3,574,655	71,929	423,218	17,422	50,913
1895....	3,358	87,372	30,836	539,088	525,396	3,563,226	71,410	406,872	17,039	52,850

Year.	Magnesium Sulphate.		Potassium Chloride.		Potassium Sulphate.		Double Sulphate of Potassium Magnesium.	
1891.....	23,126	$71,126	129,512	$4,282,290	37,674	$1,514,382	10,508	$199,155
1892.....	23,879	84,008	123,962	4,106,579	26,267	1,072,096	11,593	228,172
1893.....	27,548	79,240	137,216	4,326,285	27,308	1,119,835	14,199	280,792
1894.....	28,628	88,814	149,775	4,722,049	23,281	958,736	14,156	274,694
1895.....	26,028	107,183	154,427	4,921,179	19,452	805,098	9,877	193,939

(*a*) From *Vierteljahrs- und Monatshefte zur Statistik des Deutschen Reichs.*

MINERAL IMPORTS OF GERMANY. (*a*) (IN METRIC TONS; unit of value, $1000; $1 = 4 mks.)

Year.	Alabaster and Marble.		Alum.		Ammonia, Sulphate of.		Aluminum, Copper, and Nickel Ware.		Coal.		Coke.		Lignite.		Cobalt and Nickel Metal, Crude.	
1892	21,298	$959	....	....	34,207	$1,881	1,474	$1,125	4,436,983	$14,870	465,726	$2,147	6,701,309	$6,366	1,356	$1,611
1893	21,520	650	....	..	42,596	2,550	1,467	1,050	4,664,048	14,925	439,182	1,725	6,705,672	6,375	1,006	950
1894	24,784	750	....	....	36,635	2,375	1,235	725	4,805,971	15,075	404,179	1,425	6,868,162	6,175	1,431	350
1895	23,785	725	251	...	29,203	1,600	1,359	850	5,117,356	15,825	461,779	1,675	7,181,050	9,875	621	425
1896	.....	.....	276	....	32,061	.....	.....	.....	5,476,753	.....	393,881	.....	7,637,503	.....	....	.....

Year.	Copper Bars, Sheets, and Wire.		Copper, Pig.		Gold and Silver, Articles of.		Gold, Coined.		Gold, Crude and Bars.		Gold and Silver, Platinum Ores.		Iodine.		Iron Ore.	
1892..	912	$300	32,498	$7,962	30	$1,242	45	$28,045	24	$16,572	19,308	$5,551	169	$1,119	1,655,843	$5,589
1893..	448	125	38,455	8,850	32	1,300	26	16,050	27	18,975	20,516	5,125	324	2,150	1,573,202	5,250
1894..	495	125	37,032	7,900	37	1,350	80	50,350	37	25,775	13,935	3,000	431	2,850	2,093,007	6,850
1895..	426	125	44,365	10,050	35	975	20	12,470	17	11,875	6,134	1,350	113	750	2,017,136	6,825
1896..	401	.. ..	56,115	.....	....	.....	...	.....	45	.....	6,024	.....	94	.....	2,586,706	.....

Year.	Iron, Pig.		Iron Wire.		Iron, Wrought.		Iron, Various Manufactures of.		Kaolin, Feldspar, Fireclay.		Lead, Pig.		Lead and Copper Ores.		Lead, Tin, and Zinc—Fine Manufact's of.	
1892..	209,306	$2,519	4,675	$375	21,085	$831	31,885	$3,225	151,398	$1,514	17,501	$941	43,893	$3,292	3,172	$410
1893..	218,998	2,475	4,946	400	17,054	625	32,494	3,000	141,096	1,325	23,857	1,200	43,490	2,925	2,855	309
1894..	203,948	2,375	4,888	400	19,966	700	32,220	3,625	153,208	1,350	24,280	1,200	51,304	2,125	2,914	275
1895..	188,217	2,250	5,583	450	19,777	700	32,817	3,375	153,911	1,350	28,449	1,500	65,270	2,825	3,173	287
1896..	322,502	.....	6,398	.. ...	23,770	.....	....	.....	192,807	.....	33,016	.....	88,341	.....	.....	.....

Year.	Lime, Chloride of.		Limestone and Mortar.		Lubricating Oil, Mineral.		Machinery.		Petroleum.		Phosphate of Lime.		Superphosphate.	
1892..	.....	.....	174,483	$873	63,150	$2,684	28,546	$4,425	743,433	$15,185	169,798	$2,972	86,851	$1,954
1893..	.....	.....	200,165	1,000	70,576	4,400	35,620	5,275	765,100	11,825	226,923	2,825	110,876	1,800
1894..	.....	.....	214,331	1,075	65,703	2,800	38,115	5,500	785,100	11,375	286,812	3,575	124,373	2,025
1895..	7,671	.....	221,685	1,100	75,041	3,275	41,374	6,350	811,058	15,400	278,046	3,475	96,099	1,450
1896..	1,230	. ...	.....	.....	81,250	.....	.....	. ....	852,234	.....	.....	.....	....	.....

Year.	Precious Stones.		Pyrites. (b)		Salt.		Silver. Coined.		Silver. Crude and Bars.		Silver and Gold, Refuse.		Slags.		Slate.		Soda, Crude and Crystals.	
1891..	139	$2,730	238,644	$1,193	25,747	$218	28	$845	69	$2,297	34	$4,162	361,457	$949	66,587	$1,147	90	$1
1892..	95	1,762	218,272	982	24,534	93	30	783	64	1,871	35	4,230	410,631	1,283	65,527	1,095	293	5
1893..	77	1,325	274,766	1,305	23,645	325	24	575	78	2,025	41	4,905	477,183	1,670	61,543	1,075	242	4
1894..	35	950	315,115	1,550	23,440	105	20	300	190	4,075	66	5,800	632,878	2,200	61,401	1,050	328	6
1895..	60	1,125	293,446	1,450	24,286	103	18	350	98	2,150	45	4,125	537,542	1,825	52,418	950	....	.....
1896..	....	.....	343,852	.....	22,908	.....	....	....	....	.....	....	.....	.....	.....	.....	.....	....	.....

Year.	Soda and Bicarbonate of Soda, Calcined.		Soda, Caustic		Soda, Nitrate of.		Stone, Rough or Hewn.		Tin. Pig.		Tin. Plate.		Zinc. Ore.		Zinc. Pig.	
1891..	569	$17	350	$21	395,653	$17,309	691,720	$6,179	9,081	$4,200	1,199	$102	37,762	$944	7,969	$847
1892..	467	14	148	5	379,899	16,146	685,591	5,142	8,765	4,163	1,234	104	41,558	883	13,021	1,399
1893..	763	23	419	13	384,710	16,350	697,448	4,800	10,538	4,478	1,227	98	23,883	418	18,211	1,189
1894..	758	22	753	24	404,561	17,200	859,459	5,375	10,775	3,750	.....	.....	14,712	175	17,988	1,375
1895..	682	19	1,008	29	459,514	17,800	760,433	3,800	10,581	3,395	.....	.....	25,818	300	17,542	1,250
1896..	.....	.....	.....	....	.....	.....	.....	.....	13,798	.....	.....	... ..	21,493	.....	16,343	.....

(a) From *Statistisches Jahrbuch für das Deutsche Reich*; 1896 from *Chemiker Zeitung*.
(b) Including alum ore.

MINERAL EXPORTS OF GERMANY. (a) (IN METRIC TONS; unit of value, $1000; $1 = 4 mks.)

Year.	Aluminum and Nickel Ware.		Cement.		Coal.		Coke.		Copper, Bars and Sheet.		Copper, Manufactured.		Copper, Pig.	
1891..	1,417	$2,125	388,457	$3,816	9,536,374	$29,961	1,354,298	$7,138	4,145	$1,430	9,675	$5,992	6,244	$1,779
1892..	1,317	1,647	432,153	2,776	8,971,055	24,974	1,717,893	7,295	4,507	1,352	10,076	5,832	6,598	1,666
1893..	1,463	1,475	423,892	2,450	9,677,305	26,150	1,902,424	7,325	4,890	1,350	12,646	7,450	7,497	1,775
1894..	1,374	1,350	406,369	2,575	9,739,035	25,300	2,261,924	8,875	5,008	1,300	13,420	7,625	6,609	1,450
1895..	1,669	1,675	471,124	3,000	10,360,838	26,750	2,293,328	8,900	4,700	1,300	15,980	9,475	6,329	1,425
1896..	.....	.....	478,340	.....	11,598,757	.....	2,216,395	.....	5,429	.....	.....	.....	5,964	.....

Year.	Gold, Bars and Crude.		Gold, Coined		Iron Ore.		Iron, Pig.		Iron, Wrought.		Iron, Various Kinds of. (b)		Iron and Steel Rails.	
1891..	7	$4,781	41	$25,608	1,984,428	$1,501	111,154	$1,808	193,253	$5,725	176,185	$4,230	142,846	$3,571
1892..	10.6	7.421	48	30,036	2,276,155	1,711	113,391	1,568	199,064	5,340	185,616	4,046	113,712	2,559
1893..	5.3	3,650	35	21,725	2,353,398	1,750	108,675	1,475	231,297	5,900	225,902	4,600	87,360	1,850
1894..	3.5	2,450	17	10,875	2,558,729	1,925	154,647	2,000	300,559	7,375	250,173	5,100	119,410	2,475
1895..	7.2	5,050	25	15,525	2,480,135	1,850	135,289	1,775	277,991	7,000	319,485	6,425	116,627	2,505
1896..	.....	.....	....	.....	2,642,384	.....	140,449	... ..	259,461	.....	.....	.....	.....	.....

Year.	Wire Nails.		Iron, Various Manufactures of.		Kaolin, Feldspar, Fireclay.		Lead.		Leads, Crayons, and Pastels.		Lead White; Zinc White; Zinc Gray.		Lead, Tin, and Zinc, Manuf's of.		Machinery.	
1891..	49,709	$2,051	472,502	$45,025	75,396	$754	24,971	$1,514	943	$743	22,128	$2,066	5,738	$2,261	91,110	$17,139
1892.	50,323	1,887	449,806	36,725	82,983	829	25,647	1,363	948	711	24,306	2,025	4,271	4,369	91,631	14,418
1893..	54,849	2,050	483,484	39,650	79,570	800	23,945	1,175	1,090	817	25,755	1,600	11,825	4,354	96,105	16,076
1894	56,425	2,100	533,902	39,950	103,324	1,025	24,355	1,175	1,048	775	28,427	2,025	.....	22,888	9,650	.....
1895	63,662	2,300	589,069	45,700	111,277	1,125	27,855	1,475	1,191	825	27,799	2,150	.. ...	.....	... ..	.....
1896	.....	.....	.....	.....	125,674	.....	24,828	.....	.....	....	33,323	.....	.....	.....	....	.....

Year.	Superphosphate.		Potash.		Potassium Salts.		Precious Metal Ware.		Salt.		Saltpeter. (c)		Silver, Bars and Crude.	
1891..	43,318	$1,299	11,094	$1,137	76,987	$2,700	89	$7,825	254,370	$985	9,663	$1,015	311	$10,100
1892..	56,075	1,402	12,233	1,254	63,242	2,250	85	6,375	197,371	765	8,874	932	352	10,345
1893..	62,800	1,175	10,865	1,125	85,858	3,050	87	5,975	196,095	825	9,652	925	450	11,800
1894..	60,569	1,125	12,105	1,175	93,913	4,280	94	6,600	237,058	650	13,213	1,325	406	8,725
1895..	55,742	975	13,443	1,100	.....	.....	.....	.....	196,344	600	12,928	1,250	259	5,750
1896..	57,391	....	12,673	.....	.....	... ..	.....	.....	214,060	.....	11,323	.....	.....	.....

Year.	Silver, Coined.		Stone, Rough and Hewn.		Ultramarine.		Zinc. Sheets.		Zinc. Spelter.	
1891	6	$175	501,473	$2,500	4,404	$639	15,370	$1,979	57,852	$6,581
1892	38	1,000	537,220	2,675	4,353	631	16,304	1,875	53,287	5,525
1893	29	700	483,103	2,425	4,142	601	17,459	1,669	62,592	5,325
1894	8	150	578,451	2,900	.....	.....	16,038	1,350	61,799	4,525
1895	4	75	505,163	2,525	.....	.....	15,921	1,275	56,933	4,025
1896	.....	.....	.....	.....	4,416	.....	16,242	.....	58,082	.....

(*a*) From *Statistisches Jahrbuch für das Deutsche Reich.* The classification in the issues of 1892 and 1893 differs. Also from *Vierteljahrshefte zur Statistik des Deutschen Reichs*, Part II., 1893, and Part II., 1894. (*b*) Including scrap iron, structural iron, blooms, ingots, etc. (*c*) Not including Chile saltpeter.

MINERAL PRODUCTION OF BAVARIA. (*a*) (IN METRIC TONS; $1 = 4 marks.)

Year.	Antimony Ore.		Barytes.		Building Stones.		Cement.		Coal.		Lignite.	
1891..	0.59	$40	5,124	$9,962	612,657	$998,182	71,648	$68,104	756,148	$2,012,768	10,044	$11,762
1892..	.....	.....	4,765	9,543	582,784	883,062	87,571	78,798	713,052	1,815,415	13,367	15,662
1893..	.....	.....	.....	.....	568,410	836,689	79,779	80,386	802,537	2,000,357	17,167	14,518
1894..	.....	.....	4,550	5,425	547,115	869,769	81,723	83,426	806,389	1,972,941	20,687	17,079
1895..	.....	.....	3,587	4,388	613,969	854,209	85,089	85,864	903,340	2,147,045	26,531	23,305

Year.	Copper Ore.		Emery.		Feldspar.		Fireclay.		Flagstones.		Fluorspar.		Graphite.	
1891..	.....	.....	184	$1,511	1,445	$2,890	103,287	$206,674	20,772	$69,005	7,847	$8,420	3,824	$73,540
1892..	1,831	$7,706	170	1,710	1,200	2,400	108,312	237,056	19,538	79,930	4,594	5,907	4,036	63,240
1893..	650	2,762	172	1,498	1,200	2,400	110,970	217,859	26,873	110,493	3,988	4,473	3,140	52,010
1894..	.....	.....	148	1,584	1,720	3,540	121,950	219,154	18,083	77,932	3,616	6,015	3,133	45,732
1895..	.....	.....	229	2,355	1,115	3,075	106,925	180,188	18,909	84,685	3,940	5,986	3,751	50,612

Year.	Gypsum.		Iron Ore.		Iron Pyrites.		Kaolin.		Limestone.		Lithographic Stone.		Manganese Ore.	
1891..	30,197	$10,407	149,653	$150,720	1,933	$4,833	18,320	$19,284	224,185	$61,431	8,775	$219,375	260	$650
1892.	24,517	12,764	146,392	149,004	1,945	4,862	18,085	17,332	233,417	64,716	8,650	216,262	137	275
1893..	25,540	14,275	149,270	139,610	2,107	5,269	17,835	22,117	283,488	76,376	8,885	234,640	180	445
1894..	25,267	13,219	138,976	140,944	1,928	4,748	15,944	22,122	229,784	60,443	9,286	185,730	80	270
1895..	21,773	11,597	145,191	148,282	1,954	4,616	13,250	19,042	271,195	69,596	8,788	175,770	150	63

Year.	Ocher and Mineral Paints.		Quartz Sand.		Salt, Rock.		Sandstone.		Slate.		Soapstone.		Whetstone.	
1891..	5,710	$14,750	32,974	$11,271	1,042	$6,638	221,575	$307,117	1,433	$12,127	1,145	$22,930	109	$2,075
1892..	8,877	20,773	32,381	10,963	955	6,115	246,193	319,562	1,463	13,005	1,271	20,343	147	2,802
1893..	10,317	31,168	15,325	5,531	1,214	7,654	238,405	314,290	1,485	12,886	1,911	29,320	57	1,300
1894..	9,195	23,971	29,775	9,632	630	4,122	221,642	291,924	1,145	10,562	1,900	22,913	66	1,456
1895..	8,579	25,945	33,457	10,513	1,321	8,443	210,450	281,893	1,278	10,766	2,421	37,979	215	9,000

Year.	Iron, Pig.		Iron, Cast, 1st Fusion.		Iron, Cast, 2d Fusion.		Iron, Bar.		Iron, Sheet.		Iron Wire.	
1891..	76,552	$883,823	213	$7,999	52,257	$2,523,427	64,742	$2,187,057	282	$12,519	1,644	$46,080
1892..	77,598	898,302	334	12,521	48,212	2,249,683	64,646	2,020,865	592	22,349	1,008	28,035
1893..	75,209	856,182	206	6,959	49,614	2,307,051	49,908	1,483,411	1,482	57,766	307	8,211
1894..	75,669	865,543	196	6,606	53,342	2,490,670	46,860	1,351,169	268	10,087	279	7,426
1895..	77,114	884,615	293	9,898	62,131	2,868,020	48,596	1,368,013	150	5,250	100	2,551

Year.	Steel.		Salt.		Sulphate of Soda.		Other Sulphates and Alum.		Sulphuric Acid.	
1891.....	67,088	$1,834,077	40,629	$439,614	466	$3,800	662	$29,198	7,520	$70,500
1892.....	70,790	1,887,673	41,352	444,192	516	4,617	612	29,597	6,566	62,192
1893.....	60,824	1,645,074	42,153	459,596	1,229	3,750	628	27,873	7,355	64,692
1894.....	86,594	2,022,324	42,183	484,146	568	3,250	668	34,526	6,979	59,693
1895.....	96,828	2,336,689	41,106	446,628	493	2,875	637	34,170	6,515	52,800

(*a*) From the *Uebersicht der Production des Bergwerks-Hütten und Salinen Betriebes im Bayerischen Staate.*

MINERAL PRODUCTION OF PRUSSIA. (*a*) (IN METRIC TONS; $1 = 4 marks.)

Year.	Alum Shale.		Antimony Ore.		Arsenic Ore.		Asphalt.		Boracite.		Coal.		Lignite.	
1891..	2,163	$963	....	.....	2,169	$25,864	11,217	$36,021	150	$11,073	67,528,015	$131,806,263	16,739,984	$10,892,089
1892..	2,460	1,087	....	.....	1,202	12,229	12,665	35,956	168	12,953	65,442,558	117,677,458	17,219,033	11,913,033
1893..	302	418	15	$75	1,634	16,384	11,290	34,730	139	13,037	67,657,844	110,084,144	17,566,137	11,184,614
1894..	126	189	...	..	2,222	22,220	14,108	44,537	164	10,490	70,643,979	113,518,107	17,791,062	10,512,840
1895..	125	188	24	218	3,046	30,460	14,391	44,659	140	8,251	72,621,509	119,888,603	20,114,877	11,527,852

Year.	Cobalt Ore.		Copper Ore.		Epsom Salt.		Iron Ore.		Iron Pyrites.		Kainite.	
1891..	576	$9,209	578,256	$5,149,521	6,421	$13,896	3,903,811	$6,207,238	119,100	$213,457	399,007	$1,421,727
1892..	534	14,550	557,172	5,073,704	8,518	18,423	4,081,306	6,388,537	104,346	184,851	448,095	1,562,839
1893..	203	8,491	573,721	4,471,014	7,721	14,656	4,007,898	6,036,567	110,072	192,937	531,560	1,904,822
1894..	203	5,741	579,132	4,012,585	7,734	16,833	4,012,446	6,141,224	123,149	215,597	529,169	1,814,585
1895..	120	6,298	643,240	3,799,452	6,789	13,701	3,726,725	5,700,009	115,051	201,660	513,097	1,804,950

Year.	Other Potash Salts.		Lead Ore.		Manganese Ore.		Nickel Ore.		Petroleum.		Quicksilver Ore		Rock Salt.	
1891..	617,638	$1,712,988	140,123	$4,000,998	36,860	$181,899	185	$1,452	2,498	$76,050	.....	.....	283,924	$321,893
1892..	501,748	1,439,801	141,660	3,462,955	31,388	106,087	529	4,500	1,585	41,925	.....	.....	245,550	253,628
1893..	596,062	1,769,353	148,441	3,364,366	39,132	97,830	652	8,090	1,365	36,592	1.2	$25	260,727	280,259
1894..	625,662	1,897,079	144,724	2,900,092	42,526	98,950	1,341	13,413	1,600	39,790	.....	.....	305,810	329,341
1895..	558,846	1,707,929	140,991	2,911,948	39,882	105,916	2,058	4,190	1,612	46,446	*nil*	.....	278,326	327,861

Year.	Silver and Gold Ore.		Zinc Ore.		Antimony and Alloys.		Arsenic Products.		Bismuth.		Cadmium—Kilos.		Cobalt Products.	
1891..	131	$20,610	792,351	$6,230,590	165	$23,750	812	$41,379	.. ..	....	2,797	$2,468	44	$198,740
1892..	4	11,498	797,698	5,294,549	210	23,106	592	22,739	.....	.....	3,200	2,850	54	239,909
1893..	12	18,665	787,048	3,507,104	362	39,898	709	31,939	0.648	$174	5,284	5,461	44	190,315
1894..	6	9,281	727,645	2,567,053	376	40,034	1,147	57,377	3.002	7,117	6,052	6,335	46	142,675
1895..	12	23,626	706,179	2,643,282	924	73,660	1,788	98,363	1.000	1,750	7,047	9,809	45	144,103

Year.	Copper.		Copper, Pig.		Gold—Kilos.		Iron, Pig.		Lead, Pig.		Litharge.	
1891..	547	$43,150	21,236	$6,102,910	100.31	$69,790	3,288,441	$43,832,358	87,372	$5,286,817	2,246	$142,510
1892..	625	24,630	21,559	5,383,722	115.82	81,067	3,439,081	42,515,551	87,983	4,607,369	2,634	152,143
1893..	831	65,666	20,707	5,045,292	738.75	513,707	3,539,701	41,118,869	85,866	4,166,992	2,548	139,304
1894..	670	26,223	21,966	4,670,025	687.88	479,215	3,744,116	43,048,291	92,379	4,357,035	2,647	141,079
1895..	722	48,548	22,069	4,980,829	653.00	455,159	3,778,774	43,961,347	99,586	4,983,733	2,458	137,403

Year.	Manganese and Alloys.		Nickel and Alloys.		Paint, Mineral.		Salt, Common.		Silver—Kilos.		Sulphur.		Sulphuric Acid.	
1891..	33	$19,425	594	$689,405	1,813	$35,863	265,549	$1,675,428	277,546	$9,254,830	1,721	$46,528	343,826	$2,938,186
1892..	39	21,825	747	747,312	2,082	40,320	264,896	1,695,904	301,374	8,786,415	1,850	49,828	360,156	2,722,789
1893..	44	24,590	896	900,339	2,415	55,171	266,478	1,707,265	276,645	7,276,755	1,871	77,659	387,306	2,977,760
1894..	48	25,210	522	480,675	2,250	39,325	280,511	1,777,592	279,781	6,102,001	1,888	42,912	420,965	3 098,586
1895..	66	34,300	698	616,143	2,059	38,863	278,962	1,757,837	245,527	5,376,046	1,819	34,085	416,239	3,894,871

Year.	Sulphates.										Tin.		Zinc.	
	Copper.		Iron.		Nickel.		Zinc.		Mixed.					
1891.	1,285	$109,032	7,682	$51,344	45	$14,000	2,405	$36,723	223	$6,723	238	$107,778	139,147	$15,616,909
1892..	1,338	95,056	7,778	41,692	44	14,000	2,748	40,886	167	5,506	644	290,720	139,725	13,743,287
1893..	1,872	135,231	7,134	30,765	44	11,237	2,704	37,111	187	6,298	909	329,790	142,773	11,806,066
1894..	1,858	137,956	7,600	27,788	46	13,795	2,746	38,914	148	4,813	842	252,600	143,354	10,435,118
1895..	1,922	138,905	8,627	50,185	91	24,325	2,641	39,266	112	3,598	849	254,592	150,123	10,397,000

(*a*) From *Zeitschrift für das Berg-, Hütten- und Salinenwesen.*

MINERAL PRODUCTION OF SAXONY. (*a*) (IN METRIC TONS; $1 = 4 marks.)

Year.	Bismuth, Cobalt, and Nickel Ores.		Coal.		Lignite.		Iron Ore.		Manganese Ore.		Pitch Blende.		Tin Ore.	
1891..	291	$148,379	4,366,819	$10,957,819	864,376	$657,733	14,159	$36,912	2,046	$4,717	....	.....	50	$22,758
1892..	2,113	183,261	4,212,875	9,939,670	927,860	674,682	12,895	23,945	1,639	3,699	....	.....	49	18,945
1893..	3,634	174,917	4,274,064	10,128,936	940,988	663,831	1,650	2,972	.....	.....	...	.....	49	18,908
1894..	2,980	175,513	4,123,227	9,435,935	918,589	611,836	1,043	1,833	....	.....	734	$2,653	211	23,697
1895..	3,001	153,056	4,435,328	10,224,100	1,018,486	670,274	20	107	8	105	26	501	154	14,942

MINERAL PRODUCTION OF SAXONY—*Continued.*

Year.	Wolfram Ore.		Mixed Ores. (*b*)		Other Minerals. (*c*)		Totals.
1890	37	$5,722	31,498	$1,163,737	2,250	$18,249	$12,329,147
1891	42	8,212	31,909	1,164,205	3,136	16,891	13,017,626
1892	37	8,108	30,231	1,025,078	3,212	15,256	11,892,644
1893	42	7,866	31,565	1,000,077	2,425	4,547	12,002,554
1894	39	5,747	33,411	713,975	607	7,393	11,008,582
1895	29	4,289	28,247	711,031	423	6,825	11,785,680

METAL CONTENTS OF THE MIXED ORES TREATED AT THE ROYAL SMELTING WORKS AT FREIBERG. (*a*) (IN METRIC TONS; $1 = 4 marks.)

Total of Mixed Ores.		Arsenic.	Copper.	Gold. Kilos.	Lead.	Nickel and Cobalt.	Silver. Kilos.	Sulphur	Zinc.
31,498	$1,163,737	528	16	0.184	3,583	2.97	34,151	4,935	151
31,528	1,159,029	462	18	0.286	4,253	0.97	34,499	4,619	318
30,015	1,023,215	415	13	0.0496	4,163	0.55	34,908	4,208	338
31,335	869,041	406	20	.....	4,362	0.65	33,253	4,841	188
31,099	709,397	424	20	.....	4,619	0.558	31,635	5,354	134
26,679	694,131	350	15	0.1314	4,740	0.0603	29,383	3,905	82

PRODUCTS SOLD BY THE ROYAL SMELTING WORKS AT FREIBERG AND THE BLUE-COLOR WORKS AT SCHNEEBERG. (*a*) (IN METRIC TONS.)

Year.	Arsenic Products. (*d*)	Bismuth.	Cobalt Products. (*e*)	Copper Sulphate.	Gold. Kilos.	Lead Products. (*f*)	Lead, Sheet.	Lead, Shot.	Lead, other Mf'es of. (*g*)	Nickel Speiss.	Silver. Kilos.	Sulphuric Acid. (*h*)	Other Chemicals. (*i*)	Zinc and Zinc Gray.
1891...	1,170	2.5	416	1,750	950	4,159	1,200	193	179	56	83,513	13,832	726	207
1892...	1,075	2.0	407	1,975	778	6,054	565	187	356	49	94,830	10,860	932	212
1893...	1,084	2.1	445	2,024	954	5,456	642	194	454	54	95,102	14,409	690	182
1894...	1,229	2.4	404	2,140	957	4,603	1,160	161	198	75	81,322	12,679	699	221
1895...	1,217	2.5	406	1,833	850	6,603	592	238	218	2	54,770	10,205	624	163

(*a*) From *Jahrbücher für das Berg- und Hüttenwesen im Königreiche Sachsen.* (*b*) See the appended table of Metal Contents of Mixed Ores Treated at the Royal Smelting Works at Freiberg. (*c*) Fluorspar, barytes, mineral paint, etc. (*d*) Including arsenious acid, red, yellow, and white glass, and metallic arsenic. (*e*) Products of the blue-color works at Schneeberg. (*f*) Including soft lead, antimonial lead, litharge, etc. (*g*) Lead pipe etc. (*h*) Including sulphuric acid of various sorts. (*i*) Including copperas, sodium-sulphate, etc.

GREECE.

By E. Grohmann.

The mining and smelting industries of Greece have in the past year received a noteworthy impetus by legislation. Since 1880 new concessions could be obtained only from Parliament, but the old state of things has been restored so that all that is now required is a royal order, which is issued either upon the proposal of some official in high position, or by a commission, meeting periodically.

Although the output of the mines and works did not increase materially in 1896, a more vigorous activity was noticeable, several new enterprises being established, which will naturally become factors during the current year (1897). There is, for instance, an iron mine in the island Kea, producing brown hematite similar to that of Seriphos; a manganese mine in the island Andros, from which ore with 40 to 45% manganese is obtained; it is believed that argentiferous galena in paying quantity has been found in the island Kimolos; and, finally, oxidized copper ore is beginning to be mined in the island Milo. Hitherto copper has not been found in paying quantity in the dominion of ancient Hellas. Unsuccessful attempts were made in Cyprus as well as in the island Thasos, Chalcidice in Macedonia, the Othrys Mountains of Greece proper, Peloponnesus, and the island Seriphos to reopen the ancient, celebrated copper mines, which, according to tradition and the evidence of the old works and the piles of slags, were once very important. The ancients exploited the deposits of metals valuable to them in a somewhat irregular manner, but with full knowledge of their geological conditions. Moreover, they knew how to extract them from their ores, as the enormous piles of slags they left are generally worthless, containing, for instance, only 0.50 to 0.75% Cu.

The fact that the ancients bestowed little attention upon the mining and smelting of lead ores may be principally because they had little use for lead, and furthermore, no occasion to work it for silver, since all the latter metal needed for coinage was supplied by their richer lead ores. Laurium owes its present great industry to this circumstance. The easy and profitable resmelting of the old slags was the primary reason for establishing the present works there, and also furnished the means for the reopening of the mines, which, beside the lead ores, soon began to furnish calamine and manganiferous iron ores, both disregarded by the ancients.

The exportation of lead bullion from Greece last year amounted to 14,700 tons (containing 33,850 kgms. silver), about one-half going to England and the remainder to Antwerp; besides this, 480 tons of desilverized lead were utilized in Greece for pipe, sheet, and shot. The old heaps of slag are now entirely worked up, though small quantities are still recovered by dredging from the sea, into which they were dumped by the ancient smelters. However, a large portion of the lead produced still comes from the old dump ores, which are treated in a noteworthy manner. The richest part of their contents were long since recovered and the material they now furnish contains only 2.5 to 3% of lead, and this in a highly oxidized state, which increases the difficulty of mechanical concentration. The dressing works for this purpose are among the largest of the world, about 1000 tons of ore being crushed per diem. Included in this are several hundred tons of tailings, with 1.5 to 2% lead from the dressing-works operated in the seventies and eighties, the present tailings carrying away only about 0.5% lead.

For many years zinc carbonate occupied the first place in value among the mineral exports from Greece. Last year it was displaced for the first time, in spite of the fact that the island Antiparos figured with an output of 2500 tons. The latter industry, however, has already come to a standstill on account of unfavorable financial results, and Laurium's production of calamine has been diminishing for several years, both in quantity and quality; the rich ores which formerly contained at least 60% after roasting have become scarce, and exploiters must now content themselves with the poorer ores, which are also higher in iron.

On the other hand, the manganiferous iron ore of Laurium in 1896 attained to the highest place on the export list, both in tonnage and value, aided in the latter respect by the advance in price. Only 2000 tons of it went to Philadelphia, the remainder being shipped to England and Germany. The ore is a manganiferous hematite of black color, containing 11 to 18% manganese, 35 to 43% iron, and 0.02% phosphorus, while it is low in silica and high in lime, and is evidently, like all similar calcareous red iron ores, an alteration product of spathic iron. A proof of this origin is that the bodies of iron-manganese carbonate occurring in certain parts of the bed, of white color and resembling marble, with iron pyrites disseminated throughout, after burning in shaft furnaces are neither chemically nor by appearance to be distinguished from the natural ores. Of the total product of 166,800 tons about 4000 tons were of this calcined ore.

The ordinary iron ores showed a still more important increase, as compared with the previous year, since besides Seriphos with 140,000 tons, the Grammatikon Mine, near Marathon, turned out 80,000 tons. The island Kea will furnish another quota during the present year.

The pure and very white magnesites of Eubœa, crude and in bricks of the burned mineral, meet with ready sale. The quality of the latter improves from year to year; it contains 94% of magnesia, and its sp. gr. attains the high figure of 3.6—a proof of its extraordinary density, which hardly diminishes even in the greatest heat. For special purposes, when it is important to prevent even the most minute cracks, magnesia bricks have been made with an admixture of chromite, whereby on the one hand the fireproof quality is not deteriorated, while on the other hand the slight expansion at extreme temperatures is neutralized.

New York and Philadelphia took 6000 tons crude magnesite and 200 tons magnesia brick in 1896.

The export of Naxos emery increased to 3650 tons, of which 1300 tons went to America, although the price per ton at Naxos was advanced to 106.50f. by the Greek government, which holds a monopoly on the mining of emery.

It has been known since ancient times that no other country can compare with Greece as regards abundance and excellence of marble. The want of means necessary for mining and transporting it have hitherto prevented the exportation from becoming a large industry. A first attempt to improve these conditions was made about 15 years ago in the island of Paros, when all the old quarries were cleared up and connected with the harbor by a railroad. But the enterprise, at first in Belgian, next in Greek hands, came to grief, because the bluish stone of inferior quality put upon the market found no takers at corresponding prices, and in the meanwhile the means to reach the strata of colorless, semi-transparent marble, obtained formerly in subterranean openings, had failed. More successful, however, appears to be the recently projected enterprise for mining on a large scale the Pentelicon marble, for which purpose a concessionary railroad from Piræus by Athens to the Pentelicon has already been commenced, so that the wealthy cities of Europe and America may soon adorn themselves at moderate prices with this admirable stone, to the indestructibility of which the ruins of the Acropolis attest.

MINERAL PRODUCTION OF GREECE. (*a*) (METRIC TONS AND DOLLARS; 1 drachma = 20 cents.)

Year.	Blende.		Calamine, Calcined.		Chrome Ore.		Emery.		Gypsum.		Iron Ore.		Iron Ore, Manganiferous.	
1892..	2,395	(*b*)	25,300	$624,160	1,470	$17,934	1,479	$19,227	100	$1,800	244,403	(*b*)	157,756	$371,190
1893..	2,727	$47,480	19,862	385,300	1,820	20,800	2,449	31,837	133	2,180	67,670	$81,200	121,352	281,540
1894..	1,030	16,700	19,800	348,480	6,000	72,000	3,570	46,410	85	1,540	149,628	157,000	76,277	(*b*)
1895..	2,710	45,230	21,321	395,250	2,740	34,120	3,055	45,825	113	2,030	150,210	179,250	152,123	342,670
1896..	1,750	31,520	20,950	301,680	1,600	22,400	3,650	77,745	120	2,150	225,600	314,700	166,850	417,120

Year.	Lead, Soft.		Lead Ore, Argentiferous.		Lead, Argentiferous.		Lead Fume.		Lignite.		Magnesite, Crude.		Manganese Ore	
1892..	(*b*)	(*b*)	3,550	(*b*)	14,426	(*b*)	1,129	$9,800	6,550	(*b*)	10,100	$28,400	11,716	$84,120
1893..	(*b*)	(*b*)	4,625	$97,122	14,534	$1,156,900	1,913	17,800	12,134	$24,268	8,815	24,680	5,250	36,760
1894..	479	$24,910	*c*4,376	117,726	13,955	1,073,500	1,680	13,700	122,117	181,471	13,262	59,340	50,573	133,954
1895..	7	490	1,580	147,400	19,838	1,656,500	1,406	12,020	17,748	55,500	11,096	33,288	7,250	42,750
1896 .	480	28,900	3,200	137,600	14,700	1,325,095	1,550	10,510	14,000	28,000	11,600	34,800	15,500	93,600

Year.	Millstones. Number.		Puzzolan.		Sea Salt.		Sulphur.	
1892..	10,290	$5,200	10,809	(*b*)	21,600	$345,600	1,525	(*b*)
1893..	(*b*)	8,000	32,348	$29,100	18,329	293,264	2,400	$67,380
1894..	11,448	5,535	28,550	25,700	21,310	332,436	1,946	(*b*)
1895..	(*b*)	10,020	29,810	26,830	22,238	335,800	1,480	31,970
1896..	(*b*)	8,200	31,300	30,100	22,800	344,280	1,540	35,500

(*a*) The figures for 1892 are from *L'Industrie Minerale en Grèce Rapport Pour L'Exposition de Chicago;* those for 1894 were furnished by the Bureau of Mines, Athens, for the *Second General Report of the Mineral Industry in the United Kingdom;* those for 1893, 1895, and 1896 were communicated to THE MINERAL INDUSTRY by E. Grohmann, Seriphos. The irregularity of certain of the figures for 1894 as compared with the preceding and following years indicates that they were reckoned in an entirely different manner, especially the figures for manganiferous iron ore, manganese ore, and lignite. Besides the substances reported in the above table there were produced in 1894, 602 tons of calcined magnesite, value $6020; in 1893, 1275 tons of magnesite brick, value $25,500, and in 1894, 873 tons, value $17,460. (*b*) Not stated in the original sources. (*c*) In addition, 14,939 tons of undressed lead ore were reported.

INDIA.

ACCORDING to the last report of the Chief Inspector of Mines for British India, the total number of mines and quarries in the Empire is 367, of which 174 are coal mines, 66 mica mines, 35 limestone quarries, and 46 quarries of other kinds of stone. The remainder of the list is made up by plumbago, manganese, gold, ruby, salt, slate, soapstone, and borax mines. The total number of persons employed in the coal mines was 25,768, but returns were not complete.

In 1878 there were only 46 collieries in India, of which 44 were in Bengal. In that year the total production was a little over 1,000,000 tons. In 1890 there were 82 collieries, and the output was upward of 2,000,000 tons. The greatly increased production of coal has not, however, lowered the price. On the contrary the demand is increasing, and coal and coke appear to be displacing wood for private and public cooking purposes in various parts of India. It is stated that at certain collieries the cost of production does not exceed two rupees per ton.

The mica industry in the Nellore district of the Madras Presidency has recently taken a great extension. Up to 1895 mica was produced there in small quantities for local use, but was not an article of export. During 1895 and the first half of 1896, 227,175 lbs. of mica were exported from these mines to the port of Madras.

The iron resources of India are vast, the most promising deposits occurring in the metamorphic rocks of Chanda in the central provinces. The principal ore is hematite, of which there are large bodies of exceptional purity. At Lohora there is a hill three-eighths of a mile long and 120 ft. high which is entirely composed of compact crystalline hematite, assaying 70% Fe; at Dewalgoan there is a hill of similar ore 250 ft. high; at Pitalgon there is an immense amount of ore of similar character; while at Ratanpur there is a lode of brown iron ore 40 to 50 ft. thick. These ores are all of them free from phosphorus. The deposits are only a few miles distant from the Great Indian Peninsular Railway and only 400 miles from Bombay. Freight from Bombay to London is now less than 15s. per ton, and a consignment of hematite could be brought from the Chanda district to English blast furnaces in less than a month. The climate of the district in which these deposits are situated is very healthful and the labor available there is exceedingly cheap, unskilled workmen receiving only two annas (less than 4 cents) per day, while those who have had experience in mining and smelting iron ores by native methods get only twice that amount.

Mineral Industry of Burma.*

By A. H. Bromly.

Amber occurs in the Hukong valley of the Bhamo district of Upper Burma. The practice of the government of India in regard to many of the mines of precious stones, etc., which may be in out-of-the-way districts and are worked entirely by natives, is to sell the right of collecting the duty (in this case 5% *ad valorem*) at public auction. Hence returns are difficult to obtain and usually incorrect. For amber the returns for 1895 have not been furnished. The output in 1894 was given as 87 lbs., and for 1893 as 549 cwts. Work is pushed or slackened according to the market. Dr. Fritz Noetling, government expert, reports as follows: "The amber of Burma, known as burmite, differs from ordinary amber (succinite) by containing no succinic acid. It occurs abundantly, but is of less value than common amber, on account of its color and the occurrence in it of fissures filled with calcspar. The amber occurs in a soft blue clay, probably of Lower Miocene age, in lumps of various sizes up to that of a man's head, and rounded or flattened in shape. The color varies from light yellow to dark-brown and it is fluorescent." The price at the mines usually averages from 6 to 15 rupees† a viss (3.65 lbs.), or about $1 to $2.50 per kgm., and in Mandalay from 15 to 40 rupees ($2.50 to $6.66 per kgm.). The yellow form is rarer than the red.

The amber mines are worked by Kachins in a fitful and haphazard manner. A shaft is sunk to a depth of 70 to 80 ft., and in the event of a deposit being struck a drift is run in the bed. Lumps are found weighing as much as 1000 lbs., which are carefully removed and sawed into shape and size.

Antimony is met with in the Yeu district of Upper Burma, near Udaung in the Indaing township, but is not worked. More important deposits exist in the Kyaukkaung tract of Tenasserim, Lower Burma, and have been officially reported upon as being considered of immense value. "The stibnite here, as in Borneo, occurs in irregular deposits of a pockety nature, often massive and of great value, lining and filling fissures in the limestone in immediate junction with the newer slates that overlie the limestone. The method of working is fossicking and poking about in likely localities till a run of paying ore is found, and blasting out the pockets, usually in open casts. Rarely, and then only in the very richest ground, is sinking or driving on any scale attempted. It is considered cheaper to smelt on the spot and ship the crude regulus. The necessary plant can be erected at small cost."

Coal is somewhat widely distributed, and its production forms a valuable and growing industry. None of the Indian coal, of which there is an inexhaustible supply, belongs to the Carboniferous age; it is only in Assam and Upper Burma that valuable Cretaceous coal is met with. Near Kalewa, on the Chindwin River, a 10-ft. seam of good Cretaceous coal occurs. In Pegu (Thayetmyo), and also at Mogaung, the deposits are Tertiary. Coal is also met with in Tenasserim (Mergui) and Arakan. The coal found on the Great Tenasserim River is favorably reported

* The writer is indebted for some of the facts and figures in this article to the excellent official *Review of Mineral Production in India* for 1895, by Dr. Watt.

† In this article the silver rupee is taken at 1s. 2d., say 28c.

on by T. W. H. Hughes, the three seams aggregating 7 ft. 6 in. in thickness and being stated to be above the average of Indian coals. The field is estimated to contain 1,000,000 tons, but is badly situated as regards labor and transportation. Mr. Hughes estimates the cost of mining 10,000 tons per annum at about 8s. ($1.92) per ton, transport to Mergui 2s. 10d. (68c.), making a total of 10s. 10d. ($2.60) delivered at seaboard, whence it could be shipped to Rangoon and Penang.

Coal occurs in considerable quantity in the Shan States. That of the northern states is reported by Dr. Noetling to be poorer than the southern, but more favorably situated for working; owing, however, to the want of means of transportation to the Irrawaddy, these fields are not likely to be worked for some time. Dr. Noetling considers the coal to be of late Tertiary age, but its exact position is uncertain. The seams vary from 2 in. to 30 ft. in thickness, the average being 5 or 6 ft. The following coal fields are mentioned: (1) Lashio, (2) Manzè-Namma, (3) Theinnimyo, (4) Namyaw valley, near the village of Meungyaw, (5) Mansang and Meungpat, (6) Nong-Hswan in Kyithi-Bansan State, (7) Silan on the Nammaw, or Shwely River.

The chief fields in Burma are the Thingadaw, Shwebo district, on the west bank of the Irrawaddy, consisting of three localities with seams of 4 to 5 ft. in thickness, mostly of good, hard, compact coal, which often contains imbedded lumps of amber-like resin. On this field the chief mine is at Kahbwet, near Shwebo, worked by the Burma Coal Company under a 20 years' lease of 3 sq. miles and 350 acres.

There is no official record of coal production in Burma prior to 1892. Since then the output has been a constantly increasing one, as the following returns show: 3670 tons in 1892; 9938 tons in 1893; 12,111 tons in 1894; and 17,289 tons in 1895. The output of Kahbwet is largely taken by the government for railway purposes. According to a note in the *Engineering and Mining Journal* of August 29, 1896, the composition is given as fixed carbon 33.57%, volatile 57.93%, sulphur 0.33%, and ash 8.50%. The cost of production in Burma is difficult to estimate with any accuracy. Mr. Grundy states that in India "it is not unusual to work out coal at an all-round cost of not more than 1 rupee 8 annas (42c.) per ton; good coal has sold at the pit as high as 5 rupees 4 annas to 5 rupees 8 annas ($1.47 to $1.54) per long ton." The Kahbwet pit is worked by incline and local Burman labor costs 4 to 6 annas per day (7c. to 10c.) and imported natives of India about 10 annas (17c.). Assuming an average cost of 2 rupees (56c.) per long ton for India, the cost in Burma, when working under approximately similar conditions of mining, should not exceed 3 rupees (84c.) at pit's mouth. The government duty is 2 annas (3½c.) per long ton. The value of Burma coal is 5 rupees 8 annas ($1.54) per ton.

Copper ores exist in various parts of Burma; but, as in India, the industry is dormant, awaiting modern enterprise and capital. Copper ores are found in the hills east of Kyannhyat, Ruby Mines district; also in the Katha district. The writer has seen copper carbonate from near Mansi, Katha district, and recently assayed samples from another locality, same district, which were practically solid carbonate of copper, with little or no gangue, and carrying 3 dwts. in gold. This was said to be a large deposit, easily accessible from the Irrawaddy, upon the banks of which smelting works could be advantageously located.

Fuller's Earth.—"Soapsand" is found in the Myingyan district. The production in 1894 was 190 long tons; in 1895 it was 197 tons; value, 498 rupees ($139.44). It is used for cleaning hair, dyeing cloths, and weighting paper. It is also found in Manipur.

Garnet is fairly abundant throughout Mogôk township, in the eastern portion of Twinnge township of the Ruby Mines district, and in parts of Momeik States.

Gold is very widely distributed throughout both Upper and Lower Burma, and little doubt exists that much was won by the ancients. The wealth of the country and the wide extent of the old gold washings prove this. It is found in the river sands of Tenasserim; in lodes near Tavoy; Momeik, Ruby Mines district; as dust in the dry nullahs of the Endaing jungles; near Toungoo in Yeu, by washing in the Shwegyin, Tazé, and Indaing townships; and in the gold fields of Wuntho, Katha district.

Returns of gold by washing are non-existent; as the working of alluvial is free to all, the output is unknown. The average earnings have been estimated at about 3 annas (5¼c.) per day. Far up north of Myitkyina some rich ancient mines are reputed to exist; and certainly small patches of sand are diligently washed by natives in those upper reaches of the Irrawaddy for a return, it is believed, of about 8 annas (14c.) per head per day. The sands of the Shweli are also auriferous. Gold is known to occur in the Shan States, but owing to the enormous distances and inaccessibility little or no prospecting has been done. The Wa country is reputed to be very rich, and the gold untouched, the natives regarding it with sacred veneration. The Was being head-hunters and not yet subdued to the trammels of civilization, prospecting remains for the future.

It is stated that operations will shortly be started on a working scale upon a reef in Tavoy, the prospects being reported as encouraging. The writer has seen quartz samples from that district running 3½ ozs. per ton in gold; location, 20 miles from the sea; water transportation and cheap labor.

The chief interest in gold-quartz mining at present is centered in the Wuntho division of Katha district, where quartz mining has been taken up by Europeans. Wuntho is on the Mu Valley Railway, about 160 miles northwest of Mandalay, and within 36 hours' journey of Rangoon.

The geology of the country shows igneous rocks disturbed and metamorphosed by volcanic agency. The rocks are well described as "trappoid," vast sheets of feldspaltic traps overlying chloritic schists and clay slates. Irrupted dykes of syenite and greenstone, combined with the general features of the country, point to there having been several periods of activity, but dense jungle and heavy overburden make examination difficult. The overburden, which usually carries float quartz, has been washed over in most parts to an extraordinary degree by natives. The old workings extend along the flanks and crests of ridges as much as in the hollows. So closely has the country been gone over that ordinary prospecting by tracing the gold back up-stream to its home is practically impossible.

Although considerable ground has been taken up, the only reef so far developed and worked is that belonging to the Choukpazat Gold Mining Company, Limited, located 11 miles from the Nankan railway station. The lode is a bedded lenticular body in clay slate, of an average width of 3 ft., striking northeast and

southwest and dipping 20° to 25° south. To date it has been proved 200 ft. in depth by shafts, becoming more compact, defined, and of better grade in depth. The pay streak is quartz carrying iron and copper pyrites with small quantities of other minerals. The mill returns show an average of 5 to 7% of sulphurets, value about 2 ozs. in gold per ton. The ore is very patchy, and although chutes are met with running several ounces, the milling average over 3000 tons is about 11 dwts. The bullion runs .850 fine. The ore is treated in a 10-stamp mill, built by Bowes Scott & Western, of London, the pulp from the plates passing over two 6-ft. corrugated belt Frue vanners, supplied by Fraser & Chalmers (London branch). The concentrates from these are at present stored with a view to future treatment. So far only day-running has been the rule, as night work was found very trying to the Eurasian and native subordinate staff, who were attacked with chills and fever. Consequently the monthly output is only about 300 tons, but even on that small amount the total expenses of mining, development, tramming a distance of 1 mile, milling, and general charges, exclusive of management, only amount to 5 dwts., or about 15 rupees ($4.20) per ton. It is hoped shortly to increase the plant and arrange for a 1000-ton output per month, when the costs should be less, owing to the standing charges being practically the same in both cases.

Natives of India are paid 10 annas (17½c.) per day, local Burmans 8 annas (14c.), Chinese smiths 2 rupees (56c.), and carpenters 3 rupees (84c.) per day. The mining is almost exclusively done by local Burmans, who previously had no knowledge of any craft but husbandry and forestry. They work on day wages, and working double-handed 6 men will drill 6 holes 2-ft. deep, and clear the rock per shift. This makes the cost of hand-drilling 4 annas (7c.) per ft. Driving an adit 7 by 7 ft. full costs 15 to 24 rupees ($4.20 to $6.72) per ft. advance, according to whether working in a full breast of quartz or only partly so. Driving and sinking, including the hand carrying of rock out of the shafts and winzes, tramming to dump, sorting, and the getting of say ½ ton of mullock to 1 ton of quartz, costs an average of 12 rupees ($3.36) per ft., or 6 to 7 rupees ($1.68 to $1.96) per ton of quartz, including the supervision of a European miner, drill-sharpening, explosives, and everything except general management. It is a testimonial to the adaptability of the Burman when properly handled, that 2 men are able to drill 4 ft. per shift in hard quartz and perform their share of tramming out the rock won, the said men being merely raw jungle dwellers.

Milling 300 tons per month costs 4 rupees ($1.12) per ton, exclusive of superintendence, but inclusive of about 1 rupee (28c.), with government duty, for firewood. Stripping soft ground costs 8 to 10 rupees ($2.24 to $2.80) per 1000 cu. ft., according to the length of lead.

Iron ore is not known to exist in any quantity in Lower Burma. The writer has specimens of excellent specular or hematite ore from near Wuntho, Katha district, and also others of similar ore from the same district further north. Doubtless, considering the rapid opening up and development of Upper Burma, these as well as other known mineral deposits will receive attention.

Jade.—This mineral is a variable silicate of lime and magnesia, with some iron, and sometimes soda, and is known as nephrite (a variety of actinolite) or jadeite, belonging to the amphibole group. It is green or whitish green and

gray, very hard, and is much prized by the Chinese and other oriental races as an ornamental stone for making bracelets, anklets, etc.

The jade mines are situated in the north and northwest portion of Upper Burma. Owing to their remote location and the secretiveness of the Chinese workers it is difficult for Europeans, other than government officials, to learn much about them. Consequently the following details are mainly from the official reports of Drs. Watt and Noetling: "Jade is extensively worked in the Mogaung sub-division of Bhamo, and in the Upper Chindwin and Katha districts. Its color varies from pearly gray to dark green, and it is in very general demand throughout China. A variety highly valued by the Chinese is pale gray, with bright apple-green spots, clouds and veins."

The Bhamo jade-producing country is by far the most important, lying between the 25th and 26th parallels of latitude, and bounded by the Chindwin and Mu Rivers on the west and east respectively. The mines now being worked are known as Tawmaw, Paungmo, Iku, Matiemmo, and Nienno. There are both "pit mines" and "quarry mines," the former being situated on the banks of the river Uru, and the latter near the village of Tawmaw. In the Tawmaw quarries the jade occurs as a vein in a blackish-green igneous rock. Dr. Noetling describes it as a "purely white crypto-crystalline mineral, much resembling the finest marble, containing here and there green specks of various sizes, which form the jade proper."

In the pit mines it occurs in boulders derived from other localities. At least 500 men are engaged every season in working the quarries at Tawmaw. The rock is won by fire setting, and although the Tawmaw quarries have been worked for several years the stone is by no means exhausted.

In the pit mines the pit is merely sunk and the jade boulders recovered. Good jade is also sometimes found in laterite.

The jade quarries of Katha district are situated about 12 miles north of the Mawhun hill, or about 60 miles from Katha town. The jade from these quarries is inferior to that from Bhamo. The output from Bhamo in 1894 was put at 727 cwts., value 115,091 rupees ($32,225). The output in 1895 was returned at 37,513 pieces, value 130,543 rupees ($36,552). The remarks as to accuracy of returns under the head of "Amber" also apply to jade. A royalty of 33⅓% is levied and the right to collect this duty, together with that on amber, is sold annually by auction. The Chinese licensee for 1892 purchased the privilege for 36,000 rupees ($10,080). The extension of the Burma State Railway to Mogaung, now being rapidly pushed on, will render these mines more accessible.

Lead.—Silver-lead ores (galena) exist both in Mandalay and Katha districts, with old workings, some very extensive, but not now worked. The government is not favorable to the working of lead ores by natives, on account of the possible use of the metal for casting bullets. A lease of a mine in Bawzaing, Southern Shan States, has, however, been granted to a Chinaman for 5 years from 1892, who is working same. The writer has an excellent specimen of silver-bearing galena from the neighborhood of Wuntho.

Magnesia.—Deposits of this mineral are found in Arakan and Manipur.

Manganese.—An extensive deposit of manganese ore has been reported to exist in Lower Burma. It unites three localities, the bank of the Thugoo stream, the

bank of the Thirabuen, and the Great Tenasserim River. The ores are stated to consist of black and gray oxides and wad, but further particulars are not available.

Mica is found near Chaukkyi in Twinngé township, and in various parts of Mogôk township; 7 or 8 men were engaged working during 1895. About 32½ cwts., value 80 rupees per cwt. (44c. per kgm.), was obtained.

Petroleum.—It is stated that the petroleum industry dates back some 2000 years. It is increasing and prosperous. The output in 1894 was 10,920,951 gals., value 1,057,545 rupees ($296,113); in 1895 it was 12,975,995 gals., value 1,534,951 rupees ($429,786). The increased production in 1895 amounted to 19% over the previous year, and the price rose in addition. The output in 1895 may be allocated as 11,927,544 gals. from the Magwe district; 846,262 gals. from Pokokku; 148,539 gals. from Kyaukpyu; and 53,650 gals. from Akyab.

In this connection it may be found possible in many cases to avoid the hauling of heavy boilers and expensive steam plants, in laying out mining plants in districts difficult of access in Upper Burma, by the use of oil motors and Burma oil. The writer found that with oil at 7 annas (12c.) per gal. delivered at Sagaing, the cost of power in the Wuntho gold fields was about the same for fuel whether using wood or oil; but in favor of the latter is the absence of drivers and firemen, who are difficult to obtain and unreliable, besides saving in first cost of plant.

Rubies.—The principal mines are at Mogôk, 90 miles northeast of Mandalay, being reached by 50 miles of road from the Irrawaddy River. The district is hilly, the mines being at an elevation of 5000 ft., and fever is prevalent, probably owing to chills and the scarcity of fresh food. These mines are now held by the Burma Ruby Mines, Limited, the first lease of 7 years expiring at the end of October, 1896. The company has the right of mining over an area of about 60 sq. miles, and to license natives to work in ground they may see fit to grant. The royalties collected from these lessees amounted to 468,039 rupees ($131,051) during the fiscal year ending February 29, 1896. In return for this the company paid a yearly rent of 400,000 rupees ($112,000) and one-sixth of the profits to the government. The capital of the company is £300,000 ($1,500,000). From various causes the operations of the company have not been particularly successful financially, and the lease has been renewed for 14 years under the following terms: The rent is reduced to 315,000 rupees ($88,200), but the government is to receive one-fifth of the profits in lieu of one-sixth. Out of 10 lakhs* ($280,000) rent in arrears, 6 lakhs ($168,000) are remitted, while the balance is to be paid by half the net profit remaining after the payment of a 5% dividend.

Recently a new ruby tract at Nanyaseik, in the Bhamo district, was declared open, and operations, entirely by natives, are said to be extensive and profitable. The rubies occur in limestone, but work is confined to washing in the alluvial resulting from weathering.

Another ruby tract exists at Sagyin, 24 miles north of Mandalay, but at present is reserved by the government.

At Mogôk the rubies occur in hard cystalline limestone, also in the filling of rock fissures, and in gravel resulting from degradation of the country.

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* One lakh equals 100,000 rupees = $28,000.

The operations of the Ruby Mines Company are twofold—washing the gem-bearing gravel (byón) and driving into the hills to reach pipes and fissures carrying gem stuff. After washing, the gems are recovered by hand sorting. Spinels, sapphires, and garnets are also found. The gems appear to be valued at about 5 rupees ($1.40) per carat. Heavy expenses are incurred in unwatering the valley bottoms to get at the byón, and naturally the best and more accessible ground having been gone over by natives in the past, remunerative operations become increasingly difficult. However, in spite of this the position of the company is improving, as the following figures show:

Year ending.....	Feb. 28, 1894.	Feb. 28, 1895.	Feb. 29, 1896.
Number of loads washed.....	20,089	61,080	148,740
Value of rubies found.....	$24,132	$78,572	$133,017
Royalties received.....	83,555	87,636	131,051
Expenses in Burma, exclusive of rent...	88,662	86,281	107,922

Reference to the balance sheet shows that the cost of mining, washing, and carriage of byón for the last year was 2 rupees (56c.) per load, exclusive of management and general charges, and the value 3 rupees 2 annas (88c.). The difference between these figures and those given further on by Mr. Wynne is probably due to the cost of sorting, as well as washing, being included here, in addition to the charges for unwatering to enable excavation to proceed.

For some 5 years the company have only paid one-half of the rent due to the government, or 2 lakhs ($56,000) per annum. By this means the past year's work shows a profit of £17,000 ($85,000). Had the full rent been paid this figure would have been reduced to about £5000 ($25,000).

Some figures of cost were recently given in a paper by Mr. T. Trafford Wynne, read before the Institute of Mining and Metallurgy, from which the following are taken: The daily pay of a coolie is 1 rupee (28c.), some of the best drillers receiving 1 rupee 8 annas (42c.). Driving a tunnel 8 by 7 ft. through hard calcspar rock, using partly machine and partly hand drilling, costs 80 rupees 9 annas ($22.56) per ft. Another tunnel 10 by 10 ft., inside timbers, driven partly through hard gneissic rock and partly through decomposed sandy rock, much trouble owing to water and unstable ground necessitating difficult timbering, costs 42 rupees ($11.76) per ft. A third tunnel driven through heavy red clay with peaks of calcspar and occasional boulders, size 8 by 7 ft., costs 32 rupees ($8.96) per ft. The cost of excavating gem ground was 0.43 rupee (12c.) per load of 15 cu. ft., and where similar ground contained large boulders, 0.64 rupee (18c.), both cases including delivery to washer, but exclusive of management, etc.

Mr. Wynne states that drilling on a contract rate of 4 annas (7c.) per ft., the best Maingtha drillers make 2 rupees (56c.) per day. This means that one man drilling single-handed has to put in 8 ft. per day, or double-handed the pair have to put in 16 ft. The writer has not been fortunate enough to come across any labor in the East capable of this, in hard ground; but his experience at Chouk-pazat is that with labor much less efficient the cost of drilling is the same, viz., 4 annas per ft., but on day-work and with a very much less cost per foot of advance, taking the Ruby Mines Company's cost for drifting soft ground at 0.43 rupee

(12c.) per 15 ft., it works out at 28 rupees ($7.84) per 1000 cu. ft., a figure far above the writer's experience and Indian costs generally.

Salt.—The manufacture from brine wells extends all through Burma, and from sea water at Amherst. The production was 13,234 tons (of 2240 lbs.) in 1892–3; 17,516 tons in 1893–4; and 19,245 tons in 1894–5; but room for expansion still exists, as nearly 50,000 tons are imported annually.

Soapstone.—This mineral is quarried in the Prome, Pakôkku, Katha, Bhamo, Kyaukpyut, and Minbu districts. On the hills round Prome it lies about 18 in. beneath the surface. The Minbu and Kaukpyu quarries are in a group on the borders of Arakan and Upper Burma, the steatite occurring in beds of 6 to 12 in. thick. In Bhamo district soapstone is met with north and northeast of the jade and amber mines. Several of the Minbu quarries are annually leased to Burmans, the leases being disposed of at auction. In some cases, besides a lump sum down, a royalty of 10 to 20% is levied by the government. The Minbu production in 1894 was 16 tons, and in 1895 19 tons.

It is presumed that the future may see a greatly increased foreign demand, in competition with German steatite.

Tin.—The only known deposits of any extent are those occurring in the Mergui district of Lower Burma, being a portion of the stanniferous belt stretching throughout the Malay Peninsula. The lodes outcrop in granite and are mostly disturbed and shattered by earth movements. The lodes, so far, have been scarcely touched commercially, except for a little fossicking by Chinese. The supply is therefore obtained from the alluvial deposits resulting from the degradation of the hills carrying the tin lodes. Large pits are opened, to an occasional depth of 30 ft. or more, and the pay dirt (kareng) raised and washed. The resulting black tin is smelted in small furnaces using charcoal fuel, and the block tin mainly shipped to the Straits.

There are four principal fields: (1) Maliwun, (2) Karathuni, Banhuni, (3) Bokpyin, (4) Migyaungchaung, Yayugon. Figures of production are unreliable and inaccurate. That for 1894 is given as 98 tons (2240 lbs.), and for 1895 it appears to have been from 20 to 25 tons. One lease has been granted to a European.

Mining Regulations, etc.—The province of Burma is under the administration of the government of India, and mining is therefore subject to the Indian rules and regulations. These comprise: (*a*) Exploring licenses; (*b*) prospecting licenses; (*c*) mining leases.

(*a*) Exploring licenses are granted for one year at a nominal sum. They entitle the holder to search government land for minerals and give him preference when applying for prospecting licenses or mining leases over rival unlicensed applicants. One of the curiosities of the rules in this connection is that the explorer may not turn up the ground to a greater depth than 2 ft.

(*b*) Prospecting Licenses.—Should the prospector desire to investigate any ground to a greater extent than the exploring license permits, he may apply for a prospecting license. Under this a definite area of ground, varying in size with the minerals searched for, is marked out and granted at a moderate rent, plus royalty, for one year, with possible renewal for a second year under given circumstances. To obtain this the applicant must be "approved" by the powers that be and prepared to deposit not less than 1000 rupees ($280) security. This

license empowers the holder to sink shafts and generally open up the ground, subject to regulations and rules duly set forth, some of which would seem obviously intended to be evaded. Should he not continue his operations, he must fill in, fence, or generally make good the ground if called upon.

(*c*) Mining Leases.—Having found sufficiently good prospects the miner next applies for a mining lease. These are granted over a reduced area, subject to dead rent or royalties (per schedule) for a period not exceeding 30 years. In order to prevent overspeculation, and the taking up of land for purely speculative purposes, the regulations comprise one exceedingly important to the ordinary prospector and it is here given in full: "(VI.) The lessee shall not assign the lease, or underlet the whole or any portion of the premises comprised in such lease without the previous consent in writing of the local government; and no such assignment or underlease shall be made to a company without the previous sanction of the Governor-General in Council."

This means that the average prospector should not try Burma, as unless he is prepared to find the money, or has the money behind him to continue operations in his own name, in the event of the government refusing him leave to transfer he will lose his labor and time. In short, the government expects the capitalist to personally do his own prospecting and mining, or the prospector to be a capitalist and mine owner, which is contrary to usage in the mining world. Of course the regulation is framed to afford power to prevent speculators locking up land to the detriment of would-be workers, and the government would doubtless always be ready to consider a case on its merits, and if the intention of honest work were evinced to facilitate same.

The working or dry season in Burma extends from the end of November to about June 1, varying with the locality. The personal element enters into matters considerably in India, an individual known to the government being able to push matters much more rapidly than a stranger. Owing to this fact, complicated regulations, relations between the different departments, and the varied languages and labor of the East, intending speculators should be careful to obtain the services of men experienced in the country and its ways. That done, there is little doubt but that the mineral riches existing in Burma, which are practically unprospected, will yield a rich return to the early exploiters.

ITALY.

ITALY is not rich in mineral wealth, though it is believed by many that its resources have been underestimated by geologists, and that closer search will disclose deposits hitherto overlooked. The most important mineral district is that of Southwestern Sardinia, of which Iglesias is the center. In Sicily there are the famous sulphur mines, and in the Alps there is a considerable mining industry, particularly in the northern parts of the Provinces of Brescia and Bergamo. The Apennines are remarkably poor in minerals, and though in the northern half there are a few good mines, south of Rome there is practically nothing. The mining industry as a whole is not prosperous. Since 1891 there has been a steady decrease in the number of mines worked, number of persons employed, and gross value of production. In 1891 the last amounted to 79,890,726 lire; in 1895 it was only 49,138,129 lire. The value of the sulphur forms nearly one-half of the total value of mineral products. The latest official statistics of the kingdom are those for 1895.

Coal is found only in small quantities, and nearly all is lignite. Outside of Tuscany the output is trifling, except for one mine in the Province of Perugia. The annual average production is less than 300,000 tons, and this shows a marked tendency to decrease. The number of persons employed in coal mines in 1895 was 2361; the quantity of coal mined per person was 129 tons. It is said that some of the abandoned mines of lignite at the foot of the Tyrolese Alps could be worked at a profit if sufficient capital were supplied for installing proper pumping and winding machinery.

The number of persons engaged in mining iron ore in 1895 was 1700. By far the most important mines are in the island of Elba, the production of which forms nearly seven-eighths of the total. These mines are owned by the government, which leases them to private concerns. The new iron works at Piombino are likely to make an increased demand for Elba ore. Outside of Elba the only mines of importance are scattered in the valleys at the foot of the Alps, chiefly in the Provinces of Brescia and Bergamo.

The number of persons engaged in sulphur mining in 1895 was 27,000. About seven-eighths of the production of the kingdom is derived from Caltanisetta, Girgenti, and Catania (central and southern provinces of Sicily), with a comparatively small amount from the Province of Palermo. Sulphur is also found in the Apennines, in the Provinces of Ancona, Forli, and Pesaro, and at Avellino in the Campania, but the total production of these districts in 1895 was but little over 19,000 tons, all of which was consumed at home. The conditions of the sulphur industry in Sicily in 1896 are described under the caption "Sulphur."

Of zinc and lead mines there were 79 in Italy in 1895, employing 10,135

persons. Next to sulphur the zinc industry is the most important of the kingdom. Its condition in 1896 has been referred to under the caption "Zinc and Cadmium." The production of lead ore in 1895 was approximately 30,000 tons, of which 95% comes from the mines at Iglesias, the remainder being mined in the northern provinces of the mainland, chiefly in Brescia.

The production of copper ore has been increasing since 1890. The number of persons employed in this branch of mining in 1895 was 1260. The most important mines are in the Province of Grosseto, in Tuscany, but there are mines at Sestri Levante, near Genoa, which yielded about 12,500 tons.

In Tuscany there are four quicksilver mines which are said to be very profitable, although the production is falling off. The number of men employed in 1895 was 501.

The gold and silver production of Italy is very small. A small amount of silver ore is extracted in connection with the lead at Iglesias. The gold mines are situated in the Province of Novara in the western section of the Alps. The most important are those at Battigio and Pestarena, the latter worked by an English company, while another English company is about to reopen mines on the shore of Lake Lugano. The ores are pyritic.

There are asphalt mines at Ragusa, in the Province of Syracuse, Sicily, which in 1895 produced 36,000 tons, or about two-thirds of the total for the kingdom. The chief mines at Ragusa are worked by the Limmer and Val de Travers companies. The asphalt is of good quality. Nearly the entire product is exported to Germany, through the port of Pestarelli, about 16 miles distant from the mines, the price f. o. b. being $4.90@$5.40 per ton. A small amount of asphalt is mined in the Province of Chieti, in the Abruzzi.

The mines of boracic acid in the Provinces of Pisa and Grosseto employed 351 men in 1895; there were 12 mines in operation. Petroleum is found in the Province of Piacenza, in Emilia, and in smaller quantity in the Province of Parma, in the Abruzzi. Rock salt exists in Sicily and Calabria, but is little worked. One salt spring in the Province of Pisa produces about 10,700 tons per annum; another in the Province of Parma produces about 600 tons. A large amount of salt is won from the sea in various parts of the kingdom.*

MINERAL PRODUCTION OF ITALY. (*a*) (IN METRIC TONS AND DOLLARS; 5 lire = $1.)

Year.	Alum.		Alunite.		Antimony Ore.	Asphaltum.		Boracic Acid.		Copper Ore.		Gold Ore.		Graphite.	
1891	1,029	$20,080	4,000	$3,840	891	$65,742	28,180	$135,780	3,831	$444,780	53,059	$565,867	7,729	$93,276	2,415	$6,593
1892	1,695	36,050	4,000	3,840	621	45,672	34,580	168,184	2,560	321,910	102,427	552,288	6,612	94,696	1,650	3,778
1893	1.545	32,485	4,200	4,031	1,193	40,402	25,980	113,160	2,847	313,170	96,299	509,509	7,393	132,746	1,465	3,080
1894	1,065	20,680	6,000	4,800	1,504	45.059	60,493	280,678	2,746	236,156	92,886	445,629	7,748	132,782	1,575	2,520
1895	b1,115	21,000	7,000	5,600	2,241	40,254	46,713	204,750	2,633	184,310	83,670	367,516	7,099	129,887	2,657	8,599

Year.	Iron Ore.		Iron Pyrites.		Lead Ore.		Lignite.		Manganese Ore.		Manganiferous Iron Ore.		Mercury.	
1891	216,486	$553,437	19,868	$54,118	30,233	$1,196,846	289,286	$441,170	2,429	$12,919	.....	.....	330	$356,400
1892	214,487	554,431	27,670	71,987	33,310	1,139,098	295,713	426,073	1,243	8,359	4,622	$8,320	325	305,500
1893	191,305	356,539	29,460	72,227	29,004	813,107	317.249	454,300	810	6,548	8,805	14,960	273	264,737
1894	187,728	417,831	22,638	59,269	29,882	758,422	271.295	378,792	760	4,700	5,810	9,296	258	203,448
1895	183,371	405,711	38,586	85,742	30,632	813,324	305,321	433.555	1,569	14,128	5,860	9,376	199	189,308

*This review is an abstract of *British Consular Report No. 417*, March, 1897.

Year.	Petroleum.		Salt, Rock.		Salt from Brine.		Silver Ore.		Sulphur, Crude.		Zinc Ore.	
1891.....	1,131	$68,420	31,285	$75,890	9,258	$53,390	2,006	$294,697	395,528	$8,905,091	120,685	$2,544,121
1892.....	2,548	150,900	15,504	49,619	8,217	38,455	1,680	345,898	418,555	7,844,333	129,731	2,722,099
1893.....	2,652	159,010	16,790	56,910	8,602	39,810	1,236	235,594	417,671	5,923,335	132,767	2,093,415
1894.....	2,853	169,452	19,467	56,974	11,326	57,861	1,103	169.445	405,781	5,053,591	131,777	1,839,791
1895.....	3,594	186,099	18,710	56,019	10,605	54,171	870	128,273	c370,766	4,134,588	121,197	1,542,905

METALLURGICAL AND REFINED PRODUCTS OF ITALY. (a) (IN METRIC TONS AND DOLLARS.)

Year.	Antimony.		Asphaltum.		Boracic Acid.		Borax.		Briquettes.		Copper.	
1891....	218	$37,714	9,375	$66,020	1,775	$177,500	2,056	$267,280	644,005	$4,099,570	5,977	$2,243,229
1892....	315	53,822	9.270	78,570	1,089	130,680	1,471	191.230	612,069	3,883,650	6,039	2,159,469
1893....	376	59,560	8,905	65,711	2,847	313,170	1,099	104.287	540,790	3,273,730	6,911	2,386,247
1894....	251	57,626	7,820	49,826	2,746	236,156	1,141	136,872	433,309	2,901,012	7,048	2,466,927
1895....	423	56,986	14,491	75,920	2,633	184,310	944	113,280	451,470	2,527,564	6,133	2,097,712

Year.	Gold—Kgms.		Iron, Pig.		Iron, Wrought.		Lead.		Petroleum, Benzine, etc.,	
1891....	284	$166,839	11,930	$315,209	152.668	$7,996,459	18,500	$1,139,600	813	$74,448
1892....	330	202,579	12,729	411,258	124,273	6,278,691	22,000	1,144,000	1,573	154.495
1893....	362	224,203	8,038	230,921	138,046	6,826,843	19,898	994,300	2,613	258,676
1894....	349	252,057	10,329	301,339	141,729	6,643,615	19,605	968,446	1,640	193,520
1895....	280	199,453	10,287	331,475	163,824	6,962,837	20,353	1,078,920	4,191	440,953

Year.	Salt, Sea.		Silver—Kgms.		Steel.		Sulphate of Alumina.		Sulphur, Refined. (d)	
1891....	347,274	$613,555	37.600	$1,203,200	75,925	$3,948,766	859	$17,108	59,350	$1,679,929
1892....	395,269	800.831	43,000	1,281,400	56,543	2,756,499	1,890	40,450	59,370	1,521,265
1893....	397,506	929,522	40,095	1,130,679	71,380	3,397,426	2,050	43,878	70,707	1,306,120
1894....	402,515	830,382	58,626	1,289,772	54,614	3,353,852	2,480	45,960	71,295	1,152,576
1895 ...	448,335	957,529	44,182	1,007,460	50,314	3,794,256	2,950	54,575	75,329	1,125,494

(a) From *Rivista del Servizio Minerario.* (b) Including 120 tons of soda alum.
(c) To this is to be added 13,340 tons of ground sulphur, valued at $58,696, and 1091 tons of sulphur from springs, valued at $12,151.
(d) The production of ground sulphur from 1891 to 1895, both years inclusive, was as follows: 95,215 (value $2,719,329), 123,260 ($3,074,019), 94,624 ($1,912,478), 90,561 ($1,654,465), 91,517 ($1,473,063). Included in these figures were, in 1893, 35,957 tons already counted as "refined," and in 1894, 35,636 tons.

MINERAL IMPORTS OF ITALY. (a) (IN METRIC TONS.)

Year.	Coal. (b)	Iron, Pig.	Iron and Steel. (c)	Machinery.	Marble, Alab'ster R'gh.	Marble, W'kd	Ores, Various. (d)	Sulphur
1891	3,916,685	137,233	80,488	23,077	296	266	7.346	20
1892	3,877,571	145,723	80,889	22,392	216	379	16,497	50
1893	3,724,401	176,978	86,190	21,403	550	245	49,267	25
1894	4,696,258	157,153	83,367	23,075	250	210	15,406	267
1895	4,304,787	135,707	81,242	28,778	344	305	13,002	255

MINERAL EXPORTS OF ITALY. (a) (IN METRIC TONS.)

Year.	Coal. (b)	Iron, Scrap	Iron and Steel. (c)	Machinery.	Marble and Alab'ster	Marble M'f's.	Ores, Various. (d)	Sulphur.
1891	13,322	143	576	1.104	69,474	53,958	327,326	269,377
1892	12,919	256	683	1,076	77,816	57,917	272,278	291.081
1893	12,655	198	1,501	1,429	72,887	56,447	307,685	310,867
1894	13,692	67	645	1,836	72,812	51,210	308,555	299,090
1895	16,497	70	559	1,805	75,462	58,370	290,147	317,566

(a) From *Movimento Commerciale del Regno d'Italia.* (b) All kinds of fuel. (c) Wrought iron and steel. (d) Including iron, copper, zinc, lead, tin, manganese, and cobalt ores. The zinc ore exports were in 1891, 104,656; 1892, 119,246; 1893, 113,218; 1894, 123 268; 1895, 111,227 metric tons. The iron ore exports were in 1891, 202,309; 1892, 124,755; 1893, 156,273; 1894, 159,205; 1895, 164,367 metric tons. The lead ore exports were in 1891, 7328; 1892, 6695; 1893, 5561; 1894, 6354; 1895, 6622 metric tons. The copper ore exports were in 1891, 10,116; 1892, 12,719; 1893, 12,811; 1894, 7789; 1895, 5859 metric tons.

JAPAN.*

THE most important mineral industries of Japan are coal and copper mining. The country has a large supply of coal, and nearly 1500 mines have already been opened.

The principal coal beds belong to the Cenozoic formation, there being comparatively a small portion in the earlier Mesozoic. It is to the Cenozoic formation that most of the coal beds found in Hokkaido and in Kiushiu (Fukuoka), the two most important coal districts in Japan, belong. The total available coal from these two districts and from a certain part of the northeastern portion of the main island, is estimated at above 600,000,000 tons, at depths not exceeding 500 ft. below sea level.

The three mines, which are collectively known as the Miike Colliery, have several coal seams, but only the first and second can be worked to advantage. The first averages fully 8 ft. in thickness, and the second lying only 6 to 10 ft. below the first, averaging about 6 ft. near the outcrops, though more irregular and uncertain in the deeper portion. The coal from the second seam is inferior in quality to that from the first.

The coal from the Miike Colliery is an excellent bituminous coal, dark-brown in color, and having a rusty appearance. When kindled, it first gives forth great volumes of black smoke and is softened like pitch, but soon hardens, cakes and burns brightly, evolving great heat with long flames. The outcrop of the coal seams strikes east-southeast and dips gently to the south. The coal fields cover 7934 acres, and are estimated to contain about 200,000,000 tons. The output of the Miike Colliery from 1885 to 1894 was as follows:

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1885.....	248,137	1887.....	317,717	1889.....	462,271	1891.....	574,330	1893.....	816,235
1886.....	277,718	1888.....	368,109	1890.....	487,641	1892.....	468,831	1894.....	622,694

The number of miners employed in 1892 was 2562, and 3469 in 1893.

The Takashima mine, also in Kiushiu, produces a better quality of coal, but its output has decreased considerably of late. The sp. gr. of the Takashima coal is 1.252, and its chemical composition is as follows: Water, 1.8%; volatile matter,

*This article is an abstract from the chapter "Mineral Industries of Japan," in a report on the *Commerce and Industries of Japan*, by Robert P. Porter, 1896.

35.5%; ash, 6.35%; sulphur, 0.72%; fixed carbon, 56.4%—total, 100.77. This coal ranks next to Cardiff coal in quality. There are four workable seams, ranging from 5 to 18 ft. in thickness, the latter being the thickest seam in Kiushiu. The cost of mining in 1894 was $1.36 (silver) per ton. In 1894 the total output was 110,246 tons.

The following table shows the quantity and value of the output of the coal mines of Japan, which produced upward of 100,000 tons each in 1894:

Location.	Mine.	Output in 1894.	Quantity Sold.	Value.
		Tons.	Tons.	Silver.
Hokkaido	Poronai, Ikushumbetsu, Yubari, Sorachi	365,846	362,078	$1,520,619
Fukuoka	Miike	667,775	622,694	1,556,814
	Namazuda	165,588	157,172	388,192
	Akaike	138,153	140,980	357,357
	Shinniu	125,561	136,264	305,537
	Kawamiya	110,437	108,244	325,108
	Katsuno	106,138	97,663	304,168
Nagasaki	Takashima	125,572	110,246	381,338
Total		1,805,070	1,735,341	$5,139,133

Of the four principal coal mines in Hokkaido, the Poronai and Ikushumbetsu have four seams, and the Yubari two, while there are as many as thirteen seams of over 3 ft. thickness in the Sorachi mine. There is besides another mine in Kamiutausuna, three seams of which are workable. Sorachi coal is of a black color, with more or less luster and more or less caking quality, according to locality. The Poronai and Ikushumbetsu coal is of a brown color and has the finest luster. The Poronai is non-caking and smokeless, while the Yubari has a brilliant luster and gives off more smoke. The Hokkaido coal beds contain 46 seams, and are estimated to hold upward of 61,000,000 tons above the water level, 25,000,000 tons from the water level to a depth of 500 ft., and 177,000,000 tons from that line to the sea level.

Iron is produced chiefly in Iwate, Shimane, Tottori, and Hiroshima. The total product in 1894 was as follows: Cast iron, 15,760 tons; wrought iron, 4815 tons; steel, 932 tons. The ores of Iwate and of the Kamaishi mines produce iron of very good quality. Iron from Shimane, Tottori, and Hiroshima is produced chiefly from iron sand and is well suited for manufacturing steel. According to the geological survey of various iron mines by the Department of Agriculture and Commerce, the quantity of ore available is as follows: Kawaishi iron mine and its vicinity (Iwate), 14,038,525 tons; Senniu iron mine (Iwate), 841,700 tons; Akadani (Niigata), 373,773 tons; Yamakoshinai (Hokkaido), 400,000 tons; the estimates given relate only to those sections which are near the surface and can be easily worked. There are other districts on which no accurate survey has yet been made, while the quantity of iron sand found in Saniu and Sanyo regions, that is, those districts in the mainland of Japan south of Kobe, is immense. Iron produced from the iron sand, found so abundantly in the Sanyo and Saniu regions, really surpasses imported iron in quality.

There are in all 64 gold mines, 80 gold and silver mines, and 25 mines which yield gold, mixed with other minerals, in Japan. There are also 71 alluvial

workings. There are only two gold mines worked at present in the island of Sado, the one called Aikawa dating back 900 years. This mine has three great fissure veins, two of which are rich. The other, called Twsushi, has been worked only a few years and its veins contain a less proportion of gold than of silver and copper.

Silver is produced in far larger quantities than gold, the total number of private silver mines being 57 and of silver-copper mines 136, besides many others where silver is produced in combination with other metals. In production Akita surpasses all other districts, its output in 1893 being $1,355,467 (silver) out of the total of $2,320,444 (silver) for the whole country. Gifu, with a total of $332,403 (silver), and Fukushima, with $255,169 (silver), come next.

The total output of gold in Japan in 1894 was 23,696 ozs., and in 1895 it was about 21,000 ozs. The production of silver in 1894 was 1,956,938 ozs. and in 1895, 1,768,250 ozs. Silver mining in Japan is now at its height, while gold mining is declining. In the principal gold and silver mines machinery of American style is in use, while in copper and lead mines machinery of German pattern is employed. Only in the Ashio copper mine are electric motors used.

Since 1883 the production of copper increased rapidly, until in 1892, the year of the greatest production, the figures reached 45,300,000 lbs. In 1893 and 1894 the production decreased somewhat, but this is attributed to the demand for labor in railway works and to the prohibition of the sale of gunpowder in consequence of the Japan-China war, but as these obstacles have been removed, renewed progress may now be expected.

The Ashio mine, which is the most important copper producer, is situated about 16 miles southwest of Nikko. More than 10 veins are worked, opened by 98 pits, employing 4732 miners and women, the latter sorting ore, and 647 smelters. The total number of persons engaged in various services connected with this mine aggregates 9462 and their families 6769, making a total of 16,231. The allowance to miners paid by the day is 13c. in money, a little over one quart of rice, 5 ozs. of fermented beans, 4 lbs. of charcoal, half a gill of oil, and 3½ ozs. of powder. Those who are paid by quantity of ore extracted receive 11c. per 82 lbs. of ore, and for every extra 82 lbs. 13c. The wages for shaft men vary from 29 to 13c. per day. At present about 103 tons of sorted ore are produced per day, which contains an average of 19 to 20% of copper. The monthly working expenses are about $47,000 (silver). The cost of production per 132 lbs. of copper, which was quoted at Yokohama at $25.50 (silver) in July, 1896, is about $17.50 (silver).

The production of the chief copper mines of Japan is as follows:

Mine.	1894—Pounds.	1893—Pounds.	Mine.	1894—Pounds.	1893—Pounds.
Ashio	13,192,468	11,594,798	Osaruzawa	1,709,298	1,050,139
Besshi	5,472,898	5,247,724	Arakawa	1,595,361	1,536,473
Ani	2,512,783	1,776,841	Yoshioka	1,414,967	1,366,967

The greater portion of the copper exported from Kobe and Yokohama is the product of the Ashio and Besshi mines, that from Yokohama being chiefly Ashio copper,

and that from Kobe, Besshi. The largest customer for Japanese copper is Hong Kong, which buys nearly 20,000,000 lbs. every year; next in order comes China, buying 6,000,000 lbs.; Great Britain, buying 4,000,000 lbs.; British India, Germany, and Korea taking the remainder, the total export amounting to 33,000,000 lbs., or 80% of the production of the country.

Petroleum sinking is an industry that has lately become successful in Japan. In 1893 the production was 3,320,707 gals. The principal districts are Hokkaido, Ugo, Echigo, Shimane, and Totomi, and of these Echigo stands foremost, its yield amounting to 90% of the total output. Next to Echigo comes Totomi, with a yearly output of about 120,000 gals. Hokkaido, Shimane, and Ugo yield less than 40,000 gals. each. The chief reason for the increase in the production of petroleum was the discovery of a new reservoir in 1890 in the vicinity of the town of Nagaoka, Echigo, and the introduction of American methods at Amase. It is confidently believe that as the wells are in general less than 600 ft. deep, there being only one or two that reach 1600 ft., the supply, judging from the experiments carried on in the western oil-yielding districts, must be quite abundant, and the more extensive introduction of American methods must mark an epoch in the industry. For the present, however, Japan buys its oil of America and Russia.

Sulphur, called in Japanese "iwo" or "yuwo," is found as a glossy product of sublimation, often covering the crater walls, crevices, and clefts of active and extinct volcanoes throughout the country. By far the greatest amount of Japanese sulphur is formed by decomposition of the sulphureted hydrogen of the very numerous *solfataras*. There are no figures of production more recent than those of 1893, when 23,889 tons were produced.

MEXICO.

THE Mexican Government collects no statistics of mineral production. The most important mineral products of the country, however, are gold, silver, lead, and copper, and reviews of these industries, together with statistics to the end of 1896 compiled by THE MINERAL INDUSTRY, will be found under those captions respectively elsewhere in this volume. The imports and exports of Mexico, reported by the Federal Government, are given in the subjoined tables. In general the mineral industry of Mexico is prosperous, and with the rapid development of the country there is a good deal of progress in most branches of it, although the production of lead and silver decreased in 1896.

"None of the industries dependent on the department of encouragement is so active in its expansion as the mining industry," said President Diaz in his last annual message. "Since 1892 the progress in mining has been remarkable, as is evidenced by the number of new grants that have been issued. The grants issued since my last report number 2053 and embrace a superficial area of 9766 claims of one hectare each. The total number of grants that have been issued under the new law is 5800, embracing an area of 42,968 hectares. It has not yet proved possible to obtain full data in regard to our mineral production, but an idea of its importance may be gathered from the returns of exports. Of $105,000,000, at which the exports were estimated in the fiscal year 1895–96, $71,000,000 represented mineral products, including not only the precious metals, which form the largest part, but copper, lead, coal, and marble in the rough. The output of gold has increased remarkably."

The mines of Pachuca, which have not been large producers lately on account of the flooding of several of the important workings, will soon resume their former output, arrangements having been consummated for unwatering them. The State of Guerrero is now attracting a good deal of attention, its well-known mineral resources having been rendered accessible by the construction of the Mexico, Cuernavaca & Pacific Railway. In Durango the great Promontorio mine maintains its large output, but other ore shipments through the city of Durango have not been so large as was anticipated when the railway was built thither. In Coahuila the Sierra Mojada mines have fallen off in production, but lately a body of ore assaying 10% Cu has been uncovered in the lower workings. The old mining district around Parral, Chihuahua, is reported to be in a very prosperous condition. The great gold mines in Las Prietas district, Sonora,

continue to be large producers. The old mining districts of Zacatecas and Guanajuato show few changes. With respect to the tin and quicksilver mines of the Republic reference should be made to the articles on those subjects.

MINERAL IMPORTS OF MEXICO. (*a*) (IN METRIC TONS.)

Year.	Coal.		Coke.		Copper.		Iron and Steel.		Lime and Cement.		Marble.	
1892*c*...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1893*b*...	254,610	$1,356,666	62,386	$315,327	688	$529,175	57,233	$3,071,824	9,662	$140,251	245	$8,862
1894*b*...	162,613	626,070	71,619	378,522	659	462,985	43,128	2,496,000	4,466	28,664	152	8,099
1895*b*...	230,595	701,913	67,853	287,075	939	600,916	33,765	2,429,136	9,877	79,162	198	4,390
1896*b*...	182,375	496,376	122,295	492,167	1,254	730,882	84,796	4,190,272	10,655	93,651	431	7,215
1896*d*...	236,124	666,774	109,314	523,410	1,068	770,618	100,955	4,575,082	9,786	93,051	410	7,427

Year.	Petroleum.		Quicksilver.		Soda and Potash, Caustic.		Sulphur.		Sulphates of Copper, Iron, and Ammonia.	
1892*c*...	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1893*b*...	22,404	$370,674	727	$603,525	834	$52,085	898	$18,996	1,442	$82,500
1894*b*...	16,061	242,027	711	506,643	1,972	113,137	234	8,878	1,511	113,496
1895*b*...	14,865	250,292	819	541,664	3,716	167,487	321	12,037	1,819	123,948
1896*b*...	23,758	500,367	854	574,153	4,795	210,909	256	6,734	2,442	166,560
1896*d*...	22,005	469,674	724	501,168	4,090	178,675	267	7,875	1,921	136,377

(*a*) From the *Boletin Semestral de la Direccion General de Estadistica* and the *Estadistica Fiscal*. (*b*) Fiscal years ending June 30. (*c*) No official figures were published for the fiscal year 1891–92. (*d*) Calendar year.

MINERAL EXPORTS OF MEXICO. (*a*) (IN KILOGRAMS.)

Year.	Gold and Silver.	Copper.		Copper Ore.		Lead.		Tin.	
1891*b*.....	$36,256,372	5,364,871	$940,920	33,600	$850	8,138,194	$1,125,468	.....	.....
1892*b*.....	49,137,304	4,348,702	860,378	123,901	8,937	10,676,548	2,363,521	.....	.....
1893*b*.....	56,504,305	12,820,282	2,265,881	435,685	11,512	1,418,976	345,646	2,862	$1,420
1894*b*.....	46,484,360	11,516,382	1,979,728	2,877,321	65,441	1,146	100	58,416	13,231
1895*b*.....	52,535,854	13,338,788	2,148,184	3,707,585	564,201	39,931,656	1,807,402	29,563	16,879
1896*b*.....	64,838,596	21,596,140	3,909,485	203,923	11,610	50,881,571	2,531,624	9,440	6,032
1896*c*.....	65,070,108	20,659,320	4,008,504	144,081	5,210	48,663,147	2,440,784	5,040	2,265

Year.	Coal.		Marble.		Guano.	
1891*b*.....	39,482,132	$160,702	1,111,612	$87,556	1,492,047	$29,000
1892*b*.....	55,969,921	221,154	1,400,355	169,654	.....	.....
1893*b*.....	8,279,968	33,960	3,367,591	349,233	559,936	12,021
1894*b*.....	49,729,184	205,605	1,039,841	108,512	.....	.....
1895*b*.....	53,192,661	232,919	1,518,827	167,136	.....	.....
1896*b*.....	66,174,597	270,176	2,378,152	263,023	1,986,256	44,762
1896*c*.....	75,540,754	312,738	2,257,810	210,836	1,001,571	12,663

(*a*) From the *Annuario de la Estadistica*, 1894. (*b*) Fiscal years ending June 30. (*c*) Calendar year.

NICARAGUA AND COSTA RICA.

By J. Francis Le Baron.

The mines of Nicaragua and Costa Rica are numerous and rich in gold and silver. Many of them were worked by the Spanish conquerors and by the Indians before them. The ruins of their dams and mills are frequently found, and their tailings yield from ¼ to 1 oz. of gold per ton.

Nearly all of the mines exploited to-day are worked by the natives with the crudest of home-made and wooden machinery. Of the few worked by foreigners with improved appliances, some are producing from 8 to 24 ozs. gold per day, and many other mines could be made to equal or exceed this output if properly handled. The unexplored country in these republics is largely in excess of that which has been explored.

NICARAGUA.

The mean width of this republic is about 150 miles, and the mean length north and south 125 miles. The whole country, except the narrow strip between the great lake and the Pacific Ocean, is covered by an exceedingly dense forest. The mining regions being mostly at considerable altitude above the sea, have a cool, dry, and healthful climate.

Placer Mines.—The principal placer mines in Nicaragua are located near the headwaters of the Prinzapolca, at the foot of the eastern declivity of the Cordilleras. They were discovered in 1889, and created a great excitement at that time. Prinzapolca River is 170 miles north of Greytown and empties into the Caribbean Sea. The diggings are 11 days' journey by boat up the river from the mouth. There are 13 mines on the Prinzapolca, the most important being the Cuicuina, La Concepcion, El Dorado, Pis-Pis, and Cuenca del Cucolaia. Placer mines are also worked on the Wawa River, and the outturn from this and the Prinzapolca is estimated at from 600 to 800 ozs. per month. There are placer mines also in the Segovia district, and in Chontalos and Matagalpa vestiges of placer mines are found that were worked in the days of the Spanish conquerors.

Gold-bearing Veins.—In the Prinzapolca and Wawa districts gold and silver-bearing veins are found in the hills above the placer mines. The district of La Libertad, in Chontales, is the oldest and best developed, although the machinery used is of the most primitive character, consisting of mallet triturators, or the

ancient *molinate* (called in Mexico *arastra*), turned by a water-wheel or mule. With this crude machinery the mines in La Libertad yield from ⅛ to 2 ozs. gold of 14 to 20 carat fineness per ton of ore. In Boaco, Chontales, there are two mines which yield 200 and 128 ozs. per month, respectively. The mining district of La Libertad and Juigalpa adjoin each other on the eastern shore of the lake, the lake port being San Ubaldo. The proposed ship canal line passes through the center of the lake about 20 miles from the port. There are over 300 gold mines in these two districts. Among the principal are Santo Domingo, El Jabali, San Juan de Canidad, El Tigre, El Calvario, La Ponzona, La Mestiza, La California, San Miguel, Virginia, El Amparo, Babilonia, Los Angeles, El Escandalo, Santa Elena, Esmeralda, El Tope, El Chamorro, San Gregorio, Animas, Calvario, La Republica. Gold has been discovered in the San Juan River at San Carlos, Mochuca, and at Zavalas, 8 miles above Castillo, and also in the two Roma rivers. Some of these veins are reported very rich, especially that which crosses the river at Mochuca. The ore found near the mouth of the San Carlos is auriferous pyrites. No gold has yet been found in the San Juan district although it has been carefully searched for. In the Segovia district the mines are richer, but the roads are worse, making the introduction of machinery difficult, and for this reason no gold veins are worked that yield less than 1 oz. per ton. All the hills and most all the rivers in this department contain gold and silver, but nearly the entire region is undeveloped, owing to the want of capital. The silver and other mines are not worked, as more money can be made at gold mining, but in the days of Spanish possession these too were worked with great profit. At present the mines of Macueslizo and Dipilto are being worked.

In the district of Jicaro are the gold mines Pacli, El Tirado, San Albino, Quilali, Los Encinos, Las Vueltas, and 14 others in operation with ancient machinery. In the Murra district there are 4 mines working with 2 machines; and in the San Fernando district, in the San Juan Mountains, which includes Pericon, there are 4 mining machines and more than 10 mines. All produce ore in abundance, carrying 1, 2, and up to 3 ozs. of gold of 16 to 19 carats per ton. In the districts of Telponeca, San Juan, and El Pericon there are 12 mines, but they are not worked for want of capital, and in the district of Cuje some are of extraordinary richness, but there is a scarcity of water. Water could be brought, however, in pipes or flumes at no great cost. In nearly all these mines the imperfect and ancient arastras are used, and only about 2 tons of rock are ground in 24 hours.

Leaving out the district of Cuje, of which no statistics are available, the monthly output of the New Segovia mines is 900 ozs. The principal veins of gold are found in the mountains that form the Nicaraguan-Honduras boundary. The upper part of the river Coco, which takes its rise in Segovia, surpasses all other placers in the republic in richness.

La Ciudad Vieja owed its prosperity to the gold washed in its vicinity. The silver mine Limon, now flooded with water, was in ancient times very rich, containing veins that yielded 1 oz. silver per lb. of ore. An official document published by the prefect of the department, October 4, 1850, stated that there were within a radius of 12½ leagues from the capital more than 40 mines of gold and silver. Those of San Marcos, Mocuelizo (Las Animas, San Jose, Santa Rosa), and Dipilto (Santa Ana, La Gloria), situated near the Honduras line, are still

worked at a profit, and at Gracias in Honduras there is a very great mineral deposit. The mines of El Potosi and El Corpus, situated in Nicaragua near the Honduranian boundary, yielded so much gold in olden times that the Spanish government established there a special office to collect the taxes imposed on them.

All along the northern front of the republic there is an abundance of argentiferous galena, and copper, iron, zinc, antimony, and tin ores are also found. Cinnabar occurs at various points in Chontalas between the Sierra and the lake. In the mountains and Dipilto, opals as good as those of Honduras, and emeralds have been discovered.

Matagalpa District.—This district lies between Chontalas (La Libertad), Seguia, and east of the department of Leon, and contains numerous mines, some of which are very rich. In the eastern part, in the Canton of San Ramon, are the Monte Grande, La Leonesa, Ojoche Quijano, San Pablo, and Ucalca Mines, and two at Muy Muy Viejo, which are recent discoveries, and are reported to be yielding 8 ozs. gold per ton of ore. To the west, mines are scattered all along the Cordilleras. Those of Jicaro, near Trinidad, have been long celebrated, as also are those of Santa Rosa (El Pilar), Achuapa, San Francisco, and Sebaco. All through this district other minerals are abundant. For example, the natives obtain lead by a rude smelting process, and iron also. Large quantities of blue and green carbonate of copper are found.

Departments of Leon and Chinandega.—In Leon there is a diversity of minerals, but only three gold mines have come to my notice, viz., La Palmita, Esquipulas, and Santa Rosa, the first being worked by an English company and the others by natives. In Chinandega there are two mines, one at Villanueva, and the other called San Francisco, both in the northern part.

In various places in the republic sulphur is found; also marble, lime, alabaster, gypsum, alum, saltpeter, chrome ore, and at San Lorenzo, Chontales, lithographic stone. Specimens of hard coal from the vicinity of San Ubaldo, on the lake, have been shown.

Altogether there are probably over 600 mines in the little republic, of which only about a dozen are now being worked, and there is apparently a great opportunity here for scientific mining. The native miners, however, are bad and unskillful workmen, being averse to the use of explosives; nearly all the ore is at present extracted by bar and pick and carried to the surface in *tenates* (leather bags) by men.

COSTA RICA.

This republic adjoins Nicaragua on the south. Its average length north and south is 230 miles, and its breadth east and west 100 miles, its area being about 23,000 sq. miles. The first mines of precious metals were discovered in 1815 by the Spanish Bishop Garcia, in the Aguacate Mountains, and they have been worked ever since. There are five denunciations (each containing several mines) now being worked, all of which are rich. Besides these mines nearly all the rivers of Costa Rica run over auriferous sands, especially those of the Duy Valley, one of them being the celebrated Rio de la Estrella.

In the Province of Guanacaste there are the copper mines of Chapermal, Panama, Punta Blanca, Guacemalla, Cano Travesada, and Corazon de Maria, the

farthest being 10 miles from the Pacific coast and several being close to the shore, while all are within 10 miles radius of Sardinal. Also in the same province, on the Nicoya Peninsula, there is a gold mine between the rivers De Oro and Coyote about 2½ miles from the shore. In the Province of Alajuela there are the gold mines of San Gerardo Pena Grande, and a mine about 2½ miles north of Santa Clara. About 9 miles south of San Jose there is a coal mine on the Rio Tiriqui, and a gold and copper mine 6 miles west of it, near Santa Ana, on the same river.

In the Province of Talamanca, now called the Comarca of Limon, there are rich gold mines which have been known since the time of the Spaniards. There are also deposits of copper, magnetic iron, hard coal, and fountains of petroleum.

There are gold mines also at the headwaters of the river Parrita Grande, in the Dota Mountains, 12½ miles south of Cartago, and also one, the Rio Macho, in the Sierras de las Cruces, 10 miles southeast of Orosi. There is gold and silver on the river Zent near the headwaters of the river Toro, 17½ miles southwest of Port Limon, and 13 miles south of Port Limon are some very old mines, situated near the headwaters of the Rio del Norte or Estrella, and 4½ miles from the shore of the Caribbean Sea, which are supposed to be ancient Spanish workings. Nine and a half miles further down the coast, in a bend of the river Toba and 2 miles from the shore, are mines of gold and silver. In the extreme southern part of the State, near the head of the Gulf of Dulce on the river Esquinas, 7 miles from the gulf, are other gold mines.

The mines of the Monte del Aguacate, from which $15,000 in gold has been taken in one day, and where bonanzas have been found yielding $37,500 per ton of ore, are 16.77 miles distant from the port of Puntarenas and 21.12 miles from San Jose, the capital.

In 1892 it was stated by the minister of Nicaragua at Washington, the Hon. David J. Guzman, that $7,000,000 in gold and silver had been taken from Costa Rican mines.

Other Minerals.—At Santa Rita are found, besides gold and silver, lead, terra alba, and chalk, and in other places in the republic, kaolin, lime, sulphur, fuller's earth, and various iron ores. In Santa Cruz, Province of Guanacaste, occur argentiferous zinc ores and carbonate of copper. In the Province of Alajuela, cinnabar and plumbago. In Puriscol, zinc ore and anthracite coal. In the volcanoes of Poaz and Miravalles, sulphur. In Atenas, lignite and opals. In Catalina and at Cabo Blanco, white, black, and green marbles, and alabaster. In San Sebastian, serpentine marble.

Copper is found at Cervantes, 6½ miles east of Cartago, and coal 6 miles further at Naranjo, both being quite near the national road to Port Limon. All these deposits occur in the more populated part of the country. In the southern part there is an area of 70 by 90 miles that is almost a terra incognita, and only one or two deposits have been reported in it, and those on the immediate borders. The entire region is covered with such an exuberant growth of vegetation that it is not strange that so few of its hidden treasures have been found.

NORWAY.*

THE state silver works at Kongsberg produced, during the fiscal year 1895–96, 5112 kgms. of silver, an increase of 160 kgms. from the previous year. Included in this is the output of the new extraction works opened in March, 1895, which during the 17 months of its activity worked 6270 tons of tailings and 750 tons of slime, yielding 561 kgms. of silver in the form of sulphide. The working expenses were 32,000 kronen, and the receipts 44,000 kronen.

The outlook for the profitable working of the mines which were in operation in the eighties is unfortunately very poor, while most of those which put out the larger part of the silver from 1830 to 1880 have now become unproductive, although the mine called "God's Help in Need," which earned this appellation in the last century, has again shown rich ore deposits.

The gold deposits at Bömmelöen, on the west coast of Norway, were worked during the summer of 1896 by the Bremsnäs Gold Company, Limited, with a force of 25 men, which was later increased to 80.

At Sande, in Ryfylke, some zinc blende was produced in 1896.

During the decline in the value of nickel in 1896, the works at Ringerike confined itself to the smelting of old stocks of regulus. Also, the cobalt works at Modum operated on a limited scale, employing 36 men.

Of the iron works which were formerly so numerous in Norway, only that at Näs near Tvedestrand was still in activity; it employed 100 men and produced principally cast steel. Some iron ore was exported from the Rödsand mine in Nordmöre and from the Grönöen mine on the island of Hitteren. Some titaniferous iron ore from Saggendal was exported to Hungary (probably for the lining of smelting furnaces).

The copper works and pyrites mines were worked under decidedly improved conditions, owing to the more favorable prices and the increased consumption. Those at Röros, with 600 men, produced 636 tons of refined copper, and 10,000 tons of pyrites for export. Last autumn the great electrical plant was completed, whereby 600 ergs from the Kuraa Falls will be utilized in the mines. Through the technical reforms which are made possible thereby, a reduction of 40% in the running expenses of the mines is counted upon. It is proposed to utilize the above-mentioned power-transmission in substantially enlarging the working of the mines. For the current year (1897) a production of 17,800 tons of pyrites for smelting (formerly 6000 to 7000 tons) and 26,000 tons for export is counted upon.

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* Translated from the *Chemiker Zeitung*, No. 12, 1897.

The Killingdal iron pyrites mine, a little to the north of Röros, produced in 1896, with 70 to 80 men, 6000 tons of pyrites. It is the purpose here to arrive at an output of about 20,000 tons. The copper mines at Sulitelma produced in the fiscal year (November 1, 1895, to October 31, 1896), working with 500 to 600 men, 23,803 tons of pyrites. In all 26,908 tons were exported. Besides this, 10,826 tons were produced, from which 8566 tons were dressed and 1083 tons of matte, assaying 35% copper, were obtained. There were 713 tons of matte exported. The pyrites exported contained 45% sulphur and 3.75 to 4.50% copper. Of that smelted, the first class contained 7 to 8% copper, and the second class 4 to 5%. The outlook for these mines is very promising, since several new and rich deposits have been discovered. Although heretofore only copper matte has been produced at the smeltery, it is purposed to erect a Bessemer plant and later a refining furnace. Already there has been put up near the Balmi Falls a larger turbine to furnish 500 ergs for the Bessemer plant, electric lighting, and transmission of power to the mines.

The Bosmo iron pyrites mine in Ranen, with a working force of 60 to 80 men, in 1896 produced 18,000 tons of dressed iron pyrites, containing 49 to 50% sulphur, and free from copper and arsenic, together with several hundred tons of lump pyrites. The copper works in Alten, the most northerly situated copper works of the world, which have lain almost completely inactive during the past 20 years, have now resumed operations provisionally, by way of experiment, but the working will probably be conducted more extensively in 1897. In Southern Norway the Aamdal Works in Telemarken produced 1520 tons of pyrites with 20.5% copper. The total number of workmen in the Norwegian pyrites and copper mines was 1650 in 1896, with a production of 70,000 tons of pyrites for export, containing 2600 to 2700 tons of copper.

One of the most important events in the mining industry of Norway in 1896 was the borings for coal on the island of Andöen in Vesteraalen. The occurrence there of thick and extensive strata of gas-coal has been confirmed, and the discovery is attended with encouraging hopes for the future.

The exports of hewn granite rocks (granite, labrador, etc.) was in 1896 about the same as in the preceding year, *i.e.*, 55,000 tons. The greater part of this goes to Germany and England. Inland consumption is understood to have undergone a steady increase. Prices were, however, a little weak on account of the sharp inland competition.

The Anker marble works produced in addition to labrador (Augite-syenite), from the vicinity of Larvik, 1900 cu. meters of white marble from the extensive quarry in Fauske, and also 200 cu. meters of black marble from the quarries at Brevig. For 1897 a production of at least 3000 cu. meters is expected. The prices of feldspar are about the same as in 1895, namely 15 to 16 kronen for first quality, and 8 to 9 kronen for second quality f. o. b. The exports were about 10,000 tons. Of quartz a few thousand tons were exported, also some feldspar from the vicinity of Kongsberg and from Telemarken. From Risör 30 tons of rutile were exported, and smaller quantities were exported from some other places. The market price of this mineral rose to about 60 oere per kgm. The working of the apatite quarries at Oedegaarden in Bamble was very limited.

MINERAL PRODUCTION OF NORWAY. (*a*) (IN METRIC TONS AND DOLLARS; 1 krone = 27 cents.)

Year.	Apatite.		Cobalt Ore.		Copper Ore.		Feldspar.		Iron Ore.	
1890.....	11,119	$260,182	213	$19,500	18,769	$186,680	11,850	$55,458	1,300	$2,340
1891.....	4,258	54,120	187	13,000	20,939	185,120	12,257	55,900	1,464	2,600
1892.....	2,427	44,200	123	8,580	18,888	134,940	5,936	27,820	860	1,430
1893.....	1,513	26,460	123	12,150	21,907	175,230	3,506	17,010	800	1,485
1894.....	2,086	36,720	89	8,100	20,226	192,240	7,836	35,910	.....	.....

Year.	Nickel Ore.		Pyrites, Iron and Copper.		Silver Ore.		Zinc and Lead Ore.		Cobalt.	
1890.....	8,181	$48,360	58,669	$262,600	1,311	$161,460	3,941	$35,620	2.60	$7,020
1891.....	12,839	32,630	49,048	209,560	1,277	149,890	498	4,550	7.00	18,200
1892.....	6,959	18,850	52,570	200,720	1,121	127,400	576	5,200	4.50	13,000
1893.....	2,397	6,480	58,570	190,080	890	117,045	.....	.....	.....	.....
1894.....	2,355	5,400	53,754	268,110	750	100,845	200	1,690	.....	.....

Year.	Copper.		Iron, Pig and Castings.		Iron, Bar and Steel.		Nickel.		Silver. (*b*)	
1890.....	466	$120,900	517	$6,708	691	$38,480	71	$45,500	5.08	$161,876
1891.....	677	174,980	421	5,252	593	35,776	135	96,980	4.68	145,600
1892.....	631	138,060	610	6,760	488	24,258	97	52,780	4.81	125,840
1893.....	.....	.....	.....	.....	.....	.....	.....	.....	4.80	.....
1894.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

(*a*) From returns furnished by the Central Statistical Office, Christiania, for the *Annual General Reports Upon the Mineral Industry of the United Kingdom of Great Britain and Ireland*
(*b*) Contained 22,549 kgms. of gold in 1890, 11,000 kgms. in 1891, and 16,177 kgms. in 1892.

MINERAL IMPORTS OF NORWAY. (*a*) (IN METRIC TONS.)

Year.	Cement	Coal.	Copper, Sheet.	Iron.						Potassium, Nitrate.	Salt.	Zinc, Sheet.
				Bar.	M'f'es.	Pig.	Plates.	Rails.	Steel.			
1889.....	11,928	819,512	1,040	26,731	10,566	19,080	10,289	3,324	2,074	156	158,586	707
1890.....	14,184	766,995	1,477	17,531	11,924	18,227	9,034	4,330	2,184	320	122,280	702
1891.....	14,372	891,453	1,652	27,271	14,514	22,255	10,590	3,673	1,738	320	126,302	921
1892.....	16,381	903,319	1,478	24,936	15,674	15,206	7,658	8,364	1,383	197	140,401	901
1893.....	14,694	900,154	1,331	22,974	17,722	17,070	10,121	8,482	1,897	290	162,978	1,027

MINERAL EXPORTS OF NORWAY. (*a*) (IN METRIC TONS.)

Year.	Apatite	Building Stone.	Chrome Ore.	Copper Ore.	Feldspar.	Iron Ore.	Limestone.	Nickel Ore.	Iron Pyrites.	Silver Ore.	Whetstones.	Zinc Ore.
1889.....	10,665	43,163	310	4,823	10,995	1,143	483	270	63,217	348	78	2,875
1890.....	11,119	54,623	100	1,233	11,850	370	555	.....	54,851	368	149	3,444
1891.....	4,258	41,345	90	1,649	12,257	752	388	40	45,689	304	146	1,102
1892.....	2,427	45,598	21	1,845	5,936	69	769	.....	47,748	474	129	629
1893.....	1,513	48,691	30	11,874	3,506	510	857	340	52,757	154	109	.....

EXPORTS OF METALS FROM NORWAY. (*a*) (IN METRIC TONS.)

Year.	Cobalt.	Copper.	Nickel.	Iron.					
				Bar.	Castings.	Plates.	Nails.	Pig and Scrap.	Steel.
1889.....	76	310	181	140	174	10	8,578	2,013	120
1890.....	96	557	171	23	224	1	9,788	3,610	178
1891.....	44	597	151	118	173	4	10,019	2,091	154
1892.....	.....	350	56	155	185	17	8,916	2,980	119
1893.....	35	378	177	62	193	2	8,559	4,767	104

(*a*) From *Tabeller vedkommende Norges Bergverksdrift*, Christiania, and from *Meddelser fra det statistiske Centralbureau.*

RUSSIA.

THE present is witnessing a great development in the mineral industry in Russia, whose resources are still largely virgin and are being rendered exploitable by the construction of new railway lines, especially the Transiberian. During the past year a large amount of foreign capital was invested in the empire, chiefly French and Belgian, although there was a considerable amount of German. English capitalists have not yet come to look favorably on Russian investments, probably owing to the great field offered in their own colonies.

Among the new enterprises undertaken in Russia in 1896 may be mentioned the purchase by the Dortmund Metallurgical Company of an extensive tract of land at Ekaterinoslaw, where it proposes to erect works for the fabrication of a large number of metallurgical products. The firm of Krupp, of Essen, purchased at Nordecksof, in the immediate vicinity of Riga, a large piece of land for the location of iron- and steel-works.

In Poland the exploitation of zinc and coal mines and the operation of zinc-works is about to be undertaken on a large scale by a new French company, which has an Imperial concession for a long term of years.

In the Urals and in the Transcaspian great undertakings for the development of the natural resources have been begun. The following companies have received exploration rights: 1. The Chemical Gold Production Co., for winning gold, silver, and platinum in the Urals and other parts of the empire, capital 2,500,000 rubles gold; 2. The Transcaspian Company, "Damurgan," with a capital of 2,300,000 rubles gold, for the exploitation of ores other than those of the precious metals, erection of iron-rolling mills, foundries, etc., in the Transcaspian; 3. The Transural Mining Co., with a capital of 3,000,000 rubles gold, for working gold, silver, platinum, copper, and other deposits in the Perm government of the Wsewolodblagodatsch district.

The American consul at St. Petersburg reports on the increase in local industry as the construction of the Siberian Railway progresses and Siberia becomes connected with the rest of Russia. Naturally the first main influence of the line is on the industries connected with the construction of the line itself, such as iron- and coal-mining. Immense iron-works have been established at Werchoturia, with a capital of 18,000,000 rubles, which must be of great importance to the mining industry of the East. The founders are English, and they

intend to construct a branch from their works to the main Siberian Railway. Coal-mining is still in its infancy, and coal is more expensive than wood for fuel. Large mines exist at Tomsk, which supply some of the factories in the neighborhood. As the mining industry becomes more developed, however, coal will replace wood.

The latest official statistics of the mineral production of the Russian Empire are those for 1894. Unofficial statistics of the production of gold, silver, iron, steel, coal, copper, manganese ore, and zinc in 1895 and 1896 are to be found under the respective captions.

MINERAL PRODUCTION OF RUSSIA. (*a*) (IN METRIC TONS AND DOLLARS (*b*).)

Year.	Asbestos.		Asphaltic Mastic.		Chrome Iron Ore.		Cobalt Glance.		Coal.		Copper.		Glauber Salt.		Graphite.	
1890	....	....	....	.....	2,370	.....	15	.....	6,014,793	$10,281,760	5,725	$2,236,800	....	....	....	.....
1891	1,184	$8,000	14,039	$108,000	3,097	$37,800	10	.....	6,233,025	7,901,480	5,456	1,542,000	3,932	$38,400	164	$8,000
1892	1,239	8,000	15,442	118,400	3,007	7,509	20	.....	6,945,994	8,416,720	5,315	1,476,427	2,481	18,099	82	4,000
1893	1,059	6,836	14,925	114,800	14,585	35,101	3	.....	7,613,724	9,225,200	5,463	1,586,526	5,039	36,754	311	19,200
1894	570	3,400	16,054	176,422	7,537	18,760	9	.....	8,762,340	11,473,600	5,409	1,320,800	4,014	28,800	....	.....
1895	.....	....	.....	.....	.....	.....	....	.....	.....	.....	.....	.....	.....	.....	....	.....

Year.	Gold—Kgms.		Iron, Cast.		Iron Pyrites.		Lead.		Manganese Ore.		Naphtha.		Phosphorites.	
														$
1890	39,369	$24,322,320	.....	.....	....	....	837	$40,8 0	182,468	$310,320	3,979,383	$7,774,160	.....	.....
1891	39,083	17,980,000	1,004,746	$16,710,800	18,353	$44,800	558	30,680	113,081	198,800	4,756,417	3,190,280	17,497	49,200
1892	42,997	20,921,600	1,071,772	18,357,597	14,109	45,200	883	49,160	203,436	370,800	4,896,179	1,990,686	23,238	40,445
1893	44,865	21,869,708	1,148,902	19,100,000	16,271	52,200	844	46,920	268,621	669,100	5,790,692	2,781,851	13,706	36,994
1894	42,932	20,895,200	1,332,465	22,200,000	19,801	48,000	743	36,000	243,469	490,800	5,191,552	3,916,800	14,072	31,680
1895	.....	.....	.....	.....	....	.....	....	.....	.....	.....	.....	.....	.....	.....

Year.	Platinum—Kgms.		Porcelain Clay.		Quicksilver.		Salt.		Silver—Kgms.		Sulphur.		Tin.		Zinc.	
1890	2,834	$553,600	.....	.....	292	$428,000	1,389,961	$2,715,440	16,560	$738,720	161	$4,400	13	$6,400	3,774	$442,400
1891	4,242	776,800	5,202	$34,320	325	356,000	1,351,186	2,586,280	13,726	350,400	336	4,120	9	3,440	3,676	594,800
1892	4,575	558,400	4,199	16,874	343	376,000	1,458,508	2,404,000	11,187	340,000	403	4,920	9	3,520	4,369	446,000
1893	5,094	917,231	3,197	12,848	201	220,484	1,351,056	2,224,234	9,500	276,696	589	7,200	7	2,501	4,501	428,604
1894	5,209	890,400	7,742	30,400	196	153,200	1,354,218	1,723,200	7,513	227,200	1	18	4	1,440	5,014	367,336
1895	.....	.....	.....	.....	....	.....	.....	.....	.....	.....	....	.....	....	.....	.....	.....

(*a*) The figures for 1890 are from the "Industries of Russia; Mining and Metallurgy," vol. IV., St. Petersburg; for 1891–94 inclusive from the Russian official report "Sbornik statisticheskekh Svedenie o Gornozavodskoĭ Promyshlennostye Rossie v zavodskom Godu," St. Petersburg.

(*b*) In the Russian reports the quantities are stated in poods and the values in 1890 in gold rubles and in the succeeding years in silver rubles. In making the reductions it was assumed: 1 pood = .01638 metric ton, 1 gold ruble = 80 cents, 1 silver ruble = 40 cents.

SPAIN.

THE production of the principal ores in Spain, namely, lead, copper, zinc, iron, and manganese, in 1896 is discussed under the respective captions in the preceding part of this book. Of the less important mineral substances there were produced in the province of Gerona 105 tons of barytes and 592 of steatite. Three tons of fluorspar were produced in Barcelona. The salines of Cardona (Barcelona) produced 4270 tons of salt in 1896, and Dr. Oriol estimates the total production of salt in Spain at 350,000 tons.

MINERAL PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Alum Ore.		Antimony Ore.		Asphalt Rock.		Clay.		Coal.		Cobalt Ore.		Fluorspar.	
1891	340	$1,700	693	$20,805	248	$523	180	$360	1,262,510	$2,525,020	60	$1,804	3	$54
1892	550	2,750	395	12,467	503	1,051	132	264	*b* 1,611,761	3,835,676	.....	.....	2	36
1893	650	3,250	88	2,784	820	1,280	60	30	1,484,794	2,280,720	18	194	56	272
1894	310	1,549	15	300	985	2,009	.....	.....	1,659,274	2,327,677	52	624	18	230
1895	240	1,198	44	700	790	1,581	.....	.....	1,739,085	2,648,367	.....	.....	27	405
1896	.....	.....	.....	200	250	.....	.....	.....	1,883,699	2,750,548	.....	.....	.....	.....

Year.	Copper Ore.						Graphite.		Iron Ore.		Iron Pyrites.		Kaolin.	
	Argentiferous.		Copper Pyrites.		Copper and Cobalt.									
1891	.....	.....	3,495,761	$13,285,892	3,444	$61,361	63	$706	5,601,558	$6,721,868	283,721	$922,662	1,262	$13,882
1892	.....	.....	2,731,593	13,111,649	2,086	50,064	34	249	5,236,275	6,283,531	232,162	754,993	500	5,000
1893	15,219	$27,901	2,144,908	2,151,603	1,116	26,784	Nil.	.....	5,419,071	4,056,546	220,000	110,000	1,502	5,397
1894	.....	.....	2,445,241	2,638,819	853	20,472	10	30	5,352,353	3,995,912	60,000	30,000	247	1,896
1895	.....	.....	2,701,661	2,829,422	410	6,560	.....	.....	5,514,339	4,183,059	60,267	30,335	836	1,579
1896	.....	.....	2,800,000	2,900,000	.....	.....	.....	.....	6,808,000	5,500,000	200,000	100,000	.....	.....

Year.	Lead Ore.		Lead Ore, Argentiferous.		Lignite.		Manganese Ore.		Mercury Ore.		Phosphorites.		Salt.	
1891	160,418	$4,491,712	264,625	$8,071,065	27,188	$103,500	6,993	$6,973	35,169	$1,749,583	1,542	$3,084	582,833	$1,748,498
1892	215,906	5,613,558	192,517	5,852,518	*c*32,870	41,127	16,910	136,171	35,025	1,755,538	2,150	3,152	682,634	2,596,741
1893	169,707	3,263,648	179,458	4,139,695	35,315	41,967	1,460	7,666	34,309	1,628,188	211	415	151,464	85,052
1894	140,842	2,208,839	181,715	5,497,316	48,460	63,266	340	522	30,030	1,200,069	40	80	186,121	228,374
1895	124,195	2,052,312	181,433	5,042,288	44,708	58,333	10,162	17,043	33,792	1,248,815	1,040	2,081	326,320	952,096
1896	120,000	2,000,000	200,000	5,560,000	44,700	58,000	92,000	150,000	35,106	1,350,000	.....	.....	300,000	900,000

Year.	Silver Ore.		Steatite.		Sulphate of Barium.		Sulphate of Soda.		Sulphur, Crude Rock.		Tin Ore.		Topaz of Hinojosa—Kgms.		Zinc Ore.	
1891	15,085	$62,189	2,510	$10,564	670	$5,360	911	$4,012	30,577	$79,513	69	$12,456	1,900	$27,654	78,216	$651,180
1892	656	113,543	1,341	6,576	665	4,658	350	1,540	26,330	68,458	244	42,347	280	3,173	74,265	530,555
1893	4,825	257,041	4,010	18,705	643	2,943	180	270	24,793	59,830	34	3,611	81	1,874	62,616	387,101
1894	7,411	237,995	3,109	13,603	541	2,696	312	468	10,865	28,248	26	2,678	75	1,549	58,760	395,790
1895	16,299	75,836	2,347	6,219	494	2,255	460	690	8,481	22,049	17	1,500	67	1,385	54,109	370,007
1896	35,000	640,000	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	45,000	300,000

(*a*) From the official *Reports of the Comision Ejecutiva de Estadística Minera*, Madrid, 1891 to 1894, inclusive. The figures for 1895 are from the official *Report of the Junta Superior Facultativa de Minas*, Madrid, 1896. The figures for 1896 were furnished by Señor Don Roman Oriol as published in the *Revista Minera*.
(*b*) Including lignite. (*c*) Fiscal year 1891-92 ending July 1.

METALLURGICAL PRODUCTION OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1=5 pesetas.)

Year.	Arsenic Sulphide.		Refined Asphalt.		Hydraulic Cement.		Coke. (*b*)		Copper. (*c*)		Iron, Pig.	
1890 ...	60	$7,541	23	$188	135,988	$388,109	191,235	$772,975	64,373	$9,542,469	170,782	$2,732,634
1891 ...	58	4,753	192	2,531	150,721	423,355	304,030	1,257,407	58,067	8,345,389	159,250	2,002,075
1892 ...	72	4,165	845	10,985	114,428	296,598	354,644	1,493,284	81,343	11,240,821	(*d*)259,575	7,260,461
1893 ...	129	8,262	580	7,540	140,314	356,071	390,110	1,443,544	45,409	4,893,719	134,563	1,878,728
1894 ...	184	13,224	905	11,761	132,645	376,204	359,682	1,600,259	47,204	5,305,929	123,796	1,699,682
1895....	184	18,390	790	10,275	149,197	324,214	692,043	2,417,019	37,488	4,786,794	179,752	2,575,065

Year.	Iron, Wrought.		Steel.		Lead. (*f*)		Lead, Argentiferous.		Mercury.		Silver.—Kgms.	
1890....	61,445	$2,949,355	75,255	$3,100,491	91,628	$5,611,855	86,325	$8,977,779	1,813	$2,239,547	52,000	$1,770,496
1891...	64,799	3,110,347	69,902	2,804,468	82,463	5,050,034	76,622	8,427,776	1,790	1,840,108	50,051	1,704,187
1892....	(*e*)	.....	(*e*)	.....	125,550	6,981,048	93,710	7,297,619	1,657	1,607,579	72,000	2,339,817
1893 ..	58,923	2,441,290	76,583	3,048,669	77,486	3,886,695	91,832	7,138,834	1,666	1,394,630	63,605	1,774,671
1894 ..	54,214	2,209,878	92,854	3,584,400	64,189	2,910,453	88,433	7,385,797	1,610	1,286,961	192,745	4,195,184
1895....	48,462	2,053,822	56,801	2,189,222	76,808	3,637,932	83,978	6,916,320	1,506	1,343,417	58,546	1,360,996

Year.	Sulphate of Soda.		Sulphur.		Zinc.					
					Ingot.		Sheet.		Calamine, Calcined.	
1890....	128	$1,536	9,855	$221,737	3,103	$406,780	2,816	$495,040	35,348	$416,441
1891....	538	6,456	9,900	222,750	2,803	367,154	2,789	490,899	31,379	363,996
1892 ...	206	2,204	7,822	132,982	(*g*) 5,908	.....	(*h*)	... .	28,118	(*j*)1,211,304
1893....	247	3,681	4,686	82,600	3,290	405,612	2,462	433,418	(*i*) 21,548	13,963
1894 ...	174	3,059	3,417	58,086	3,052	360,276	2,421	421,354	17,768	11,514
1895 ...	227	2,946	2,231	37,930	3,149	302,266	2,487	373,125	.....	.....

(*a*) From the *Reports of the Comision Ejecutiva de Estadística Minera*, Madrid; for 1895, from preliminary official figures of production as published in the *Revista Minera, Metalurgica y de Ingeneria*, Madrid.

(*b*) Including briquettes, which were in 1890, 174,524 metric tons, value $698,096; 1891, 258,254 tons, value $1,036,192; 1893, 273,118 tons, value $1,117,672; 1894, 209,776 tons, value $852,231; in 1895, 342,985 tons, value $1,370,987.

(*c*) Including besides fine copper, precipitate and matte. The precipitate amounted in 1890 to 40,012 metric tons, value $7,402,140; in 1891, 33,401 tons, value $6,168,502; in 1893, 26,404 tons, value $3,696,628; in 1894, 29,548 tons, value $4,136,687; and in 1895 31,725 tons, value $4 439,889. The matte in 1890 amounted to 24,303 metric tons, value $2,124,673; in 1891 to 24,444 tons, value $2,116,784; in 1893 to 18,899 tons, value $1,175,956; in 1894 to 16,872 tons, value $1,012,295; and in 1895, 5,756 tons, value $345,385. The amount of the fine copper was in 1890, 58 metric tons, value $15,665; in 1891, 223 tons, value $60,102; in 1893, 106 tons, value $21,134; in 1894, 785 tons, value $156,947; and in 1895, 7 tons, value $1,520. These three products were not stated separately in 1892.

(*d*) Including pig iron, wrought iron, and steel.
(*e*) Included under iron, pig, 1892.
(*f*) In 1892 contains 99 tons antimonial lead, value $5963.
(*g*) Including ingot and sheet.
(*h*) Included under ingot, 1892.
(*i*) Estimated for the second half of the year.
(*j*) Total value for ingots, sheet, and ore.

Additional substances in 1894: Iron wire, 3728 metric tons, value $235,555; salt, 12,988 tons, value $94,444; in 1895, iron wire 3980 tons, value $258,441.

MINERAL IMPORTS OF SPAIN. (*a*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Asphalt and Pitch.		Alkaline, Carbonates.		Coal.		Coke.		Gold Bars. Kgms.		Gold Coin.
1892. ..	25,682	$513,635	23,213	$1,109,393	1,688,637	$9,020,169	175,872	$940,983	7,738	$4,797,560	$3,936
1893....	33,608	672,164	25,649	1,128,571	1,497,699	8,014,112	267,288	1,410,045	2,203	1,365,860	1,008
1894....	29,639	592,799	26,902	1,143,708	1,612,147	9,028,019	225,900	1.265,046	.....	.....	634.394
1895 ..	31,921	558,748	27,128	1,193,644	1,505,541	8,129,920	219,643	1,186,073	...	.....	119,945
1896....	35,401	566,427	27,119	1,193,256	1,447,345	7,815,663	234,033	1,263,778	337	235,900	16,231

Year.	Iron, Pig.		Iron, Bar.		Machinery.(*b*)		Petroleum, Crude.		Silver Bars. Kgms.		Silver Coin.	Soda Nitrate.	
1892	30,022	$420,312	10,177	$478,206	6,297	$1,609,381	44,656	$1,786,241	97,468	$3,898,720	$1.000,719	18,335	$1,099,917
1893	23,484	328,784	7,139	343,303	2,795	649,686	54,942	1,977,925	394	14,184	3.827,308	18,998	1,139,923
1894	26,561	373,885	9,616	417,342	3,134	698,102	44,052	1,263,514	1,935	61,920	4,339,319	23,984	1,439,060
1895	12,384	152,188	7,768	392,370	3,629	871,160	47,374	1,515,971	4,425	115,050	4,805,670	26.459	1,587,571
1896	8,607	122,306	14,832	579,022	3,639	873,620	34,427	1,101,676	5,776	110,812	20,190,398	26,694	1,599,890

Year.	Steel.		Sulphur.		Tin Plate.		Tin Ingots.	
1892	31,637	$1,214,953	8,568	$222,759	2,433	$293,341	813	$406,580
1893	15,143	717,841	6,330	162,186	2,986	268,794	1,004	395,362
1894	23,121	973,998	5,674	124,840	3,212	254,567	936	459,244
1895	18,232	753,742	6,884	192,755	1,240	86,448	825	363,120
1896	26,265	1,387,404	6,426	179,949	969	81,462	1,068	470,317

MINERAL EXPORTS OF SPAIN. (*a*), (*c*) (IN METRIC TONS AND DOLLARS; $1 = 5 pesetas.)

Year.	Antimony Ore.		Copper Ore.		Iron Ore.		Iron Pyrites.		Lead-Silver Ore.		Manganese Ore.		Phosphates	
1892	395	$12,467	511,015	$3,884,478	4,799,647	$8,639,266	443,026	$886,052	10,613	$890,315	10,410	$97 854	1,740	$3,480
1893	87	16,515	574,590	4,391,500	4,646,877	8,564,379	399,880	800,560	12,048	1,016,821	9,480	96,993	..	.....
1894	45	2,127	539,290	4,877,801	4,988,222	8,977,400	520,128	1,040,260	12,162	1,020,766	7,319	68,779	.....	.....
1895	9	582	504,407	4,539,670	5,179,761	9,315,468	488,699	981,419	9,203	598,088	29,997	282,069	..	.....
1896	119	7,190	629,546	5 665,918	6,289,257	11,320,670	464,882	920,977	6,270	392,668	87,835	827,620	.....	.....

Year.	Zinc Ore.		Copper Matte and Precipitate.		Iron, Pig—Soft Iron and Steel.		Lead. (*d*)		Quicksilver.		Zinc. (*e*)	
1892	39,574	$261,191	36,862	$4,768,135	51,975	$1,152,324	154,969	$10,519,924	1,644	$1,840,938	2,161	$216,185
1893	32,358	203,372	18,658	3,496,283	31,230	499,693	158,737	10,756,053	1,560	1,746,845	2,486	248,638
1894	34,238	225,200	36,752	4,213,024	48,749	682,112	158,735	11,091,540	870	974,340	2,680	267,999
1895	29,359	267,957	59,507	4,379,205	22,669	317,371	154,584	8,825,553	1,330	1,489,194	1,370	136,690
1896	36,656	308,272	46,417	3,780,585	23,085	233,270	166,124	9,500,935	1,559	1,746,154	3,560	356,060

(*a*) From the *Revista Minera de España* and figures furnished by Señor Don Roman Oriol, excepting the exports for 1892, which are from the *Report of the Comision Ejecutiva de Estadística Minera*, Madrid.

(*b*) Engines and boilers only.

(*c*) Other exports: Coal in 1892, 14,300 metric tons, $77,706; in 1893, 8387 tons, $45,290; copper matte, in 1892, 30,222 tons, $350,591; in 1893, 28,130 tons, $286,302; gold bullion and coin in 1892, $95,336; in 1893, $114,298; silver bullion and coin in 1892, 2,237,458 kgms., $8,349,832; in 1893, 705,703 kgms., $254,051; salt in 1892, 232,234 metric tons, $700,702; in 1893, 200,924 tons, $602,772.

(*d*) Includes argentiferous lead. (*e*) In 1892, includes zinc in pigs and sheets and calcined calamine.

SWEDEN.

THE mining of iron ore and the production of pig iron and steel are by far the most important branches of the mining and metallurgical industries in Sweden. The statistics for 1896, for which refer to the article on "Iron and Steel" in the first part of this volume, show an increase over 1895.

The Grängesberg mining district has, of late years, attracted increasing attention. The percentage of iron in the ore exported thence varies between 61 and 63%, and is guaranteed not to be less than 60%. About 200 carloads per day are dispatched to Oxelsösund, the shipping port. The following table shows the output of the whole Grängesberg district and the annual shipments for export by the Grängesberg Mining Company in long tons:

Year.	District	Gränges-berg Co.	Year.	District	Gränges-berg Co.	Year.	District	Gränges-berg Co.	Year.	District	Gränges-berg Co.
1885....	41,856	22,376	1891....	223,402	128,528	1893....	284,223	227,444	1895....	489,049	268,547
1890....	173,694	97,644	1892....	268,715	136,900	1894....	423,502	132,422	1896....	.....	400,000

Of the ore shipped by the Grängesberg Company in 1896, 302,386 tons went to Holland, and 78,630 tons to Germany. The total annual output of iron ore in the whole of Sweden has risen to about 2,000,000 tons, and the exports in 1896 amounted to close upon 800,000 tons.

Most of the machinery in the mines is worked by electric power, and the installation at Grängesberg is the first long-distance transmission of electric power in Sweden, the distance from the Hellsjön Lake to Grängesberg being about 7¼ miles. At Hellsjön, where the turbines are placed, there are three dynamos, one of 150 horse power, and two of 100 horse power each, direct-coupled to a turbine making 600 revolutions. The three-phase current, of 150 volts pressure, is subsequently transformed into one of 5500 volts. In addition to these three generators there are two more 100-horse power dynamos, each in connection with a turbine; one supplies a single-phase current for lighting purposes while the other can give either a three or a single-phase current and is held in reserve. There are altogether 12 motors, varying in capacity from 45 horse power downward.

Dynamite is used for blasting, and is manufactured on the spot; the annual production is about 80 tons, of which half is used in the Grängesberg mines. The largest mine is the Skärningen. Hitherto the ore has been put direct on

to the tracks, but it is now broken at a lower level than that of the railway. The ore is separated from the rock in the mine; an electric ore separator is being installed.

The latest official statistics of mineral production in Sweden are those for 1895.

MINERAL PRODUCTION OF SWEDEN. (*a*) (IN METRIC TONS.)

Year.	Alum.	Cement		Coal.	Cobalt Ore	Copper Ore.	Fire-clay.	Gold Ore.	Iron Ore.	Iron Bog Ore.	Iron Pyrites	Manganese Ore.	Nickel Ore.	Plumbago.	Pyrolusite (Powder'd)	Red Ocher	Silver and Lead Ore.	Zinc Ore.
		Value.																
1891	542	$487,858		198,033	244	21,823	134,909	2,680	985,255	2,150	1,659	9,079	483	17	192	1,468	15,044	61,591
1892	346	581,188		199,380	53	24,069	123,096	3,463	1,291,933	1,650	1,249	7,832	.....	15	172	1,090	19,803	54,981
1893	357	507,668		199,933	101	22,033	138,469	2,441	1,481,487	2,275	480	7,061	.....	49	.....	1,370	21,043	46,623
1894	261	.....		213.634	....	25,710	129,617	.....	1,926.523	689	656	3,359	.....	106	.....	1,564	14,825	47.029
1895	286	.....		223,652	....	26,009	120,385	459	1,904,662	.....	221	3.117	.....	7	.....	1,291	12,045	31,349

Year.	Brass.	Cobalt Oxi. Kilos	Copper.	Copperas.	Copper. Hammer'd	Copper, Sulphate	Gold. Kilos.	Iron.			Steel.			M'f's of Iron and Steel.	Lead.	Silver. Kilos.	Sulphur.	Sulphuric Acid.
								Pig.	Bloom.	Bar.	Bessemer.	Siemens-Martin	Other					
1891	293	6,260	543	419	308	612	109.580	490,913	224,651	280,430	92,985	78,197	1,592	72,436	299	3,658	23	18,172
1892	302	7,138	745	476	314	580	87.626	485,664	235,426	273,510	82,422	76,556	1,494	.....	799	5,211	46	28,147
1893	497	3,298	543	454	389	659	93.376	453,421	225,533	266,727	84,389	81,889	1,222	81,549	461	4,464	75	27,324
1894	...	1,580	350	362	.....	722	93.603	462,809	204,517	267,049	83,322	84,003	510	88,942	330	2,870	36	.....
1895	...	.....	216	94	.....	1,195	85.291	462,920	188,726	294,135	97,320	93,259	598	.....	1,256	1,188	..	.....

(*a*) From *Bidrog till Sveriges Officiela Statistik, Bergshandteringen* and *Tullkomitens Betänkande.* There were also produced of allanite 14 tons in 1891; of litharge, 21 tons in 1891. In 1892 there was also an output of 16 tons of cement copper. In 1892 and 1893 were produced also in each year 0.1 ton of molybdenum ore; in 1894, 0.03 ton of antimony ore, and in 1895, 1.5 tons.

SWITZERLAND.

THE mineral and metallurgical production of Switzerland is insignificant, and there are no official statistics to indicate the precise amounts. There is a small production of coal and iron ore, and from the beds of the Alpine streams a little gold has been obtained, although there is no regular exploitation for the precious metal, which exists in insignificant quantity. The most important metallurgical works of Switzerland are those at Neuhausen, where the manufacture of aluminum is carried on upon a large scale. Advantage has been taken of the abundant water-power at the same place to establish works for the manufacture of sodium and potassium salts electrolytically. The exports and imports of various mineral substances from and into Switzerland are given in the following tables:

MINERAL IMPORTS OF SWITZERLAND. (*a*) (IN METRIC TONS; 5f. = $1.)

Year.	Chemical Products	Coal and Coke. (*b*)		Total Iron and Steel, Wrought, Pig, and Manufactures. (*c*)		Iron and Steel, Pig & Wrought.		Iron Manufactures.		Mineral Oil, Crude and Refined.		Machinery and Locomotives.
1890..	$4,407,000	993,612	$6,092,800	154,632	$8,576,600	135,363	$5,253,200	19,268	$3,323,400	40,569	$1,763,800	$3,017,200
1891..	3,982,600	1,060,624	6,616,400	166,870	8,654,800	145,367	5,254,800	21,503	3,400,000	43,756	1,611,400	3,212,000
1892..	4,052,200	985,340	5,827,400	165,771	8,385,000	144,705	4,919,200	21,066	3,465,800	47,956	1,263,200	2,913,000
1893..	4,263,228	1,023,917	6,067,477	190,807	8,038,594	168,814	4,764,933	21,993	3,274,661	51,611	1,154,196	4,570,846
1894..	4,101,313	1,080,739	6,567,951	222,869	8,781,054	192,765	5,309,606	30,104	3,471,448	53,466	1,252,561	4,182,553
1895..	4,268,139	1,186,147	7,224,619	210,925	8,763,999	181,642	5,105,449	29,283	3,658,550	56,420	1,538,561	4,719,776
1896..	4,563,971	1,264,998	7,689,975	278,834	11,459,036	237,377	6,677,186	41,457	3,781,850	62,962	1,717,590	5,393,838

MINERAL EXPORTS OF SWITZERLAND. (*a*) (IN METRIC TONS; 5f. = $1.)

Year.	Chemical Products. (*d*)		Dyes, Coal Tar.		Iron Manufactures.		Machinery and Locomotives.
1890.....	7,591	$548,000	1,338	$1,785,000	3,317	$1,047,200	$4,461,400
1891.....	8,289	553,600	1,541	1,958,200	1,957	855,600	4,063,200
1892.....	9,078	638,900	1,756	2,276,600	2,203	800,600	4,015,600
1893.....	9,848	1,199,057	1,986	2,498,200	2,748	839,639	4,942,621
1894.....	11,500	1,399,005	3,200	2,871,410	2,821	798,160	5,335,273
1895.....	11,928	1,352,313	3,337	3,076,947	2,165	883,581	5,229,591
1896.....	13,423	1,608,797	3,188	2,921,164	2,165	907,456	6,081,755

(*a*) From *British Statistical Abstracts* and from figures furnished by the *Direction Générale des Douanes Fédérales*, Berne. (*b*) Not including briquettes. (*c*) Including iron ores. (*d*) Including spirits for technical use; 1893, 3704 tons, $290,772; 1894, 4146 tons.

IMPORTS AND EXPORTS OF SWITZERLAND IN 1896 OF SUBSTANCES NOT MENTIONED IN THE PRECEDING TABLE. (IN METRIC TONS.)

Substance.	Imports	Exports	Substance.	Imports	Exports
Aluminum, pure—kgms	3,000	661,100	Lead, rolled, etc	1,406	28
Arsenious acid, borax, soda salts, etc.	2,411	86	Lignite	8,399	2,350
Asphalt, etc	2,671	22,300	Lime, hydraulic	27,353	7,108
Briquettes	243,866	111	Mineral and tar oils, not specified	5,592	26
Caustic soda and caustic potash	2,342	198	Nickel, pure or alloyed, rolled or drawn—kgms	139,400	1,100
Cements, Roman and Portland	57,593	834	Paraffine, vaseline, refined benzol, naphthalene, etc	2,137	55
Charcoal	8,399	2,350	Petroleum	57,370	.....
Chloride of lime	1,127	5	Porcelain	799	8
Coal	1,157,435	.....	Salt, cooking, etc	8,578	59
Coke	107,563	3,992	Silver, unwrought	6,858	1,150
Copper and brass, hammered, rolled, etc	3,975	57	Soda, calcined	6,276	3
Copper and brass, in bars and scraps.	1,097	942	Soda, crude and crystallized	1,076	52
Firebrick	10,451	38	Stearine	839	32
Gold, unwrought—kgms	854,500	153,500	Sulphuric acid	4,184	135
Gold, silver, and platinum, rolled, in plates and strips—kgms	1,195,900	136,800	Talc	1,370	88
Gypsum, burnt or ground	15,559	3,034	Tin, in bars, blocks, etc	1,158	8
Iron, crude and old	82,142	6,730	Zinc, in bars, blocks, etc	295	443
Lead, in bars, blocks, etc	2,566	80	Zinc, rolled, drawn, etc	3,075	6

UNITED KINGDOM.

By C. Le Neve Foster.

The object of this article* is to give a general idea of the importance of the mineral industries of the United Kingdom, and to indicate the official sources from which further information is obtainable. The two principal criteria for judging of the extent and importance of the mineral industries of a country are: (1) Number of persons employed; (2) Quantity of mineral produced.

As the owners of mines and quarries are obliged every year by Act of Parliament to supply returns giving these data for their own particular workings, it is easy to compile very correct statistics, which are published annually as "Blue Books" by the Home office, and presented to Parliament.

In this country the words "mine" and "quarry" have certain meanings at law which are not identical with the significations put upon them by the general public; and it is, therefore, necessary at the very outset to point out that the former term applies to *underground workings*, no matter what kind of mineral is being obtained, and the latter to *open workings*. The nature of the excavation decides whether any given working is a "mine" or a "quarry," and not the nature of the mineral. The two kinds of workings are subject to different Acts of Parliament, and are therefore treated separately in the official statistics.

As the principal duty of the Government inspectors of mines is the enforcement of statutes passed for the prevention of accidents, it is natural that the official reports should deal very largely with the casualties to which the miner and the quarryman are subject in the course of their employment. The picture of the mineral industries is thus rendered more complete by giving the death-roll for each year, and by calculating what progress is being made in lessening the risks of the occupations in question.

Following the order adopted in the official reports, I arrange my remarks under the three following heads: (1) Persons employed; (2) production; (3) accidents. In each case I deal with the mines and quarries separately.

I. PERSONS EMPLOYED.

Mines.—The total number of persons employed in and about the mines of the United Kingdom during the year 1895 amounted to 733,657, of whom 584,298 or 79.5% were employed underground, and 149,359, or 20.5%, above ground.

These workers were distributed at the various classes of mines in the following proportions: 687,371, or 93.7%, at coal mines, which include mines producing coal alone or coal together with such other minerals as ironstone, fireclay, etc.; 16,087,

*Based upon my *Second Annual General Report Upon the Mineral Industry of the United Kingdom of Great Britain and Ireland for the Year 1895*, London, 1896.

or 2.2%, were employed at iron ore mines in which no coal was produced; and 30,199, or 4.1%, at all other mines not producing coal or iron ore.

Compared with the previous year these figures show a slight decrease (5719) for coal mines, although the number of mines has increased from 3347 to 3395. There is also a decrease of 361 persons for other mines and an increase of 640 for iron ore mines.

The decreased number of workers at coal mines has not affected the output of coal which is the highest yet recorded.

The employment of females below ground at mines is prohibited by law, but the number employed at the surface in 1895 is shown in the following table:

Description of Mines.	Under Ground.	Above Ground.			Total Number Employed Under and Above Ground.
	Males.	Males.	Females.	Total.	
Coal.....	554,017	128,625	4,729	133,354	687,371
Iron ore.....	12,409	3,570	108	3,678	16,087
Other mines....	17,872	11,528	799	12,327	30,199
Totals.....	584,298	143,723	5,636	149,359	733,657

The average quantity of mineral obtained by each person employed underground in coal mines was 349 tons; in iron ore mines, 632 tons; in other mines, 229 tons.

Quarries.—The passing of the Quarries Act of 1894, which came into operation January 1, 1895, enables figures of persons employed and mineral obtained at all quarries which are more than 20 ft. deep to be given. The fixing of this limit, which takes in between 8000 and 9000 quarries, still excludes many small quarries which are scattered all over the kingdom; and until these are brought under the act the statistical information will be incomplete.

The total number of persons returned as working at quarries in 1895 was 104,625—53,915, or 51.5%, of whom were employed *inside, i.e.*, within the margin of the actual pit-hole or excavation, while 50,710, or 48.5%, were engaged as brick, tile, and cement makers, slate dressers, stone hewers or breakers at works *outside* the pits, but adjacent to or connected with the quarries.

The largest quarries are situated in Carnarvonshire. The Penrhyn slate quarry, which approaches 1000 ft. in depth, employs 3000 men and boys.

The following table shows the persons employed in 1895, arranged according to the minerals worked:

Kind of Mineral Worked.	Number of Persons Employed.		
	Inside.	Outside.	Total.
Chalk.....	2,235	4,922	7,157
Granite.....	4,871	3,362	8,233
Igneous rocks, other than granite.....	4,660	1,483	6,143
Limestone, other than chalk.....	12,654	5,275	17,929
Sandstone.....	13,470	9,068	22,538
Slate.....	6,057	6,506	12,563
Other rocks and minerals.....	9,968	20,094	30,062
Totals.....	53,915	50,710	104,625

II. PRODUCTION.

The total value of the minerals obtained from mines and quarries during the year 1895 amounted to £69,129,664. Coal contributed £57,250,000 and iron ore nearly £3,000,000 of this sum. The tonnage of coal during the year reached the unprecedented figure of 189,661,362 tons. Over 2,000,000 tons of this quantity was anthracite, obtained principally from South Wales. The four largest coal-producing counties are Durham, Lancashire, Yorkshire, and Glamorganshire, each of which has an output of over 20,000,000 tons.

The average price of coal at the pit's mouth during the year was 6s. 7.43d. per ton, the prices ranging from 4s. 6d. to 11s. per ton in the various counties.

Of iron ore 12,615,414 tons were obtained during the year; 41.9% of which came from the North Riding of Yorkshire or Cleveland district.

The production of copper ore was 7531 tons, obtained principally from Cornwall and Devonshire.

The quantity of lead ore obtained in 1895 was 38,412 tons, the counties of Derbyshire, Durham, and Flintshire contributing 48% of this total.

In Cornwall and Devonshire 10,612 tons of tin ore were produced.

The zinc ore figures for the year are 17,478 tons, mainly obtained from the Isle of Man and the counties of Cumberland, Cardigan, Denbigh, and Flint.

III. ACCIDENTS.

The statistics of the United Kingdom show that the death-rate from accidents is not so high among miners as that of persons engaged in engineering works, on railways, or in shipping. In the year 1895 there were 1096 deaths from mine accidents, of which 55, or 5%, were caused by "explosions of fire-damp or coal dust," 447, or 41%, by "falls of ground," 110, or 10%, by accidents "in shafts," 358, or 33%, were "miscellaneous accidents," and 126, or 11%, occurred on the surface.

Explosions of Fire-damp or Coal Dust.—This class of accidents shows a decrease of 262 deaths compared with the year 1894. The worst explosion during the year resulted in the death of 13 persons. The use of naked lights is responsible for more than half the fatal explosions. Since the year 1895 provision has been made by an act of Parliament for dealing with the dangers of shot-firing in dusty mines.

Falls of Ground.—The heavy death-roll of 447 persons is mainly due to a want of "systematic timbering."

Accidents in Shafts.—The breaking of a rope attached to a cage, and the oscillation of buckets by which men were being drawn to the surface, were the principal causes of accidents under this heading.

The deaths caused by the use of man-engines during the year were two. On the whole there are fewer accidents in shafts than most persons would expect.

Miscellaneous Accidents.—The total of deaths has been considerably swelled by serious inundations, that of Audley Colliery in Staffordshire alone resulting in the death of 77 persons. The 79 deaths from accidents on inclined and engine planes suggest the desirability of improvement and whitewashing manholes so as to enable men to see and reach these refuge places more speedily. Accidents by "trams and tubs" resulted in 86 deaths, the heaviest contributor being the Newcastle district. Some of the underground fires were caused by the furnaces used

for ventilation, and such accidents naturally point to non-inflammable materials as being best for lining main shafts. Most of the deaths from accidents at the surface of mines occurred on railways or sidings. The following table gives a comparison of the coal fields in the United Kingdom with regard to persons employed, mineral raised, and deaths from accidents for the years 1894 and 1895:

Coal Field.	1894.			1895.			Death-Rate from Accidents per 1000 Persons Employed.				Number of Deaths per 1,000,000 Tons of Mineral Raised.	
							1895.			1894.		
	Total Number of Persons Employed.	Total Mineral Raised.	Total Number Deaths from Accidents.	Total Number of Persons Employed.	Total Mineral Raised.	Total Number Deaths from Accidents.	Under Ground	Above Ground	Total.	Total.	1895.	1894.
	%	%	%	%	%	%						
1. Scotch coal fields	12.8	11.6	10.1	12.8	15.5	15.0	1.81	1.35	1.73	1.26	5.1	5.0
2. Northern coal field	19.7	22.2	12.7	19.7	20.9	13.9	1.19	0.53	1.05	1.03	3.5	3.3
3. Yorkshire, etc., coal field	22.7	22.0	13.0	22.5	21.2	14.4	0.98	0.84	0.95	0.91	3.6	3.4
4. Lancashire and Cheshire coal field	12.7	12.6	9.0	12.7	11.8	12.3	1.54	1.03	1.45	1.14	5.5	4.2
5. Midland coal fields	9.2	10.1	7.3	9.0	9.7	15.3	3.19	0.45	2.51	1.27	8.3	4.2
6. Small detached coal fields	3.0	2.2	2.0	3.0	2.1	2.5	1.36	0.77	1.25	1.06	6.2	5.3
7. North Wales coal field	1.8	1.7	2.4	1.9	1.5	2.0	1.89	0.46	1.63	2.06	6.9	8.1
8. South Wales coal field	18.0	17.5	43.5	18.3	17.2	24.0	2.05	1.34	1.94	3.87	7.3	14.3
9. Irish coal fields	0.1	0.1	.....	0.1	0.1	0.6	8.44	.....	6.28	.....	46.0	.....
Totals for United Kingdom	100.0	100.0	100.0	100.0	100.0	100.0	1.63	0.87	1.48	1.60	5.3	5.8

The following table will show the accidents which would properly come under the Quarries Act; it will be seen that a very large number of the deaths resulted from falls of ground. In order to guard against possible accidents a code of special rules is in force at many quarries.

Place and Cause.	Number of Separate Fatal Accidents.	Number of Deaths from Accidents.	Number of Non-fatal Accidents.	Number of Persons Injured.	Percentage of Total Number of Deaths.
Inside the Quarries—					
Falls of ground	43	43	117	122	42.2
By blasting	8	8	67	82	7.8
During ascent or descent	1	1	1	1	1.0
Miscellaneous	30	30	219	223	29.4
Total inside the quarries	82	82	404	428	80.4
Outside the quarries	20	20	145	147	19.6
Grand total	102	102	549	575	100

The following table is based upon the figures of a single year, and therefore cannot be considered as conclusive; but it seems that the "inside" worker, *i. e.*, the quarryman, has an employment with almost as much risk as the miner:

Kind of Mineral Wrought.	Death-Rate per 1000 Persons Employed Inside the Quarries.	Death-Rate per 1000 Persons Employed Outside the Quarries	Death-Rate per 1000 Persons Employed Inside and Outside the Quarries.
Chalk	0.89	0.41	0.56
Limestone	1.34	0.38	1.06
Sandstone	1.78	0.22	1.15
Granite	1.03	0.30	0.73
Slate	2.31	0.77	1.51
Igneous rock, other than granite	0.43	.....	0.33
Other minerals, etc	1.81	0.40	0.86
Totals	1.52	0.39	0.97

MINERAL PRODUCTION OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Alum Shale.		Arsenic.		Arsenical Pyrites.		Barytes.		Bauxite.		Clays.	
1891..	5,564	$3,410	6,147	$292,965	5,179	$21,850	27,317	$160,600	10,939	$16,140	3,274,855	$4,719,480
1892..	2,969	1,825	5,196	218,430	4,569	24,940	24,637	146,415	7,440	9,300	3,153,824	4,446,875
1893..	2,184	1,320	6,170	288,470	3,134	14,740	2,306	126,815	9,022	20,750	3,114,251	4,087,095
1894..	4,035	2,280	4,877	243,070	3,340	19,115	20,985	107,050	8,097	28,090	3,315,988	4,118,505
1895..	2,096	1,290	4,875	260,990	2,998	13,925	21,509	115,295	10,574	12,530	9,952,723	9,198,035

Year.	Coal.		Copper Ore.		Copper Precipitate.		Fluorspar.		Gold Ore.		Gypsum.	
1891.....	188,519,767	$370,499,080	8,981	$101,070	327	$21,775	143	$935	14,348	$62,000	154,195	$300,190
1892.....	184,713,649	330,252,255	6,092	57,965	274	15,560	174	940	10,151	45,840	149,915	291,135
1893.....	169,659,680	279,049,040	5,535	64,805	....	12,050	238	805	4 634	38,285	148,142	296,845
1894.....	191,389,965	313,650,895	5,844	69,545	445	11,565	128	345	6,708	67,865	155,905	331,775
1895.....	192,687,003	286,141,815	7,651	109,560	264	14,275	37	270	13,478	82,920	180,738	359,175

Year.	Iron Ore.		Iron Pyrites.		Lead Ore.		Manganese Ore.		Ocher and Umber.		Oil Shale.	
1891.....	12,987,159	$16,779,300	15,716	$40,010	44,578	$1,783,915	9,632	$31,065	13,825	160,515	2,399,826	$3,535,885
1892.....	11,494,809	14,853,160	14,142	34,785	40,668	1,482,420	6,176	22,170	12,326	83,910	2,123,585	2,612,420
1893.....	11,567,132	14,139,735	16,351	36,460	40,664	1,402,695	1,380	3,810	10,876	69,400	2,020,027	2,445,650
1894.....	12,565,288	15,953,235	15,771	40.212	41,250	1,334,975	1,837	3,700	8,652	70,200	2,018,167	2,482,980
1895.....	12,870,260	14,328,545	9,172	20,570	39,026	1,366,960	1,293	3,405	7,747	84,945	2,282,815	2,808.580

Year	Phosphate of Lime.		Salt.		Slates and Slabs.		Sulphate of Strontium.		Tin Ore (Black Tin).		Wolfram Ore.		Zinc Ore.	
1891.	10,164	$100,000	2,077,072	$4,884,120	421,829	$4,835,000	8,193	$20,150	14,726	$3,676,200	140	$16,705	22,580	$567,225
1892.	12,396	111.250	1,988,024	4,307,005	424,975	5,129.610	5.147	6.330	14,588	3,672,825	127	15,000	24,264	520,080
1893.	3,386	28,853	2,015,027	3,676,110	453,242	5,538,138	6.000	11,625	14.132	3,185,265	22	2,100	24,524	406,350
1894.	711	6,385	2.271,686	3,818,145	469,060	5,856,830	6,932	9,810	13,116	2,437,615	*Nil*	.....	22,170	336,555
1895.	508	4,375	2,218,025	3,548,755	591,068	6,370,730	12,529	17,645	10,781	1.852,650	...	.....	17,758	227,150

PRODUCTION OF CERTAIN METALS IN THE UNITED KINGDOM. (*a*) (IN METRIC TONS.)

Year.	Fine Copper.	Lead. (*b*)	Tin.	Zinc.	Silver from Lead—Kgms.
1891.....	731	32,733	9.503	9.037	8,673
1892.....	503	30,013	9,419	9.500	8,436
1893.....	439	30,173	9,124	9.585	8,663
1894.....	454	30,162	8,462	8,260	8,575
1895.....	588	29,464	6,754	6,760	8,722

(*a*) From *Mineral Statistics of the United Kingdom*, London. (*b*) Smelted from native ores.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Alkali.		Brimstone.		Copper Ore.		Copper Regulus and Precipitate.		Copper, Unwrought and Wrought.		Chemical Products.
1891....	4,428	$252,020	21,766	$647,485	90,423	$3,664,550	125,223	$16,507,525	44,958	$11,819,550	$6,845,365
1892....	2,857	207,580	26,499	738,905	93,399	2,926,865	136,328	16,450,530	35,626	8,279,710	7,511,470
1893....	4,456	395,520	26,171	624,985	82,099	2,430,110	120,621	14,164,700	42,694	9,571,585	6,778,230
1894....	7,908	519,120	24,281	513,140	84,138	2,468,125	80,114	8,844,835	57,970	11,821,590	6,876,435
1895....	9,353	572,930	25,319	496,835	101,353	2,873,015	92,732	11,164,775	43,219	9,159,030	7,449,615
1896....	8,949	479,650	22,990	470,060	88,732	2,833,955	92,252	11,432,545	62,148	14,412,265	8,350,460

Year.	Glass of All Sorts.		Guano.		Iron Ore.		Iron, Bar, Angle, Bolt, and Rod.		Iron, Girders, Beams & Pillars		Iron Manufactures.	
1891..	76,659	$9,045,205	24,003	$693,210	3,231,750	$12,267,035	78,674	$3,757,935	75,227	$2,549,400	157,540	$13,823,605
1892..	81,111	9,351,095	28,323	947,165	3,841,369	13,584,100	77.137	3,461.295	75,781	2,512,870	146,093	12,660,590
1893..	134,665	12,216,265	18,600	473,605	4,131,323	13,960,140	66,810	2,965,163	73,480	2,096,915	153,139	12,728,195
1894..	106,478	11,346,775	28,661	720,555	4,485,880	14,917,985	64,265	2,777,790	70,537	2,141,150	157,387	12,164,025
1895..	101,399	10,564,945	49,849	1,961,975	4,521,516	14,889,760	68,933	2,747,610	70,307	2,181,920	166,063	14,204,230
1896..	114,364	11,732,770	20,537	522,770	5,504,156	18,808,610	72,195	2,852,065	76,400	2,337,275	192,333	20,535,665

Year	Steel, Unwrought.		Lead, Pig & Sheet.		Nitrate of Soda.		Paraffine.		Petroleum. Liters.		Phosphate of Lime and Rock.	
1891	8,572	$437,840	172,456	$10,688,370	123,998	$5,249,090	28,418	$4,025,170	494,378,537	$13,426,840	260,906	$3,141,975
1892	6,587	312,430	185,725	9,882,180	121,486	5,100,960	28,074	3,748,520	492,754,491	12,234,530	319,187	3,328,544
1893	9,079	452,530	194,252	9,276,075	88,169	4,030,985	39,025	4,069,685	577,151,861	12,733,800	328,461	2,972,335
1894	8,726	384,294	163,970	5,259,275	127,697	5,845,765	31,835	3,165,295	616,963,562	12,424,880	386,102	3,614,675
1895	11,036	475,015	165,531	8,370,215	124,649	4,994,485	36,619	3,778,950	583,285,093	16,844,520	365,414	3,171,070
1896	17,771	790,730	170,276	9,267,340	108,148	4,182,760	36,850	3,555,250	861,597,854	18,610,280	295,923	2,329,655

Year.	Pyrites, Iron, Copper, or Sulphur.		Quicksilver.		Saltpeter.		Silver Ore.	Tin in Blocks, Ingots, Bars, or Slabs.		Zinc, Crude, in Cakes.		Zinc Manufactures.	
1891	626,148	$5,631,235	2,135	$2,537,515	14,163	$1,255,105	$18,831,695	28,661	$12,825,370	59,455	$6,647,525	20,483	$2,571,980
1892	614,142	5,317,530	1,939	1,980,920	15,666	1,339,655	15,160,845	29,942	13,719,070	53,643	5,512,955	19,263	2,315,400
1893	622,634	5,327,025	1,767	1,705,930	12,323	1,069,370	15,354,655	34,093	14,460,535	57,842	5,036,985	18,739	1,991,765
1894	625,968	5,258,510	1,952	1,516,810	14,694	1,327,795	12,199,775	39,777	13,592,495	54,768	4,098,745	18,844	1,775,110
1895	591,782	4,935,825	1,685	1,672,790	11,607	1,068,265	8,732,220	42,269	13,155,190	63,523	4,669,455	19,674	1,875,855
1896	598,471	4,999,965	1,604	1,688,645	17,309	1,392,420	6,483,665	38,989	11,448,440	77,890	6,225,065	21,339	2,097,185

MINERAL EXPORTS OF THE UNITED KINGDOM. (*a*) (METRIC TONS AND DOLLARS; £1 = $5.)

EXPORTS OF DOMESTIC PRODUCE.

Year.	Alkali.		Brass and Manufactures.		Cement.		Coal, Coke, Cinders, and Fuel.		(*b*) Coal, etc., for Steam'rs	Coal Products. (*c*)	Copper, Unwrought, in Ingots.	
1891	316,383	$11,676,405	5,793	$2,588,030	585,242	$5,703,485	31,584,570	$94,475,390	8,673,933	$7,849,570	35,928	$9,823,740
1892	299,018	10,597,200	5,498	2,272,685	500,546	4,514,550	30,944,282	84,053,790	8,738,591	6,661,865	42,972	10,307,220
1893	296,481	9,289,640	5,877	2,291,165	444,898	3,722,120	29,512,740	71,877,380	8,257,207	6,375,310	28,802	6,836,460
1894	303,786	8,148,895	6,527	2,023,575	432,782	3,519,755	33,600,000	86,879,035	9,444,000	6,319,215	19,770	4,255,585
1895	318,023	7,800,700	5,439	2,094,490	401,727	3,209,590	33,641,447	77,213,520	9,558,314	7,863,515	30,108	6,744,500
1896	245,927	6,210,140	6,055	2,439,535	359,525	2,902,590	34,810,270	75,802,885	10,096,302	9,031,070	23,704	5,698,495

Year.	Copper, Wrought or Manufactured.		Mixed or Yellow Metal.		Earthenware.	Glass of All Sorts	Hardware and Cutlery.	Implements, Tools and Parts.	Iron, Pig.		Iron, Bar, Angle, Bolt, and Rod.	
1891	16,425	$5,384,245	66,650	$19,140,560	$9,791,720	$5,067,085	$12,637,875	$6,557,865	853,580	$11,027,835	220,617	$7,314,500
1892	16,478	4,864,985	74,619	18,817,645	9,523,560	4,426,525	10,973,630	6,310,245	779,402	9,873,725	176,206	5,738,410
1893	17,277	4,861,505	66,889	3,410,870	9,117,405	3,829,700	10,233,030	6,130,725	853,391	9,857,590	151,329	4,647,760
1894	15,664	4,146,425	16,213	3,435,155	8,038,700	3,582,385	9,190,630	5,922,985	843,815	9,561,750	121,269	4,112,785
1895	15,918	4,217,380	15,205	3,165,550	9,197,705	3,945,270	9,314,790	6,238,540	880,446	10,377,745	146,348	4,445,250
1896	15,649	4,460,200	11,470	2,585,795	8,895,670	4,162,540	10,606,340	7,068,405	.....	.....	180,974	5,521,760

Year.	Iron, Railroad, of All Sorts.		Iron and Steel Wire and M'f's Except Tel. Wire		Iron Hoops, Sheet, Boiler, Armor. (*d*)		Iron, Galvanized Sheets.		Tin, Plates and Sheets.		Iron, Cast and Unwrought, and Manufactures.	
1891	713,553	$19,263,820	68,603	$5,715,635	160,000	$6,252,705	166,541	$11,550,540	455,598	$35,833,275	370,778	$24,032,005
1892	475,538	11,236,110	48,112	3,969,575	142,373	6,318,035	159,649	10,385,340	401,866	26,651,080	325,059	21,811,445
1893	567,823	12,555,070	37,735	3,237,305	141,845	6,096,035	169,407	10,228,750	385,480	24,956,500	285,095	18,825,125
1894	430,416	9,427,840	35,380	3,107,110	131,803	5,237,555	172,536	9,760,330	359,782	21,751,085	270,589	17,190,325
1895	465,359	9,509,010	42,932	3,555,350	104,722	3,822,800	207,469	11,245,465	371,838	21,223,975	294,682	18,704,435
1896	751,091	19,827,820	57,270	4,520,515	122,856	4,233,115	248,986	14,225,045	271,226	15,179,655	372,405	23,621,660

Year.	Old Iron for Remanufacture.		Steel, Unwrought.		M'f's of Steel or Steel and Iron Combined.		Total of Iron and Steel.		Lead, Pig, Sheet, Piping, and Manuf'res.		Locomotives.	Mining Machinery.
1891	112,856	$1,771,845	152,874	$8,660,365	17,311	$2,962,475	3,292,312	$134,385,000	49,010	$3,412,395	$8,086,270	.....
1892	108,202	1,638,080	151,532	8,703,270	15,491	2,503,770	2,783,381	108,828,840	59,098	3,542,550	4,918,760	.....
1893	120,461	1,671,370	170,407	8,509,170	18,826	2,378,110	2,903,753	102,962,885	49,723	2,763,005	2,313,300	$2,047,525
1894	87,387	1,134,365	215,428	9,881,090	19,158	2,490,465	2,699,345	93,655,700	47,905	2,594,340	3,750,195	1,933,555
1895	97,186	1,243,365	213,462	9,743,495	24,082	3,109,915	2,883,559	98,476,910	42,353	2,472,670	3,997,985	3,604,275
1896	129,382	1,694,160	301,579	12,548,345	37,410	4,634,955	3,609,123	119,066,195	41,907	2,616,140	5,383,110	5,318,520

Year.	Salt, Rock and White.		Telegraph Wire and Apparatus.	Tin, Unwrought.		Zinc or Spelter, Unwrought & Wrought.	
1891	682,078	$2,982,040	$7,170,100	5,249	$2,457,460	7,798	$808,665
1892	664,668	2,696,310	4,549,585	5,738	2,720,335	9,974	894,880
1893	646,389	2,522,500	5,314,435	6,848	3,031,995	9,895	793,155
1894	771,398	3,020,675	7,087,230	5,944	2,160,950	9,372	635,645
1895	753,338	2,732,925	3,949,050	5,748	1,915,145	10,023	650,675
1896	671,269	2,355,155	4,334,980	6,317	1,992,360	9,647	673,810

EXPORTS OF FOREIGN PRODUCE. (*a*) (IN METRIC TONS AND DOLLARS; £1 = $5.)

Year.	Chemical Manufactures and Products.	Copper, Unwrought and Part Wrought.		Glass of All Sorts.		Guano.		Iron, Bar, Angle, Bolt, and Rod.	
1891	$808,890	11,874	$3,290,605	5,229	$504,765	1,073	$52,290	57,034	$2,282,105
1892	1,083,545	11,261	2,797,665	4,516	460,360	1,961	60,000	47,225	1,914,320
1893	984,285	13,050	3,066,395	4,702	431,860	1,472	63,670	28,533	1,199,960
1894	1,081,660	6,597	1,394,975	4,735	372,505	1,549	53,785	18,663	700,685
1895	1,050,435	8,093	1,788,920	4,334	380,285	2,001	63,085	23,185	874,945
1896	1,093,585	9,809	2,338,310	3,950	312,460	1,727	69,465	16,882	638,705

Year	Steel, Unwrought.		Iron and Steel Manufactures.		Petroleum. Liters.		Quicksilver.		Saltpeter.		Tin, in Blocks, Ingots, Slabs, or Bars.	
1891	4,291	$230,795	29,750	$2,367,790	4,370,092	$267,855	1,928	$2,222,845	1,219	$111,715	14,856	$6,694,470
1892	3,004	147,390	38,761	2,638,255	9,010,298	527,005	1,713	1,662,885	1,574	131,335	16,627	7,665,500
1893	4,383	246,440	47,860	2,925,770	5,988,157	339,155	1,298	1,200,240	1,396	110,255	19,377	8,441,810
1894	3,789	146,520	35,521	2,117,910	7,497,161	288,525	1,482	1,296,890	2,716	237,195	21,887	7,615,235
1895	5,389	234,030	36,687	2,564,485	6,665,467	268,690	1,253	1,234,620	1,459	130,020	21,043	6,597,305
1896	3,999	167,345	39,099	3,761,350	11,709,010	354,425	1,198	1,150,020	1,913	152,765	19,006	5,669,945

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.*
(*b*) Coal, etc., shipped for the use of steamers engaged in the foreign trade. This not being an export in the ordinary acceptation of the term, the value thereof is not given in the returns.
(*c*) Including naphtha, paraffine, paraffine oil, and petroleum.
(*d*) Prior to 1895 iron black plates for tinning were included under this heading.

UNITED STATES.

STATISTICS and full particulars of the mineral production of the United States will be found in the Introduction and the articles on the different substances. We give below the mineral imports and exports for five years.

MINERAL IMPORTS OF THE UNITED STATES. (*a*)

Year.	Aluminum.							Antimony.			
	Crude.				Leaf.		Manufactured.				
	Lbs.	Kgms.	Value.	Value per Kgm.	Packages	Value.	Value.	Lbs.	Kgms.	Value.	Value per Kgm.
1892....	43	19	$51	$2.68	14,540	$1,202	$1,036	3,950,864	1,792	$392,761	$219.17
1893....	7,816	3,546	4,683	1.32	18,700	1,903	1,679	*Nil.*	*Nil.*	*Nil.*	*Nil.*
1894....	5,303	2,406	2 524	1.04	10,780	1,210	386	1,205,752	547	79,265	145.09
1895 .	25,294	11,473	7,814	.68	6,610	646	1,841	4,051,022	1,817	230,946	126.00
1896....	698	316	591	1.87	4,057	523	2,365	3,583,390	1,625	170,092	104.67

Year.	Antimony Ore.				Asbestos.			Asphaltum.			
	Lbs.	Kgms.	Value.	Va ue per Kgm.	Crude.	Manufactured.	Total.	Long Tons.	Metric Tons.	Value.	Value per Metric Ton.
1892 ...	192,344	87,247	$7,338	$0.08	$262,433	$7,209	$269,642	98,581	100,058	$336,868	$3.36
1893. ..	116,495	52,842	5,253	.10	175,602	9,403	185,005	76,996	78,228	196,314	2.51
1894 ...	375,468	170,312	18,068	.10	240,029	15,989	256,018	106,014	107,710	313,685	2.91
1895 ..	668,610	303,282	14,718	.05	225,147	19,731	244,878	79,194	80,461	212,407	2.63
1896....	1,180,828	535,620	21,392	.04	229,084	15,654	244,738	95,233	96,757	286,099	2.95

Year.	Barium Sulphate.								Bauxite.			
	Manufactured.				Unmanufactured.				Lbs.	Metric Tons.	Value.	Value per Metric Ton.
	Short Tons.	Metric Tons.	Value.	Value per Met. Ton.	Short Tons.	Metric Tons.	Value.	Value per Met. Ton.				
1892..	1,389	1,260	$15,419	$12.23	2,789	2,530	$7,418	$2.93	12,804,253	5,808	$57,948	$9.97
1893..	1,032	936	11,457	12.24	2,983	2,706	7,612	2.81	11,431,678	5,185	28,217	5.44
1894 .	836	758	10,556	13.92	1,884	1,709	5,270	3.08	2,305,783	1,046	6,661	6.36
1895..	1,629	1,478	17,112	11.57	2,551	2,314	7,561	3.26	12,985,525	5,890	29,194	4.95
1896..	.....	.....	.....	.....	.....	.....	.....	.....	4,765,974	2,161	10,477	4.84

Year.	Brass and Manufactures of. Value.	Calcium Chloride.				Cement. (*b*)			
		Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Barrels.	Metric Tons.	Value.	Value per Met. Ton.
1892..	$235,788	109,888,561	49,846	$1,962,084	$39.36	2,440,705	442,801	$3,378,331	$7.62
1893 .	215,650	98,618,147	44,735	1,843,410	41.20	2,674,149	485,326	3,470,169	7.16
1894..	134,356	96,256,251	43,673	1,697,038	38.85	2,638,107	478,785	3,396,729	7.09
1895..	158,822	103,317,968	46,855	1,628,877	34.76	2,975,082	543,844	3,873,123	7.12
1896..	157,798	97,510,926	44,231	1,441,337	32.36	2,667,676	484,020	3,393,199	7.01

Year.	Chrome Ore.				Chromic Acid.				Clays or Earths, Including Kaolin.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Kgms.	Value.	Value per Kgm.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	4,930	5,009	$55,589	$11.09	772	347	$204	$0.59	75,283	76,495	$605,022	$7.91
1893	6,354	6,456	58,629	9.08	3,708	1,682	641	.38	73,387	74,586	564,547	7.56
1894	3,470	3,525	38,364	10.88	6,680	3,030	1,045	.34	78,698	79,983	616,506	7.79
1895	5,230	5,314	82,845	15.59	2,083	945	414	.43	98,862	100,450	716,590	7.13
1896	8,669	8,808	187,400	21.27	2,429	1,002	387	.39	98,243	99,815	707,826	7.09

Year.	Coal.										
	Anthracite.				Bituminous.				Total.		
	Long Tons.	Metric Tons.	Value.	Per Met. Ton.	Long Tons.	Metric Tons.	Value.	Per Met. Ton.	Metric Tons.	Value.	Per Met. Ton.
1892	65,058	66,099	$197,583	$2.98	1,143,304	1,161,597	$3,744,862	$3.22	1,227,696	$3,942,445	$3.21
1893	53,768	54,628	148,112	2.79	1,082,993	1,100,321	3,623,892	3.29	1,154,949	3,772,004	3.27
1894	90,068	91,509	234,024	2.55	1,242,714	1,262,597	3,785,513	2.99	1,352,665	4,019,537	2.97
1895	141,337	143,598	328,705	2.28	1,212,023	1,231,415	3,626,623	2.94	1,375,013	3,955,328	2.87
1896	.....	.....	.....	.....	1,246,991	1,266,943	3,488,115	2.75	.....	.....	.....

Year.	Coke.				Cobalt Oxide.				Fine Copper in Ore.			
	Short Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Kgms.	Value.	Value per Kgm.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	24,482	22,210	$86,350	$3 88	32,833	14,775	$60,067	$4.06	8,107,582	3,677	$499,574	$135.86
1893	33,165	30,080	99,683	3.31	28,164	12,775	41,105	3.21	7,723,387	3,504	467,988	133.55
1894	29,137	26,427	70,358	2.66	24,020	10,900	29,857	2.73	3,870,769	1,756	297,118	163.50
1895	29,622	26,867	71,366	2.65	36,155	16,400	39,839	2.42	8,921,920	4,047	213,689	52.80
1896	43,372	39,347	114,713	2.91	27,189	12,333	36,212	2.93	2,636,480	1,196	126,580	105.83

Year	Copper Ingots, Old, etc.				Copper, Manufactures. Value.	Copper, Total Value.	Cryolite.			
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	1,724,772	782	$131,608	$168.29	$96,627	$727,809	7,241	7,357	$96,932	$13.17
1893	5,536,690	2,510	488,710	194.70	83,752	1,040,450	9,574	9,728	126,688	13.02
1894	2,289,324	1,039	170,368	163.97	66,256	533,742	10,684	10,855	142,494	13.12
1895	9,381,800	4,255	839,603	197.32	60,344	1,113,636	9,500	9,652	127,165	13.17
1896	11,397,273	5,170	961,920	186.05	85,123	1,173,623	3,009	3,057	40,056	13.10

Year.	Earthen, Stone, and China Ware	Emery Grains.				Emery Rock.				Other Manufactures. Value.	Total Value.
		Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.		
1892	$8,987,111	566,448	257	$22,586	$87.87	5,280	5,365	$95,625	$17.82	$2,412	$120,623
1893	8,769,778	516,958	234	20,073	86.12	5,066	5,147	103,875	20.18	3,819	127,767
1894	6,978,572	597,713	271	18,645	68.80	1,642	1,668	51,487	30.86	1,830	71,962
1895	10,523,925	678,761	308	25,066	81.38	6,803	6,912	80,386	11.62	27,586	133,038
1896	9,690,234	751,464	341	26,520	77.77	6,289	6,390	119,667	18.72	1,971	148,158

Year.	Guano.				Iron Ore.				Pig Iron.			
	Long Tons.	Metric Tons	Value.	Value per Met. Ton	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	3,073	3,122	$46,014	$14.77	806,585	819,571	$1,795,644	$2.19	70,125	71,247	$1,604,806	$22.52
1893	5,951	6,045	97,890	16.19	526,951	535,435	906,687	1.69	54,394	55,264	1,224,347	22.15
1894	5,646	5,736	105,879	18.45	168,541	171,238	267,241	1.56	15,582	15,831	407,638	25.74
1895	4,066	4,131	48,917	11.84	524,153	532,571	786,207	1.47	53,232	54,084	1,337,978	24.73
1896	6,838	6,947	88,231	12.70	682,806	692,999	1,036,917	1.49	56,272	57,172	1,207,890	22.87

Year.	Scrap Iron and Steel.				Bar Iron, Rolled and Hammered.				Bars, Railway, of Iron or Steel.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	29,223	29,691	$418,622	$14.09	43,191,218	19,592	$776,927	$39.66	347	353	$10,607	$30.04
1893	6,273	6,373	97,552	15.30	33,367,805	15,136	603,985	39.90	2,888	2,934	57,584	19.62
1894	2,380	2,418	43,710	18.07	20,669,825	9,576	377,397	39.41	300	304	4,292	14.11
1895	6,066	6,163	103,523	16.79	44,910,023	20,381	772,822	37.91	1,447	1,470	27,076	18.41
1896	8,250	8,382	130,012	15.51	36,458,326	16,538	686,730	41.53	7,796	7,921	207,648	26.21

Year.	Hoop, Band, or Scroll.				Ingots, Blooms, Slabs, Billets, etc.				Sheet, Plate, and Taggers Iron or Steel.			
	Lbs.	Met. Tons.	Value.	Value per Met. Ton.	Lbs.	Met. Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	2,635,462	1,194	$72,950	$61.09	68,512,730	31,077	$1,591,092	$51.19	60,352,051	27,376	$1,400,980	$51.17
1893	2,278,828	1,034	65,111	62.96	60,183,663	27,299	1,293,834	47.39	83,598,034	37,920	1,746,035	46.04
1894	1,803,104	818	51,111	62.48	21,266,348	9,646	809,184	83.89	64,500,887	29,258	1,197,671	40.93
1895	33,168	15	2,925	195.00	58,811,651	26,677	1,610,889	60.38	32,549,127	14,764	727,638	49.28
1896	59,105	27	5,654	209.40	45,598,270	20,683	1,657,805	80.15	15,201,124	6,896	328,421	47.62

Year.	Ties for Baling Cotton.				Tin Plates, Terne Plates, and Taggers Tin.			
	Lbs.	Metric Tons.	Value.	Value per Metric Ton.	Lbs.	Metric Tons.	Value.	Value per Metric Ton.
1892	2,265,079	1,027	$65,819	$64.08	601,378,109	272,783	$17,102,487	$62.69
1893	3,801,488	1,724	78,325	45.43	567,065,948	257,174	15,559,423	60.50
1894	123,414	46	3,392	73.74	481,751,601	218,521	12,053,167	55.15
1895	7,806,304	3,541	102,309	28.89	491,880,513	223,100	11,482,380	51.46
1896	16,031,468	7,272	235,830	32.43	266,943,267	120,803	6,140,161	50.82

Year.	Wire Rods.				Wire, and Articles made from.				Manufactures, Value.	Total Value of Iron Imports.
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton		
1892	91,480,980	41,496	$1,655,935	$39.90	8,580,505	3,892	$577,853	$148.46	$8,604,369	$33,882,447
1893	78,438,146	35,579	1,387,066	38.98	9,286,323	4,212	588,817	139.79	6,954,460	29,656,539
1894	50,640,787	22,971	923,675	40.21	9,604,368	4,357	514,484	118.08	4,457,855	20,843,576
1895	60,107,521	27,264	1,090,707	40.00	12,931,695	5,866	716,901	122.21	7,796,988	25,772,136
1896	42,509,008	19,281	811,224	42.07	8,231,248	3,734	492,953	132.01	7,602,258	19,506,576

Year.	Lead, Ore, and Dross.				Lead, Pig, Bars, Scrap, and in Ore.			
	Lbs.	Metric Tons.	Value.	Value per Metric Ton.	Lbs.	Metric Tons.	Value.	Value per Metric Ton.
1892	54,249,291	24,607	$1,278,114	$51.94	1,549,771	702	$110,593	$157.54
1893	58,487,319	26,529	1,004,295	37.85	3,621,525	1,643	141,405	86.06
1894	33,020,250	14,978	437,999	29.24	139,961,911	63,486	4,228,706	66.60
1895	45,052,674	20,435	687,222	33.62	201,617,855	91,453	2,910,547	31.82
1896	37,829,583	17,159	631,381	36.79	160,318,517	72,720	1,780,265	24.48

Year.	Lead, Sheet, Pipe, and Shot.				Lead, Other Manufactures.	Total Value of Lead Imports.	Manganese. Ore.			
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	90,153	40	$6,207	$156.75	$2,063	$1,394,977	58,364	59,298	$830,046	$14.00
1893	56,678	26	2,914	112.08	1,691	1,150,305	67,717	68,807	860,832	12.51
1894	44,080	20	2,050	102.50	1,552	4,670,307	44,655	45,384	432,561	9.53
1895	128,008	58	5,030	86.72	12,961	3,615,760	86,111	87,489	747,910	8.54
1896	216,820	98	8,513	86.86	5,587	2,425,746	31,489	31,993	250,468	7.82

Year.	Manganese—*Continued.* Manufactured. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Marble and Stone, and Manuf's of. Marble and Manufactures of. Value.	Stone and Manufactures of. Value.	Total. Value.	Metals, Metal Compositions and Manufactures of, N.E.S. Bronze Manufactures Value.	All Other. Value.	Total. Value.
1892..	208	211	$10,305	$48.83	$1,011,823	$513,448	$1,525,271	$706,406	$5,764,512	$6,470,918
1893..	.....	.....	.....	.....	1,062,843	574,322	1,637,165	631,818	5,693,489	6,325,307
1894..	.....	.....	.....	.....	787,657	864,733	1,652,390	473,672	3,293,477	3,767,149
1895..	.....	.....	.....	.....	865,243	411,122	1,276,365	330,684	2,556,235	2,886,919
1896..	.....	.....	.....	.....	808,030	387,979	1,196,009	279,268	2,038,645	2,317,913

Year.	Mica. Value.	Mineral Substances N. E. S. Value.	Nickel. Value.	Oil Mineral. Gallons.	Liters.	Value.	Value per Liter.	Ozokerite. Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892 ...	$100,846	$275,615	$426,817	909,885	3,443,913	$49,487	$0.014	1,250,000	567	$150,000	$264.55
1893....	120,864	261,646	384,628	523,800	1,842,583	31,801	.017	1,744,905	792	133,111	168.06
1894....	126,184	139,688	45,927	152,870	578,659	27,667	.047	.....	.....	.....	.....
1895....	174,886	111,447	56,890	1,917,823	7,259,804	54,963	.007	.....	.....	.....	.....
1896 ...	169,086	74,395	32,532	68,803	260,419	14,062	.054	.....	.....	.....	.....

Year.	Paints & Colors. Value.	Platinum, Manufactured. Kgms.	Value.	Value per Kgm.	Platinum Ore, Sponge, Plate. Kgms.	Value.	Value per Kgm.	Platinum Vases, Retorts, etc. Kgms.	Value.	Value per Kgm.
1892....	$1,366,844	.....	.....	.....	1,774.96	$505,852	$284.99	170.63	$53,763	$315.08
1893 ...	1,294,857	.....	$1,412	.....	1,993.20	538,478	270.15	.....	69,823	.....
1894...	1,045,251	.....	.....	.....	1,670.90	485,272	290.42	.....	88,195	.....
1895....	1,363,657	.....	171	.....	2,320.13	690,584	297.64	.....	27,354	.....
1896....	1,189,781	2,521	906,671	$359.64	2,584.16	437,189	169.18	.....	106,286	.....

Year.	Potash, Chromate and Bichromate. Lbs.	Metric Tons.	Value.	Value per Metric Ton.	Potash, Muriate. Lbs.	Metric Tons.	Value.	Value per Metric Ton.
1892..... ...	1,193,972	541	$94,708	$175.16	70,227,971	31,900	$1,098,267	$23.48
1893.....	979,706	444	78,981	177.88	74,663,116	38,407	1,192,516	31.04
1894.....	1,483,762	673	125,796	186.91	101,597,074	46,097	1,540,081	33.40
1895.....	2,045,910	928	181,242	195.30	81,833,531	37,120	1,296,184	34.62
1896.....	952,794	432	80,538	186.43	86,525,983	39,248	1,392,504	35.47

Year.	Potash, Nitrate. Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Precious Stones. Rough & Uncut. Value.	Cut. Value.	Pyrites. Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Average Sulph. % Contents
1892..	13,012,087	5,900	$382,771	$64 87	$1,032,869	$13,427,774	210,000	213,381	....	.....	43
1893..	13,374,016	6,066	369,274	60.87	802,075	10,022,371	194,934	198,072	$721,699	$3.64	(*d*)
1894..	9,375,950	4,254	249,842	58.73	839,806	6,710,472	163,546	166,163	590,905	3.55	25
1895..	11,419,090	5,180	305,207	58.91	111,033	6,623,669	190,436	193,483	673,812	3.48	....
1896..	20,085,327	9,111	479,899	52.67	78,815	3,367,723	199,678	202,873	1,140,571	5.62	.....

Year.	Salt. Lbs.	Metric Tons.	Value.	Value per Metric Ton.	Soda, Nitrate. Long Tons.	Metric Tons.	Value.	Per Metric Ton.
1892	456,613,733	207,213	$768,734	$3.70	95,293	96,818	$2,933,174	$30.29
1893....	332,939,120	151,515	568,202	3.75	115,012	116,852	3,673,537	31.43
1894	420,226,039	191,118	643,167	3.36	98,026	99,594	3,195,356	31.98
1895	566,869,046	257,130	760,811	2.95	118,477	120,373	3,778,360	31.38
1896...	526,504,050	238,820	696,197	2.91	115,504	117,352	3,566,744	30.39

Year.	Soda, Bicarbonate.				Soda, Caustic.			
	Lbs.	Metric Tons.	Value.	Value per Metric Ton	Lbs.	Metric Tons.	Value.	Value per Metric Ton.
1892.....	1,466,595	665	$25,874	$38.90	54,384,120	24,670	$1,339,500	$54.29
1893.....	1,380,426	526	23,136	43.98	52,116,492	23,641	1,171,878	49.56
1894.....	4,479,524	2,028	63,625	31.37	44,772,512	20,309	911,942	44.90
1895.....	8,682,028	3,937	123,425	31.35	72,019,114	32,668	1,211,090	37.13
1896.....	4,343,786	1,970	60,878	30.90	47,827,247	21,694	859,809	39.63

Year.	Soda Ash and Sal Soda.				Other Soda Salts.			
	Lbs.	Metric Tons.	Value.	Value per Metric Ton.	Lbs.	Metric Tons.	Value.	Value per Metric Ton.
1892.....	361,648,637	164,073	$4,698,379	$28.63	40,954,822	18,580	$284,853	$15.33
1893.....	348,934,906	158,346	3,983,039	25.16	29,850,109	13,544	205,523	15.17
1894.....	321,300,874	144,971	3,665,836	25.28	17,717,714	8,016	132,385	16.51
1895.....	307,026,094	139,266	2,321,612	16.67	9,943,915	4,510	155,006	34.37
1896.....	191,780,537	86,991	1,443,417	16.59	5,623,071	2,551	100,917	39.56

Year.	Sulphur, Crude.				Sulphur, Flowers.				Sulphur, Refined.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	100,721	102,342	$2,189,307	$21.19	158	160	$5,439	$33.99	26	27	$4,106	$152.07
1893	107,601	109,333	1,903,191	17.40	241	245	5,746	23.45	43	44	1,017	23.11
1894	124,467	126,500	1,734,643	13.71	165	168	4,145	24.67	41	42	1,207	28.74
1895	115,959	127,974	1,593,148	12.45	581	590	12,888	21.84	229	233	4,378	18.79
1896	145,318	147,643	2,085,076	14.12	665	675	13,266	19.65	447	454	8,226	18.11

Year.	Tin.				Zinc.					
					Block, Pig, and Old.				Manufactures of, Net Value.	Total Value.
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.		
1892 ...	46,821,958	21,238	$9,415,889	$443.35	410,896	186	$23,307	$125.25	$22,709	$46,016
1893 ..	40,184,556	18,228	8,007,292	439.27	425,998	193	22,931	118.86	20,756	43,687
1894 ...	39,268,648	17,812	5,944,065	333.71	512,932	233	17,271	74.12	12,342	29,613
1895 ...	54,252,045	24,609	7,405,619	300.93	864,113	392	29,352	74.87	12,183	41,535
1896 ...	44,639,324	20,248	5,848,933	288.86	856,044	388	25,904	66.76	15,278	41,632

(*a*) From *Summary Statements of the Imports and Exports of the United States.* The following substances, not given in the above table, were imported in 1893: Bromine, 780 lbs., $234; plaster of paris, unground, 164,300 long tons, $180,263; ground, 1368 long tons, $22,643; calcined, 2565 long tons, $18,316; ocher and ochery earths, dry, 6,126,006 lbs., $53,943; ground in oil, 51,804 lbs., $3321; spiegeleisen and ferro-manganese, 37,457 long tons, $879,731; quicksilver, 30,191 lbs., $12,507; talc, 2,720,528 lbs., $12,825; litharge, 42,582 lbs., $1310; red lead, 832,152 lbs., $26,885; white lead and white paint containing lead, 682,912 lbs., $33,973; in 1894, lead ore and dross, 33,020,250 lbs., $437,999. In 1894: White lead and white paint containing lead, 796,479 lbs., $40,439; red lead, 947,873 lbs., $29,064; litharge, 38,595 lbs., $1064; zinc oxide, dry, 3,371,292 lbs.; in oil, 59,291 lbs.; bromine, 20 lbs., $11; ocher and ochery earths, dry, 4,937,738 lbs., $45,276; ground in oil, 22,387 lbs., $2100; plaster of paris, unground, 162,500 short tons, $179,237; ground, 340 short tons, $5966; calcined, 1687 short tons, $10,857; quicksilver, 7 lbs., $6; talc, 1,242,171 lbs., $6815; borax, 1,029,786 lbs., $39,256. In 1895: Borax, 4,478,749 lbs., $110,190; ocher and ochery earths, dry, 7,107,983 lbs., $56,020; *idem*, ground in oil, 41,152 lbs., $2239; plaster of paris, unground, 192,549 tons, $215,705; *idem*, ground, 135 tons, $848; *idem*, calcined, 3157 tons, $20,678; quicksilver, $2503; talc, 6,329,268 lbs., $25,843; litharge, 97,667 lbs., $2812; red lead, 1,764,274 lbs., $53,129; white lead and white paint containing lead, 1,897,891 lbs., $79,888; zinc oxide, dry, 4,546,038 lbs., $146,369; *idem*, in oil, 129,343 lbs., $7272. In 1896: Ocher and ochery earths, dry, 8,954,253 lbs., $68,196; *idem*, ground in oil, 27,023 lbs., $1561; plaster of paris, unground, 180,270 tons, $193,549; *idem*, ground, 2 tons, $74; *idem*, calcined, 3293 tons, $21,984; quicksilver, $2037; talc, 3,901,314 tons, $18,693; litharge, 51,050 tons, $1615; red lead, 1,543,262 tons, $47,451; white lead and white paint containing lead, 1,183,540 lbs., $52,411; zinc oxide, dry, 4,572,781 lbs., $146,-395; *idem*, in oil, 311,023 lbs., $14,793.

(*b*) Custom-house returns for these years are given in pounds, which are reduced to barrels of 400 lbs., for convenience of comparison. (*d*) Containing above 25% of sulphur.

Under the direction of Mr. Worthington C. Ford, who is an accomplished statistician, the reports of American imports and exports have improved very much in thoroughness and itemization, and consequently in value to industry in general. We have in this volume converted all English into metric measures, a necessary proceeding in view of the cosmopolitan nature of THE MINERAL INDUSTRY, and have also computed the value of each substance per metric ton. These averages should be reliable indices of trade fluctuations.

MINERAL EXPORTS OF DOMESTIC PRODUCTION OF THE UNITED STATES. (*a*)

Year.	Asbestos.	Brass & Manufactures of.	Cement.				Chemicals, Drugs, & Medicines.	Coal. Anthracite.			
	Value.	Value.	Bbls.	Metric Tons.	Value.	Value per Met. Ton.	Value.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	$4,800	$438,486	(*b*) 79,170	14,364	$137,167	$9.54	$6,618,827	851,639	865,265	$3,722,903	$4.30
1893	17,134	741,317	(*b*) 93,485	16,961	154,461	9.10	7,002,879	1,334,287	1,355,736	6,241,007	4.63
1894	5,762	779,875	(*b*) 108,000	19,596	165,809	8.46	7,722,532	1,440,625	1,463,675	6,359,021	4.34
1895	.. ....	730,424	(*b*) 83,632	15,174	117,646	7.75	8,749,090	1,470,710	1,494,241	5,937,130	3.96
1896	.....	1,026,191	54,639	9,913	86,757	8.75	9,261,354	1,350,000	1,371,600	5,925,506	4.32

Year.	Coal—*Continued.* Bituminous.				Total Long Tons.	Total Metric Tons.	Total Value.	Copper Ore. (*c*)			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.				Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	1,645,686	1,672,017	$4,999,289	$2.98	2,497,325	2,537,282	$8,722,192	113,113,616	51,308	$6,479,758	$126.29
1893	2,324,591	2,361,784	6,009,801	2.54	3,658,878	3,717,520	12,250,808	93,527,950	42,424	4,257,128	100.35
1894	2,195,716	2,230,847	4,970,270	2.22	3,636,341	3,694,522	11,329,291	9,748,480	4,422	440,129	99.53
1895	2,211,983	2,247,375	4,816,847	2.14	3,682,693	3,741,616	10,753,977	30,965,760	14,046	1,631,251	116.13
1896	2,276,202	2,312,621	5,072,818	2.19	3,626,202	3,684,221	10,998,324	41,452,240	18,797	2,393,914	127.35

Year.	Copper, Pig, Sheet, Old.				Copper. Manufactures. Value.	Copper. Total Value.	Earthen, and China Ware.	Glassware.	Gold and Silver in Coin and Bullion. (*e*)	
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.			Value.	Value.	Gold.	Silver.
1892	*d* 30,515,736	13,742	$3,438,948	$250.25	$245,064	$10,163,770	$245,731	$938,154	$76,533,056	$35,975,834
1893	*d* 138,984,128	63,043	14,213,378	225.45	464,991	18,935,497	151,308	971,503	79,775,820	46,288,721
1894	162,393,000	73,681	15,324,925	207.99	378,040	16,143,094	138,289	917,519	101,819,924	47,044,205
1895	121,328,390	55,034	12,222,769	222.09	1,084,289	14,938,309	139,297	1,002,328	104,605,623	53,833,153
1896	259,223,924	111,758	27,822,289	248.95	819,017	31,035,211	169,342	1,115,252	56,742,844	63,029,336

Year.	Gold and Silver in Ores. (*f*)		Iron Ore.				Iron, Pig.			
	Gold.	Silver.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	$9,262	$1,592,934	806,585	819,490	$1,795,644	$2.19	15,427	15,675	$282,290	$11.63
1893	190,849	190,846	526,951	535,382	906,687	1.69	24,587	24,989	379,437	11.18
1894.....	149,501	149,501	168,541	171,238	267,241	1.56	24,480	24,872	309,222	12.43
1895... .	... ...	.....	524,153	532,539	786,207	1.47	26,164	26,584	371,297	13.96
1896 ...	209,621	992,675	682,806	693,731	1,036,917	1.49	61,071	62,048	943,022	15.35

Year.	Iron, Bar.				Iron, Boiler, Band, Hoop, Scroll, and Sheet.				Iron Nails and Spikes, Cut.			
	Long Tons.	Metric Tons.	Value	Value per Met. Ton	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	963	979	$60,463	$61.76	31	31	$1,762	$56.84	15,268,454	6,925	$226,110	$32.63
1893	1,842	1,872	94,239	50.34	255	259	10,467	40.41	13,191,018	5,983	282,176	47.16
1894	3,195	3,248	130,374	43.22	99	100	5,380	53.80	18,321,904	8,311	331,133	39.84
1895	3,329	3,383	147,798	43.68	198	201	8,169	40.64	17,639,414	8,001	339,825	42.47
1896	3,505	3,558	168,680	47.38	268	272	12,303	45.23	23,708,847	10,754	458,758	43.59

Year.	Iron Nails, Spikes, Wire, Rod, Horseshoe, and all other, inc. Tags.				Plates and Sheets of Iron.				Plates and Sheets of Steel.			
	Lbs.	Metric Tons	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	2,158,732	979	$158,825	$163.23	581,335	264	$19,900	$75.37	252,356	114	$8,582	$75.28
1893	2,745,147	1,245	173,275	139.18	4,254,617	1,930	97,677	50.61	1,219,902	553	35,885	64.89
1894	3,892,015	1,765	196,175	111.15	4,987,830	2,262	111,586	49.33	1,856,832	842	53,641	63.17
1895	5,301,208	2,304	239,610	104.00	946,470	429	32,170	74.98	1,818,319	825	60,889	73.80
1896	9,563,812	4,338	362,291	83.51	1,725,779	784	48,905	62.37	4,144,127	1,880	90,568	51.36

Year.	Rails of Iron.				Rails of Steel.				Wire.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Met. Tons.	Value.	Value per Met. Ton.
1892	486	494	$13,568	$28.01	7,496	7,616	$242,757	$31.87	33,791,704	15,328	$1,009,225	$66.49
1893	164	167	5,059	30.29	19,712	20,027	590,929	29.56	37,504,808	17,012	1,006,014	59.13
1894	1,327	1,348	32,306	24.34	12,229	12,425	323,880	26.07	59,386,012	26,937	1,304,794	48.44
1895	6,792	6,900	139,470	20.21	8,807	8,948	222,061	24.88	66,024,358	29,948	1,375,195	45.90
1896	628	638	18,581	28.18	72,503	73,663	1,712,716	23.30	35,216,157	15,974	1,788,095	111.93

Year.	Lead Manufactures.	Marble, Stone, & Manufactures of.	Mica.	Nickel. (*g*)	Petroleum (1 = 1000 in Quantities and Value).							
					Crude.				Naphtha.			
	Value.	Value.	Value.	Value.	Gallons	Liters.	Value.	Value per Liter.	Gallons	Liters.	Value.	Value per Liter.
1892	$154,375	$765,165	$250	$197,763	104,397	395,143	$4,696	$0.012	16,393	62,047	$1,038	$0.017
1893	508,090	964,616	480	378,577	115,091	435,619	3,966	.009	16,344	61,862	1,004	.016
1894	497,993	1,009,704	659	495,218	114,825	424,612	4,661	.010	14,915	56,453	912	.016
1895	214,856	959,871	*nil*	239,897	116,108	439,469	6,286	.014	12,922	48,910	1,000	.024
1896	607,373	1,156,051	*nil*	606,833	118,133	447,133	6,032	.013	13,641	51,631	1,123	.021

Year.	Petroleum—*Continued.*											
	Illuminating				Lubricating.				Residue, etc.			
	Gallons	Liters.	Value.	Value per Liter.	Gallons	Liters.	Value.	Value per Liter.	Gallons	Liters.	Value.	Value per Liter.
1892	589,418	2,230,947	$31,827	$0.014	34,027	128,792	$5,131	$0.039	403	1,562	$38	$0.024
1893	711,828	2,710,269	31,796	.011	35,645	134,916	5,069	.037	543	2,104	35	.016
1894	734,057	2,778,406	30,299	.019	39,946	151,195	5,369	.035	119	450	10	.022
1895	686,006	2,596,533	43,540	.017	47,877	180,214	6,289	.035	170	643	14	.022
1896	758,076	3,869,317	49,704	.013	51,705	195,705	6,770	.034	521	1,972	28	.014

Year.	Petroleum—*Continued.*				Quicksilver.				Tin Manufactures. Value.
	Paraffine.								
	Gallons.	Liters.	Value.	Value per Liter.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	
1892	69,876	264,481	$4,160	$0.016	269,104	122	$133,626	$1,095.29	$204,429
1893	99,061	373,346	4,553	.012	1,272,271	577	542,410	940.05	258,449
1894	84,983	321,661	3,277	.012	1,102,187	500	397,528	795.05	301,503
1895	114,234	432,376	4,505	.010	1,188,955	504	482,085	956.91	252,223
1896	112,517	425,877	4,563	.010	1,525,726	692	618,437	895.02	268,581

Year.	Zinc Ore and Oxide.				Zinc Sheets, Pigs, Bars.				Manufactures. Value.	Total Value.
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.		
1892	2,058,560	934	$41,186	$44.09	12,494,335	5,667	$669,549	$118.15	$166,794	$877,529
1893	109,760	50	1,271	25.42	7,446,934	3,379	413,673	122.13	224,787	639,731
1894	.....	.....	5	.....	3,607,050	1,619	144,074	89.49	99,406	243,485
1895	53,760	24	1,008	42.00	3,060,805	1,388	153,175	110.35	50,051	204,234
1896	4,648,000	2,008	47,408	22.49	20,260,169	9,160	1,013,620	111.85	51,001	1,112,029

MINERAL EXPORTS OF FOREIGN PRODUCE FROM THE UNITED STATES. (*a*)

Year.	Asphaltum or Bitumen (Crude).				Brass and Manufactures of.	Cement.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Value.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	84	85	$2,513	$29.58	$8,702	8,614,537	3,907	$32,371	$8.28
1893	378	384	4,838	12.60	3,430	5,710,675	2,590	20,202	7.80
1894	502	510	9,211	18.06	6,909	3,889,928	1,764	15,072	8.54
1895	72	73	1,906	26.11	8,711	3,578,230	1,623	13,895	8.56
1896	113	115	2,615	22.74	3,488	4,437,261	2,013	16,558	8.22

Year.	Chemicals.											
	Salts of Potash. (*h*)				Chloride of Lime.				Nitrate of Soda.			
	Lbs.	Kgms.	Value.	Value per Kgm.	Lbs.	Kgms.	Value.	Value per Kgm.	Lbs.	Kgms.	Value.	Value per Kgm.
1892	.....	.....	.....	.....	.....	.....	.....	.....	479,360	217,488	$8,355	.04
1893	109,778	49,794	$4,922	.09	2,506	1,164	$231	.19	5,438,720	2,467,003	105,624	.04
1894	719,361	326,529	14,436	.04	19,943	9,046	550	.06	1,229,760	557,819	19,819	.04
1895	379,767	172,260	12,555	.07	10,600	4,808	305	.06	2,428,160	1,101,413	44,847	.04
1896	774,452	351,291	22,333	.06	5,500	2,495	105	.04	1,462,720	663,490	24,254	.04

Year.	Chemicals—*Continued.*											
	Caustic Soda.				Sal Soda and Soda Ash.				All Other Salts of Soda.			
	Lbs.	Kgms.	Value.	Value per Kgm.	Lbs.	Kgms.	Value.	Value per Kgm.	Lbs.	Kgms.	Value.	Value per Kgm.
1892	1,408,986	639,125	$37,872	.06	46,876	21,263	$735	.03	63,591	28,845	$942	.03
1893	1,760,856	798,724	45,724	.06	193,556	87,797	2,519	.02	68,759	32,189	1,135	.04
1894	2,929,473	1,328,809	66,017	.05	101,886	46,216	1,305	.03	131,118	59,475	5,648	.09
1895	1,592,150	722,199	31,691	.04	269,765	122,365	2,423	.02	68,127	30,902	1,174	.04
1896	1,703,173	772,559	33,433	.04	470,102	213,238	4,098	.02	36,839	16,710	380	.02

Year.	Clays or Earths of All Kinds, including China Clay.				Coal, Bituminous.				Copper.			
									Copper Ore, Fine Copper Therein.			
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892	60	61	$514	$8.42	127	129	$1,310	$10.15	707,739	321	$38,088	$118.65
1893	175	178	1,531	8.60	9	9	126	14.00	1,012,267	459	81,187	176.88
1894	92	94	616	8.67	877	891	4,934	5.54	235,140	106	20,008	188.68
1895	89	90	740	8.22	3,440	3,495	7,699	2.20	2,457,280	111	111,007	100.00
1896	32	33	109	3.30	5,204	5,287	7,990	1.51	975,200	442	32,715	74.01

Year.	Copper—*Continued.*					Earthen, Stone, and China-ware.	Fertilizers.			
	Pigs, Bars, Ingots, Old, and Other Unmanufactured.				Manufactures. Value.		Guano.			
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.			Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892	1,274,410	578	$97,364	$168.45	$6,711	$11,458	1	1	$44	$44.00
1893	1,007,554	457	80,236	175.57	20,546	15,657	6	6	295	49.16
1894	1,150,169	521	78,544	150.75	13,056	19,541	106	107	1,620	15.13
1895	475,588	215	43,193	200.89	11,626	25,482	.....	.....	.....	.....
1896	.....	.....	.....	.....	7,307	28,267	26	26	657	25.27

Year.	Fertilizers—*Continued.*					Glass and Glass-ware.	Gold-bearing Ores, N. E. S.	Gold and Silver, Manufactures of.	Graphite.			
	Phosphates, Crude or Native.				Other Fert'z's							
	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Value	Value.	Value.	Value.	Long Tons	Metric Tons.	Value.	Value per Met. Ton.
1892	.....	.....	.....	.....	.....	$12,761	$13,272	$69,314	.....	.....	.....	.....
1893	.....	.....	.....	.....	$222	11,366	17,057	17,308	164	167	$5,615	$33.62
1894	.....	.....	.....	.....	8	26,509	....	55,065	43	44	4,371	99.33
1895	743	755	$12,775	$16.92	2,154	15,303	362,379	53,716	5	5	305	61.00
1896	.....	.....	.....	.....	1,111	13,761	.....	29,482	6	6	296	46.00

Year.	Iron Ore.				Iron and Steel, and Tin Plate, etc.							
					Pig Iron.				Scrap Iron and Steel, fit only to be remanufactured.			
	Long Tons.	Metric Tons	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.
1892.	31	32	$211	$6.59	133	135	$1,544	$11.43	340	345	$6,164	$17.86
1893.	.....	.....	.....	.....	35	36	350	9.72	117	119	1,800	15.12
1894.	.....	.....	.....	.....	83	84	982	11.45	1,526	1,550	18,422	11.88
1895.	.....	.....	.....	.....	210	213	2,536	11.89	92	94	1,024	10.89
1896.	.....	.....	.....	.....	599	609	12,220	20.06	240	244	2,160	8.85

Year.	Iron and Steel, and Tin Plate—*Continued.*											
	Bar Iron, Rolled or Hammered.				Railway Bars of Iron or Steel or in Part of Steel.				Ingots, Blooms, Slabs, Billets, and Bars of Steel, and Steel in Forms, N. E. S.			
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Long Tons.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.
1892.	93,883	43	$1,268	$29.49	100	102	$2,433	$23.86	39,794	18	$1,938	$107.67
1893.	7,067	3	148	49.33	1,243	1,263	24,297	19.23	181,111	82	10,232	124.78
1894.	72,160	33	1,606	48.66	546	555	10,449	18.82	64,650	29	8,094	286.00
1895.	8,207	3	288	96.00	204	207	3,755	18.14	43,143	19	2,499	131.52
1896.	19,107	9	610	67.77	.....	.....	....	.....	.....	.....	.....	.....

Year.	Iron and Steel, and Tin Plate—*Continued.*									Lead and Manufactures of.	Marble and Stone and Manufactures of
	Steel Plate and Taggers, Wire Rod, Wire and Wire Rope, and Structural Iron or Steel.				Tin Plates, Terne Plates, and Taggers Tin.				Manufactures. Value.		
	Lbs.	Metric Tons.	Value.	Value per Met. Ton.	Lbs.	Metric Tons.	Value.	Value per Met. Ton.		Value.	Value.
1892..	296,023	132	$9,535	$72.23	548,134	248	$16,433	$66.26	$83,260	$3,348,124	$1,892
1893..	336,682	152	8,503	55.94	1,256,393	570	36,542	64.11	164,721	5,478,182	10,736
1894..	234,857	107	5,141	48.04	920,426	418	29,402	70.34	175,727	3,316,481	18,832
1895..	516,067	235	12,026	51.17	666,458	302	18,446	61.08	126,576	713,271	6,998
1896..	3,102,098	1,407	46,381	32.96	1,674,187	760	40,920	53.84	114,472	1,504,937	14,227

Year.	Metal Compositions and Manufactures.	Mineral Substances.	Oil, Mineral.				Paints and Colors.	Platinum, Unmanufactured.				Precious Stones.
	Value.	Value.	Gallons	Liters.	Value.	Value per Liter.	Value.	Lbs.	Kgms	Value.	Value per Kgm.	Value.
1892	$105,904	$16	310	1,173	$247	.21	$3,402	1	½	$83	$41.50	$5,956
1893	91,624	2,572	4,776	18,077	1,045	.06	10,271	1	½	35	17.50	311,660
1894	270,557	7,196	40,350	152,725	12,291	.08	9,180	.....	.....	.....	....	7,071
1895	66,389	4,655	23	87	43	.49	24,147	.....	... ..	.....	.....	22,967
1896	43,757	....	.....	.....	... . .	.....	23,703	.....	.....	.....	.....	18,979

Year.	Salt.				Silver-bearing Ores.	Sulphur or Brimstone (Crude).				Tin in Bars, Blocks, Pigs, or Grain, or Granulated.				Zinc or Spelter, M'f'res.
	Lbs.	Met. Tons.	Value.	Value per M. Ton.	Value.	Long Tons.	Met. Tons.	Value.	Value per M. Ton.	Lbs.	Met. Tons.	Value.	Value per Met. Ton.	Value.
1892	3,542,113	1,607	$4,657	$2.89	$305,640	.....	.....	$3	.....	55,532	25	$11,212	$448.48	$44
1893	1,155,217	524	2,897	5.53	69,027	65	66	1,360	$20.60	310,266	141	63,580	450.92	570
1894	4,693,678	2,129	5,618	2.63	62,584	.....	.....	... ...	... ...	168,484	76	26,537	349.17	216
1895	3,116,595	1,414	6,500	4.59	377,933	220	224	3,685	16.45	395,456	134	63,584	474.50	2,875
1896	7,088,134	3,192	14,947	4.68	*nil*	484	492	8,535	17.34	289,266	131	41,874	319.64	215

(*a*) From *Summary Statements of the Imports and Exports of the United States.*
(*b*) Includes lime.
(*c*) Ore, so called, is mostly matte, and during 1892 matte exports averaged about 55% fine copper.
(*d*) Sheets are not included these years, but are reported with manufactures.
(*e*) Total exports of coin and bullion, that is, includes both domestic and foreign not elsewhere specified.
(*f*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte which is classified as copper ore, and no record is kept of its silver contents. In the above table the value of silver in copper matte so far as could be ascertained from the Director of the Mint has been added to the value of silver ores, the values being calculated at the commercial rate each year. The gold in copper matte exported is not included in the exports of gold given in the above table.
(*g*) Including nickel oxide and matte.

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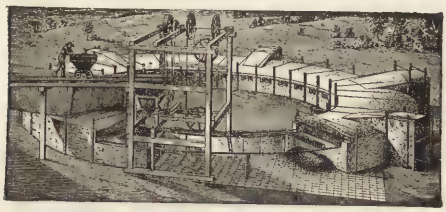
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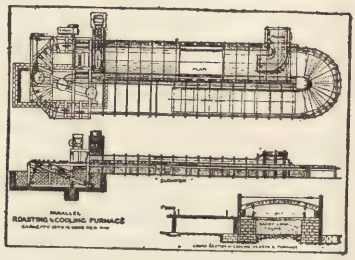
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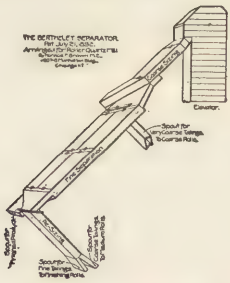


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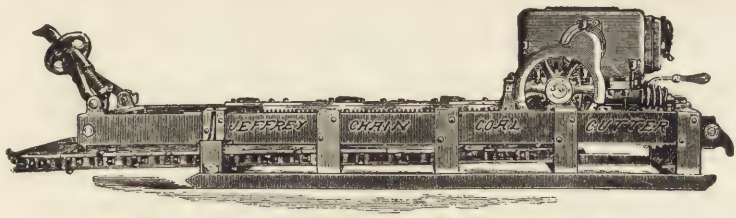
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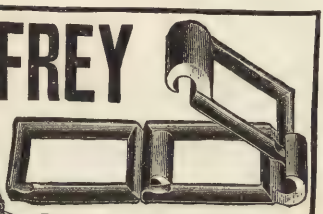
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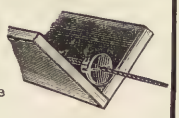
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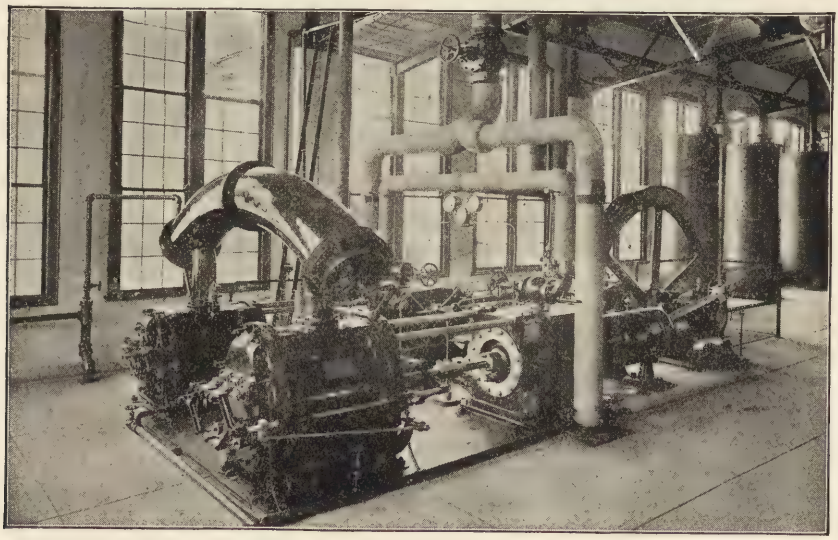
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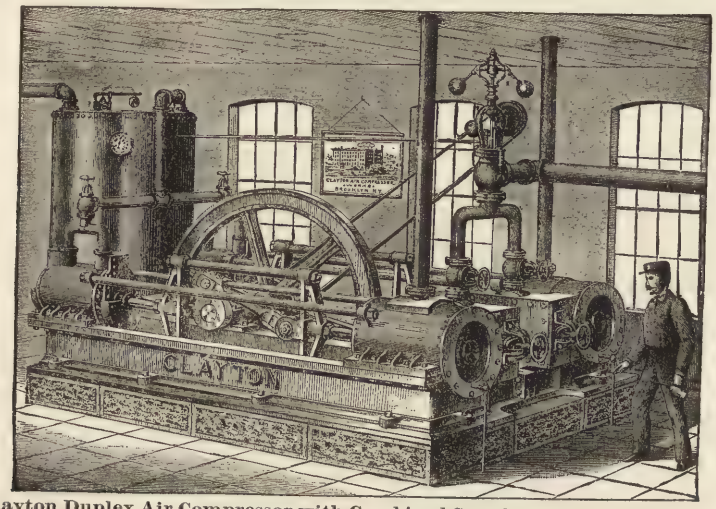
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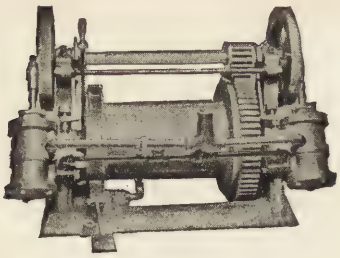
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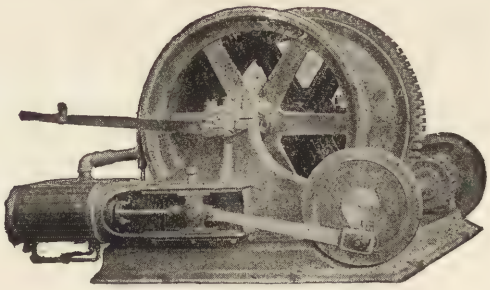
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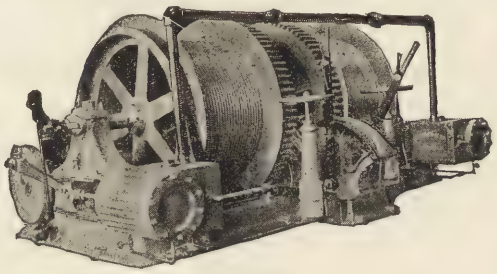
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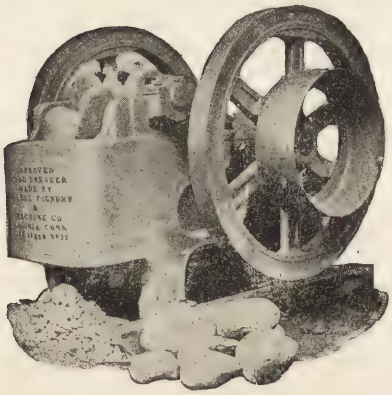
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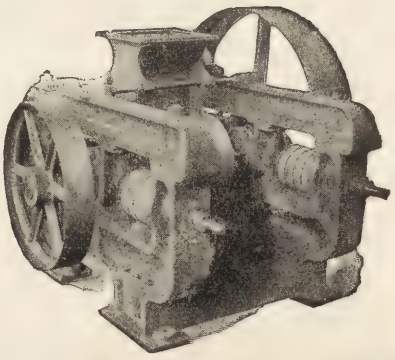
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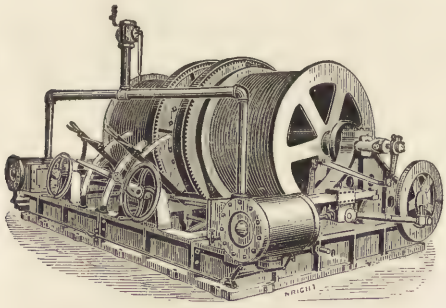
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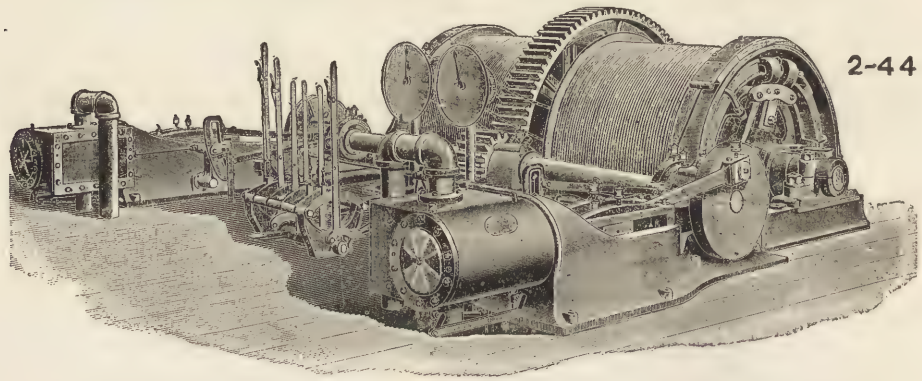
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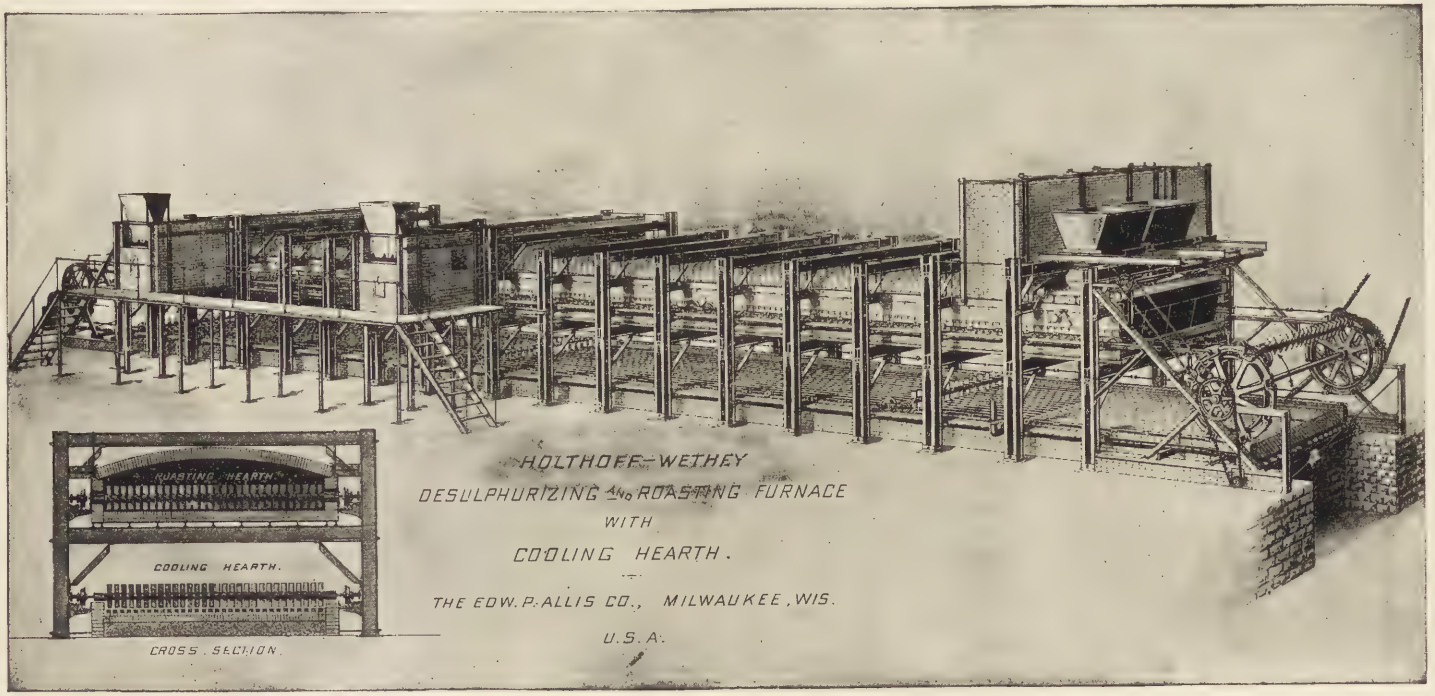
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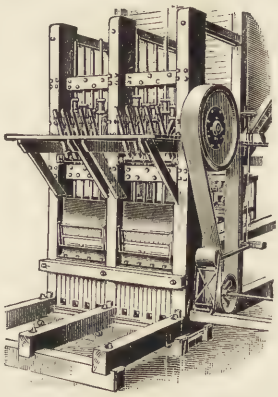
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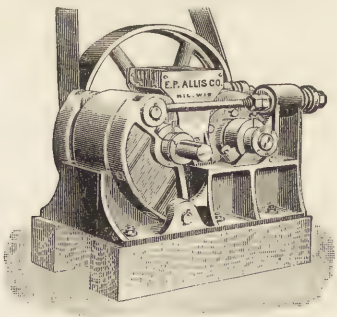
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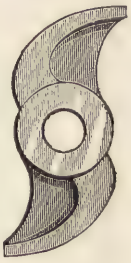
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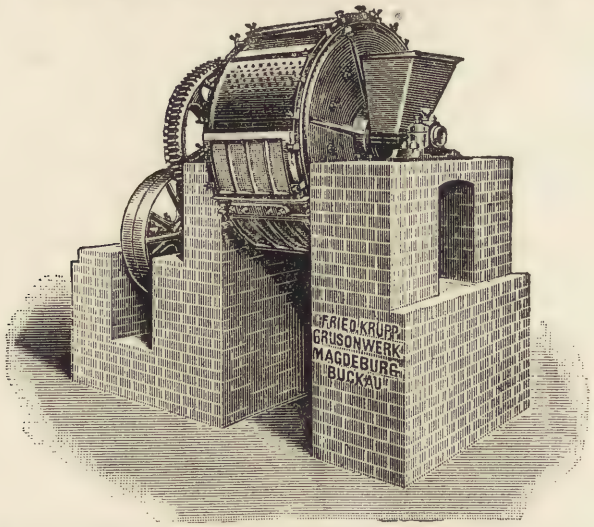
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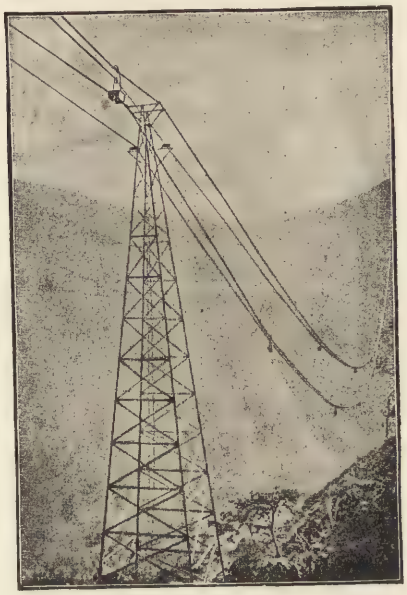
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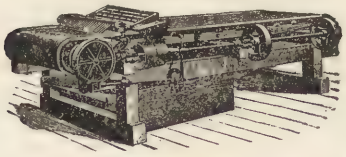
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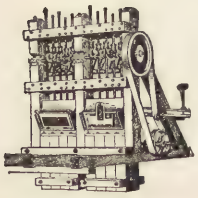
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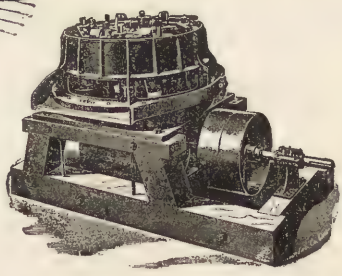
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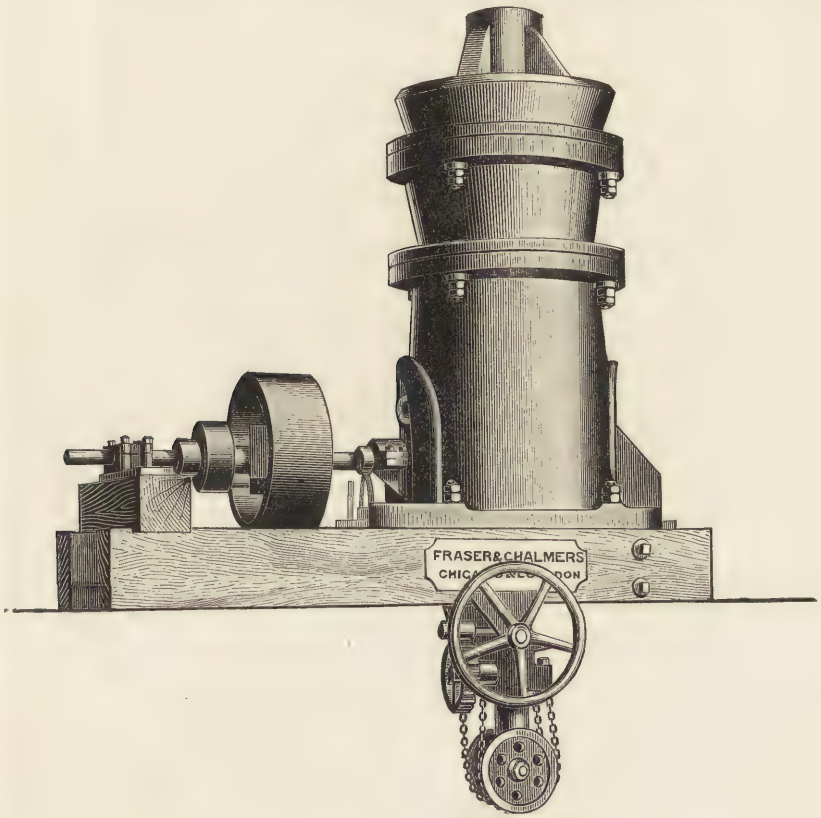
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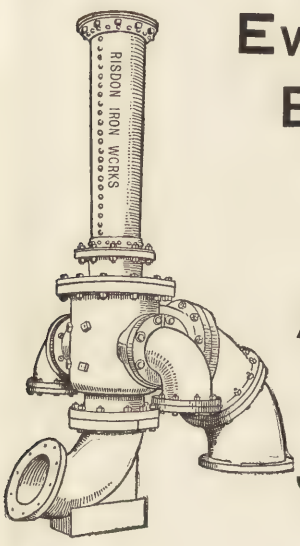
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